MODERN STEEL MAKING HANDBOOK

R. H. Tupkary, PhD
V. R. Tupkary

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Sarhan Musa, PhD
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info@merclearning.com
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CHAPTER 1

INTRODUCTION

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1.1. Steels and Their Classifications

Steel is not any specific product. It is essentially a malleable alloy of iron and one or more other elements such as carbon, chromium, nickel, silicon, vanadium, tungsten, or almost any active element (other than gas) from the periodic table of elements. There are however some isolated exceptions. There are as many as a few thousand odd varieties of steels in use. These specifically differ in their chemical compositions. However, a couple of hundred varieties are predominantly in use. The chemical composition of steels broadly divides them into two major groups, viz. (*i*) plain carbon steels and (*ii*) alloy steels. The plain carbon steels are essentially alloys of iron and carbon only; whereas, if one or more of elements other than carbon are added to steel in significant amounts to ensure specific properties such as better mechanical strength, ductility, electrical and magnetic properties, corrosion resistance, and so on, it is known as an alloy steel. These specifically added elements are known as alloying additions in steels.

Steels may contain many other elements such as Al, Si, Mn, S, P, O, etc., which are not added for any specific purpose but are inevitably present because of their association in the process of iron and steel making and cannot be totally eliminated during the known process of iron and steel making. These are known as impurities in steel. Every attempt is made to minimize them during the process of steel making, but such efforts are costly and special techniques are required for decreasing their contents below a certain level in the case of each element. For cheaper variety of steels, therefore, their contents at high levels are tolerated. These high levels are such that the properties of steels are not significantly adversely affected. These tolerable limits of impurities are considered as safe limits and the impurity levels are maintained below these safe limits. For example, for ordinary steels, sulfur contents up to 0.05% are tolerable, whereas for several special steels the limit goes on decreasing to as low as 0.005% or even lower. For most high quality steels now the total impurity level acceptable is below 100 ppm and the aim is 45 ppm.

1.1.1 Plain Carbon Steels

Plain carbon steels are broadly subdivided into four major types based on their carbon contents. These are not strict divisions based on carbon contents but are generally broad divisions as a basis of classification. This division is definitely useful. These are:

(*i*)	Soft or low carbon steels	up to 0.15%
(*ii*)	Mild steels	in the range of 0.15%-0.35% (generally ~ 0.2%)
(*iii*)	Medium carbon steels	in the range of 0.35%-0.65%
(*iv*)	High carbon steels	in the range of 0.65%-1.75%

1.1.2 Alloy Steels

The alloy steels are broadly subdivided into three groups on the basis of the total alloying elements present. This division is also only a broad division and not a rigid one. These are:

(*i*)	Low alloy steels	up to 5% total alloying contents (generally only a few %)

(*ii*)	Medium alloy steels	5-10% total alloying
(*iii*)	High alloy steels	above 10% total alloying

The above classifications of steels are useful from the point of view of steel making.

Steels were once classified in terms of their process of manufacture, e.g., Bessemer, acid open hearth, basic open hearth, electric, and so on. It was all right then because any steels could not be made by any process in those days. In modern context, however, such a classification has become totally irrelevant because all steel making processes are versatile enough to make many types of steels.

In terms of use, steels are often classified as structural, deep-drawing, rail, forging, flats, and the like.

Steels are very versatile in terms of their engineering properties. With appropriate thermomechanical treatments, properties of steels can be altered significantly to obtain a variety of combinations of mechanical, electrical and chemical properties. Very broadly in steels, hardness and strength increase with carbon contents as well as alloying contents.

In terms of ease of production and its cost, mild steels can be most economically produced and therefore are relatively cheap. It is difficult to produce low carbon steels; therefore, the cost of production in general goes up with falling carbon below 0.20-0.25% C. Similarly it goes up with increasing carbon contents above 0.30% C. Since alloying additions are to be made to produce alloy steels, these are generally costlier than plain carbon steels. The cost goes up with increasingly costlier alloy additions or increasing additions of the same alloy.

For the vast number of general applications, the use of plain carbon steels is more than adequate. Plain carbon steels therefore constitute almost 80% of the total steel production, and within that percentage, the mild steels, being cheap, take the lion's share. Mild steels are widely used for construction purposes. Flat products are generally produced from low carbon steels because of their much better malleability, which enables their easy rolling into thin sections. High carbon steels are used for making cutting tools because of their associated better hardness. Properties of alloy steels vary with the type of element added and its content. These also vary significantly with combinations of elements added. Far wider thermomechanical treatments are possible in the case of alloy steels, which lead to versatile combinations of properties.

1.2. Routes of Steel Making

Steel is essentially a ferrous material containing predominantly iron. Iron occurs in the form of iron-bearing mineral, which is called iron ore if iron is extracted from it. Iron ores are available in great abundance. Steel cannot be produced on a commercial level directly from an iron-bearing mineral that occurs in nature. On a laboratory level such attempts have been somewhat successful, though not conclusively established for their commercial applications. To produce steel directly, the ore must be quite pure and such ores of iron are not readily available.

Steel is produced from iron ores in a minimum of two stages, viz. production of iron by reduction smelting in the first stage, and its subsequent refining to steel in the second stage. The product obtained by reduction smelting of iron ore is called "pig iron" (also known in the parlance of steel making as molten iron, hot metal, etc.) if molten, and "sponge iron" or directly reduced iron (DRI) if solid. In the liquid state pig iron may contain inevitably several impurities like C, Si, Mn, P, S, etc., which together may add up to 8 wt% of the iron. In the solid state the sponge iron may contain the gangue oxides of the iron ore like Al_2O_3, SiO_2, CaO, MgO, etc., as inevitable impurities. The actual contents shall depend on the quality of iron ore used to produce sponge iron.

Steels can be produced broadly by four different routes as shown in Figure 1.1. The route as shown in (*a*) is the old route wherein molten iron is purified to produce first wrought iron.

Route (*a*) is no longer used and is only for historical knowledge. Route (*b*) is used for large-scale integrated steel plants of several million metric tons per annum capacity. Route (*c*) is the alternative developed where coking coal is not available and alternative fuel is available. Route (*d*) is the process of conversion of scrap into finished steel, which is then carburized in solid state to produce steel. With the advent of most modern steel making technology this route is now obsolete and is of only historical importance. Now much of the steel is produced by route (*b*) wherein iron ore is reduced by carbon in a vertical shaft furnace to produce molten iron, generally using coke as fuel but electricity may also be used. The molten iron thus produced is refined using iron ore or oxygen as the oxidizing agent to finally produce steel in molten condition. This is the most widely used route for producing steel today. The alternative to this is the route shown in (*c*) wherein first sponge iron is produced and it is invariably melted in

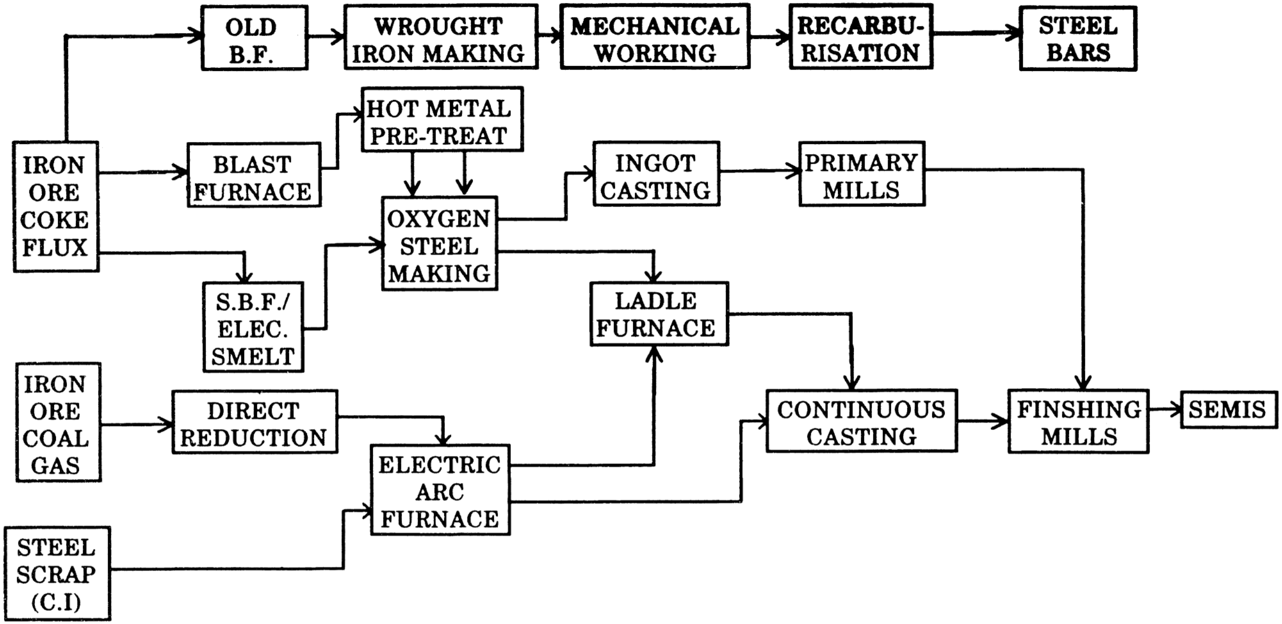

FIGURE 1.1. Schematic of different routes of steel making.

an electric arc or induction furnace to produce steel. This route is of recent origin. Steel can also be made by remelting scrap, as a secondary source, as shown in route (*d*).

Large-scale steel production in the order of millions of metric tons per annum is possible only through route (*b*). The bigger size modern steel plants have production capacities in the range of 3-15 million metric tons of steel ingots per annum. In the 1950s, the capacities were in the range of 1-5 million metric tons per annum. Such large sized steel plants must have good quality coking coal as one of the raw materials, and which is not available as abundantly as is the iron ore. Route (*c*) has therefore been developed to produce steel, but the plant size has to be relatively small in this case. A one million metric ton per annum steel plant of the type (*c*) is considered a large plant using this technological route.

The steel plants adopting the route (*d*) in India erroneously came to be known as mini-steel plants in the early 1970s. The term still continues to be in vogue. In fact small steel plants adopting any of the above routes are normally known as mini-steel plants the world over.

1.3. Principles of Steel Making

In essence, steel making is a process of selective oxidation of impurities i.e., the reverse of ironmaking. In principle it resembles fire refining of non-ferrous metals but the end product is an alloy rather than pure metal. The similarity of the two ends here. The steel making process as a whole, from molten iron, sponge iron, or even scrap, is quite complicated. The overall process includes charge preparation, melting, refining, tapping, deoxidation, decarburization, alloying, teeming, casting, stripping, etc.; all these could not have been covered by any other more appropriate term than "steel making." The term does have historical connotations as well.

During steel making the impurities are oxidized to their respective oxides (with the exception of sulfur which is reduced) and the oxides are eliminated either as gas (in the case of carbon) or liquid oxide product of suitable chemical character known as slag. The slag is in fact a sink in which the impurities to be removed from iron are to be held stably. In order to produce steel of the right quality, i.e., right chemical composition, slag of the right chemical character is aimed at and it is assessed in terms of oxidizing power and basicity of the slag. Basicity is the ratio of basic to acid oxides in the slag. Steel can be produced efficiently only if refining is adequate, i.e., right quality slag is made and clean slag and metal separation is finally brought about.

Modern steel making in the early years used iron oxide in the form of iron ore and mill scale or air as the oxidizing agents for refining. These are now almost completely replaced by oxygen gas, nearly pure in form. Modern steel making is therefore often referred to as oxygen steel making.

1.4. Classification of Steel Making Processes

Modern steel making was begun with the advent of the Bessemer process of steel making in the 1860s in the UK. The process used air for refining. Soon the open hearth and the electric furnaces were developed to melt steel scrap and make steel. These required external energies for melting scrap. The open hearth used chemical fuel like producer gas, and the electric furnace used electric energy for melting scrap. Both of these used iron ore or mill scale as a refining agent. The Bessemer process came to be classed as pneumatic, whereas the other two, because of their shallow furnace shape, were classed as hearth processes.

Depending upon the impurities to be eliminated, the slag nature has to be adjusted, even external additions if required. When only silicon, manganese and carbon are to be removed as impurities from iron, the slag produced is predominantly acidic, nearly silica saturated, and the furnace lining has to be acidic, of silica, to contain the slag. Such a process came to be known as the *acid process* of steel making. Whereas when phosphorus and perhaps sulfur, along with silicon, manganese, and carbon have to be removed, the slag should be adjusted by the external addition of flux like lime-limestone. The slag is basic in nature and basic lining of the furnace becomes necessary. Such a process came to be known as *the basic process* of steel making. In the early years both were in use but now most steel the world over is produced from basic processes of steel making. Acid steel making is exceptional, occasionally only used, in foundries and other special applications. Modern steel making is carried out in basic lined furnaces or vessels making basic slag, using oxygen as the refining media. This is often called *basic oxygen furnace* steel making or BOF.

The Bessemer, open hearth, and the electric furnace processes using iron oxide and air for refining, and because of their advent some 150 years ago at the beginning of modern steel making, are called *conventional* *processes* of steel making. The classical oxygen steel making processes in their earlier forms (of the 1950s) are also now classified as conventional oxygen steel making processes. Those using only oxygen for refining, and being of recent origin, are called modern processes of steel making. As the conventional processes had to be modernized using oxygen to withstand competition from the oxygen processes, the conventional processes using oxygen came to be known as modified processes of steel making. These were popular for a few decades after World War II . In the last few decades, the electric furnace process of steel making developed into such a technology that it could now compete with other oxygen steel making processes for making almost any variety of steel. Therefore only two basic processes of steel making are now used. The oxygen steel making in the form of LD process, along with all its modifications, is more dominantly used for steel making and these produce the large bulk of steel today. The second is electric furnace steel making, both arc and induction varieties, with all their modifications, used for steel making under specific conditions, where DRI was earlier produced to make steel. The second one is catching up with the first very rapidly. The modified conventional and the conventional oxygen processes are in use wherever they were installed and will continue to be used until they come to the stage of being scrapped for want of acceptable economics of steel production in such specific places. Any new stell plant will adopt only the modern form of BOF process or electric furnace process alone. Now only oxygen steel making is universally adopted for steel production. The modified processes are slowly giving way to these modern ones to maintain efficiency and economy of steel production.

1.5. Steel Making Processes

Prior to the 1850s molten iron was refined to first produce wrought iron, a nearly impurity-free ferrous material containing a small amount of mechanically entrapped slag particles, which were later recarburized in solid state to make steel. This was known as blister or cement steel, and the process as the cementation process of steel making. The excellent quality swords, daggers, clock springs, etc., belonging to this era of history bear testimony to the high skill achieved in making steel by this process in various parts of the world including India.

With sufficient recarburization, the melting point of steel decreased to the extent that it could be melted in furnaces, the temperatures of which were conditioned to a maximum of about 1450 °C because of combustion of chemical fuel with cold air. It was Huntsman, in 1740 in the UK, who successfully melted the cement steel in a crucible and thereby laid the foundation of the crucible process of steel making. It dominated the scene until the Bessemer process became a better alternative on a commercial level. Although it continued even after the advent of Bessemer, it had to be abandoned in favor of the electric furnace.

By 1860, Henry Bessemer perfected his process of steel making in the UK by blowing cold air through molten iron, thereby producing steel finally in molten condition. Concurrently Robert Mushet developed appropriate deoxidation of molten steel by spiegelissen (low Mn ferro-manganese) to produce reasonably sound steel ingots. This is considered to be the beginning of modern steel making in which molten iron is processed, to produce molten steel, which then can produce sound steel ingots for subsequent rolling into desired products.

Thomas modified the Bessemer process and made it applicable to irons containing phosphorus and sulfur, thereby developed the basic Bessemer or more popularly known as the Thomas process of steel making. This was around 1878 in Germany.

William and Fredrick in Germany and Emile and Pierre Martin in France developed a furnace using the heat regeneration principle to raise furnace temperature to 1600°C and thereby to melt steel scrap, refine the melt and produce steel in molten condition in 1861. This was known as the Siemens-Martin process or now more commonly as the open-hearth process of steel making because of the shallow shape of the refining vessel. The basic open-hearth process of steel making dominated the steel production for over 100 years, until World War II and even a few decades thereafter. It was however slowly replaced by the faster, economical and better-quality producing oxygen steel making processes.

By the turn of the previous century, Paul Heroult successfully put the electric arc furnace into practice for making steel by melting steel scrap. Ferranti concurrently developed the induction furnace for melting steel scrap. Both are still considered to be the best alternatives for conversion of steel scrap to steel.

The possibility of producing cheap oxygen by the Linde-Frankle process on a tonnage scale finally led to the development of LD, Kaldo and the Rotor processes of steel making using pure oxygen as a refining agent on a commercial level by the 1950s. This is the beginning of modern oxygen steel making. The top blowing process is known as LD process in Europe, as basic oxygen steel making (BOS) in the UK, as basic oxygen furnace (BOF) process in the United States, and in Far East countries. It is known as the basic oxygen process (BOP) in U.S. Steel. All these names have their own historical connotations.

The availability of oxygen and competition from newer oxygen steel making compelled the earlier conventional steel making in Bessemer, open hearth and electric furnaces to make modifications by using oxygen as far as possible in refining and also in melting. This led to the development of a series of modified conventional processes like modified Bessemer, modified open hearth, and so on. The modifications have been so drastic enough in some cases that the modified processes came to be known by different names such as Ajax, SIP, Tandem, Twin-hearth, etc. Pure oxygen could not be blown from the bottom because of the engineering limitations in the days of Bessemer, though he had the idea of doing so. The materials engineering developed to the extent that it has now become possible to blow oxygen from the bottom, as in the original Bessemer. This is known as Q-BOP or OBM process of steel making. Similarly LD-AC was developed to blow lime powder along with pure oxygen to tackle the Thomas grade iron and produce good quality steels. The latest in the field is some form of combination of bottom gas blowing and top-blowing of oxygen in the form of a modified LD process now known as *bath agitated processes,* BAP, or combination blowing processes of steel making. This is now dominating the steel making industry. All these processes are shown in Figure 1.2 in the form of schematic sketches indicating the principles involved therein.

To meet more and more stringent demand on steel quality, a series of secondary steel making techniques have lately been developed and put into practice. These fall into the category of ladle metallurgy *or* decarburization techniques, or remelting techniques, tundish metallurgy, and so on; these may be carried out with or without vacuum.

Continuous steel making using oxygen was also successfully established on pilot plant levels in the 1970s but proved to be unacceptable on a commercial level. Similarly, the *spray* steel making process was developed and put into commercial practice in the UK at a couple of places in the 1960s, but it has been discontinued.

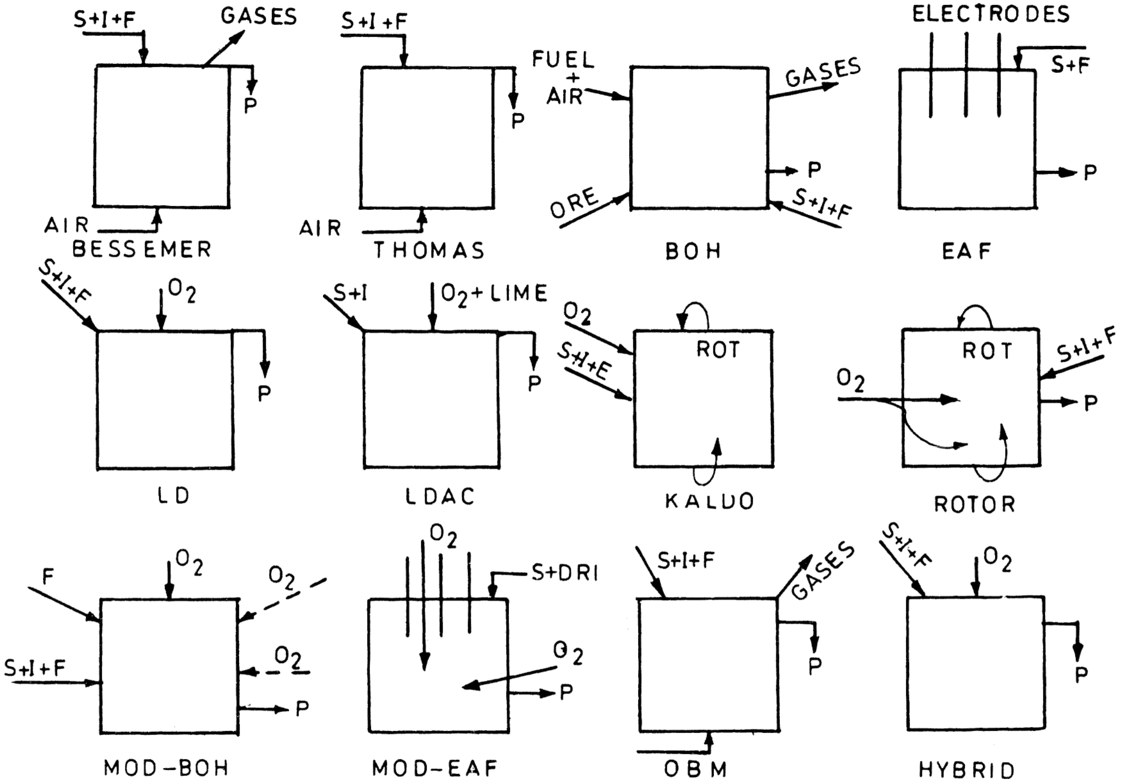

FIGURE 1.2. Illustration of underlying principles of major steel making processes. Shows the direction of charge materials, oxygen and products. The abbreviations S, I, and F mean scrap, molten iron and flux, P means products and O_2 means oxygen. Vessel rotation is indicated by arrows.

1.6. Steel Plant Products

The products of steel plants are in the form of structural shapes such as I-sections, channels, angles, plates, rails, sheets, axles, and wheels for railways, merchant products like rounds, hexagons, squares, strips, galvanized sheets, tin plates, wire rods, and so on. Rails, wheels, and axles are specifically meant for railways, which are a large consumer of steel. These are used directly as supplied by the steel plant. However the remaining products of a steel plant, by and large, are processed further in engineering industries before their actual use. These products are therefore often called semis, meaning that they are semi-finished steel products only.

For producing these semis, the liquid steel, in finished form, is cast into ingots and subsequently rolled in several types of mills, which are normally a part of the steel plant. A steel plant using iron ore, metallurgical coal (or directly coke) and flux, producing iron, refining it into steel and finally producing semis as the steel products is known as an integrated steel plant. The design is essentially based on the routes shown in Figure 1.1 (*b*) and (*c*).

The steel plant using route (*d*) in Figure 1.1 is not referred to as an integrated steel plant, but a mini-steel-plant. The mini-steel plants are more commonly used for producing alloy steels using preferably alloy steel scrap as raw material. Many times such plants produce only steel ingots and sell them; they often use continuous casting machines and produce billets, which may be rolled elsewhere into merchant products.

A few decades ago refined steel was cast in molds to produce ingots, which were later rolled to various shapes. Then came the continuous casting of steel. It has replaced the earlier inefficient mold-ingot-casting-route nearly completely. Now in all major large-scale steel plants, refined steel is continuously cast and they have almost entirely dispensed with the traditional mold casting of steel. It is in use only at some isolated plants purely because of some investment-related problems and not for its technological superiority. The blooms, slabs, or rounds are all cast continuously and cut to small pieces from which the required flat or long products are rolled through rolling mills. Small ingots are still cast in mini-steel plants where use of continuous casting, due to its small volume, is found to be uneconomical.

Now large steel plants (5-15 Mt annual capacity) can economically produce only a narrow range of products. Therefore they produce only broadly two types of products, i.e., flat products such as sheet, strip, plate, etc., or long products such as angles, channels, I-sections, bars, rails, etc. They can remain competitive only with such production strategies. Some bigger plants do produce both of these types of products in parallel routes.

1.7 SAE Steel Grades System (USA)

In the 1930s and 1940s, the American Iron and Steel Institute (AISI) and SAE were both involved in efforts to standardize such a numbering system for steels. These efforts were similar and overlapped significantly. For several decades the systems were united into a joint system designated the AISI/SAE steel grades. In 1995, the AISI turned over future maintenance of the system to SAE because the AISI never wrote any of the specifications.[1]

Today steel quotes and certifications commonly make reference to both SAE and AISI, not always with precise differentiation. For example, in the alloy/grade field, a cert might say “4140”, “AISI 4140”, or “SAE 4140”, and in most light-industrial applications any of the above is accepted as adequate, and considered equivalent, for the job at hand, as long as the specific specification called out by the designer (for example, “4140 bar per ASTM-A108” or “4140 bar per AMS 6349”) is certified to on the certificate. The alloy number is simply a general classifier, whereas it is the specification itself that narrows down the steel to a very specific standard.

The SAE steel grade system’s correspondence to other alloy numbering systems, such as the ASTM-SAE unified numbering system (UNS), can be seen in cross-referencing tables (including the ones given below).

The AISI system used a letter prefix to denote the steel making process. The prefix “C” denoted open-hearth furnace, electric arc furnace or basic oxygen furnace, while “E” denotes electric arc furnace steel.[2/3] A letter “L” within the grade name indicates lead as an added ingredient; for example, 12L14 is a common grade that is 1214 with lead added for machinability.

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[1] *Bringas, John E. (2004). Handbook of Comparative World Steel Standards: Third Edition (PDF) (3rd ed.). ASTM International. p. 14. ISBN 0-8031-3362-6. Archived from the original (PDF) on January 27, 2007.*

[2] Jeffus, p. 635.

[3] Degarmo, p. 115.

1.8. International Organization for Standardization (ISO)

ISO, the International Organization for Standardization, is an independent, non-governmental organization, the members of which are the standards organizations of the 163[4] member countries. It is the world's largest developer of voluntary international standards and facilitates world trade by providing common standards between nations. Over twenty thousand standards have been set covering everything from manufactured products and technology to food safety, agriculture, and healthcare.[5]

Use of the standards aids in the creation of products and services that are safe, reliable, and of good quality. The standards help businesses increase productivity while minimizing errors and waste. By enabling products from different markets to be directly compared, they facilitate companies in entering new markets and assist in the development of global trade on a fair basis. The standards also serve to safeguard consumers and the end-users of products and services, ensuring that certified products conform to the minimum standards set internationally.[6]

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[4] *"ISO members". International Organization for Standardization. Retrieved 2017-03-12.*

[5] *"How to use the ISO Catalogue". ISO.org. Archived from the original on 4 October 2007.*

[6] *"About ISO". ISO. Archived from the original on 4 October 2007.*

CHAPTER 2

COMMERCIAL BULK STEEL MAKING PROCESSES

In This Chapter

- Process Scenario
- Conventional Processes
- Basic Oxygen Furnace Processes
- Modified Conventional Processes
- Activities in a Steel Making Shop

2.1. Process Scenario

The modern steel making industry is about 150 years old. It began with the introduction of the Bessemer process of steel making in the 1850s. In modern terminology it was an acid process of steel making and was used as a dominant process of steel making for quite some time in the UK. After its introduction, it gradually lost its wide applicability because of its stringent requirements pertaining to the chemistry of molten iron that it could refine to steel. It, however, continued until a few decades ago for pre-refining molten iron for the basic open-hearth process, where this partially refined molten iron was later refined fully to produce steel. The Thomas process, the basic version of Bessemer, produced steel on a large scale until after World War II. The introduction of oxygen steel making finally replaced it completely a few decades ago, in spite of modifications made in the original design to keep it going.

The acid open-hearth was in operation until after the war as a dominant process of quality steel making; it also was completely eliminated soon after. The electric furnace, both induction and the arc, was responsible for its replacement. The basic open-hearth was in operation until very recently, either in the original or the modified form, as a dominant process of steel making. It has however been completely replaced by the most modern and very efficient oxygen steel making processes, except for a few isolated exceptions. The electric furnace processes of steel making still continue to dominate steel making, particularly for conversion of solid scrap or sponge iron in any form into steel. It can take any proportion of steel scrap or DRI as a charge to make steel. It can also now take liquid pig iron or hot metal as charge in any proportion to compete with any BOF process of steel making. Its use for producing medium and high alloy steels easily is still unparalleled by any other steel making process.

Steel making has been dominated by the LD process of steel making or in the form of any of its modifications, since the 1950s. The modified versions of LD in the form of combined blowing processes or the hybrid processes of steel making, in any form, are the dominant processes of steel making currently in use. In some specialized areas it is rivaled only by the electric furnace process. The gap between the oxygen blowing processes and the electric furnace processes, in terms of their applicability, is fast decreasing. The way the various steel making processes dominated the steel production all over the world since 1965 is shown in Figure 2.1.

In the OBM/Q-BOP type of oxygen steel making, the refining system almost approaches the equilibrium conditions of refining, i.e., the system goes as far as it can in reality. Many varieties of all these steel making processes are adopted at various places depending upon their specific advantages and areas of application. All these are steel making processes of bulk steel production and are described in a nutshell to make the reader conversant with the basic design of equipment and operation of steel making. These processes are described in the form of conventional processes, oxygen steel making processes, modified conventional processes, and current state-of-the-art processes of steel making.

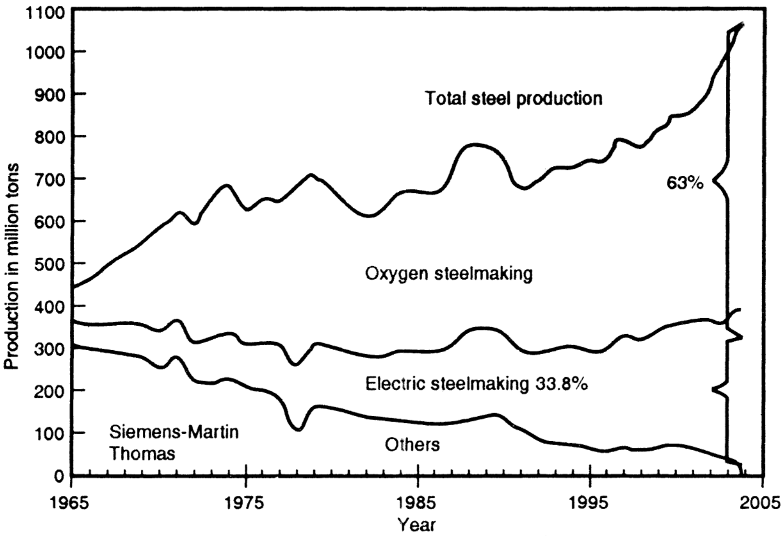

FIGURE 2.1. Share of various processes in the total production of steel on a global basis.

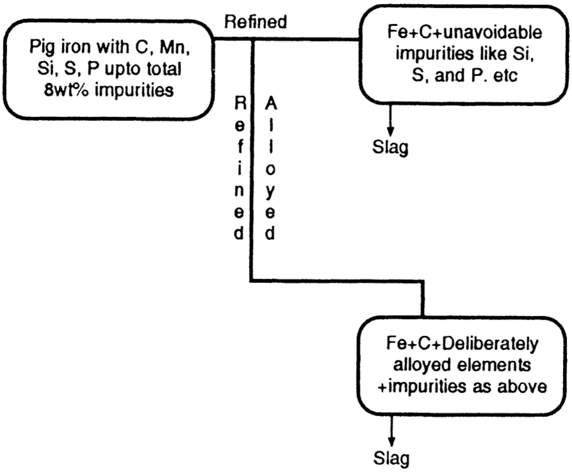

FIGURE 2.2. Depicts the principle of steel making via B.F. route.

2.1.1. Process Routes

In steel making, molten iron from an ironmaking unit, i.e., blast furnace, is refined to obtain the desired chemistry of the final product known as steel. This principle can be described as shown in Figure 2.2.

The process in principle using sponge iron as the source of iron is shown in Figure 2.3. A modern combination route wherein both hot metal and sponge iron are used for steel making is shown in Figure 2.4.

It amounts to removing

(1)	Carbon	from	4.3-4.5%	to	0.002	to 1.8%
(2)	Silicon	from	0.5-1.5%	to	up to	0.01%
(3)	Manganese	from	0.3-1.0%	to	up to	0.1%
(4)	Phosphorus	from	0.1-1.8%	to	up to	0.02%
(5)	Sulphur	from	0.05	to	up to	0.001%

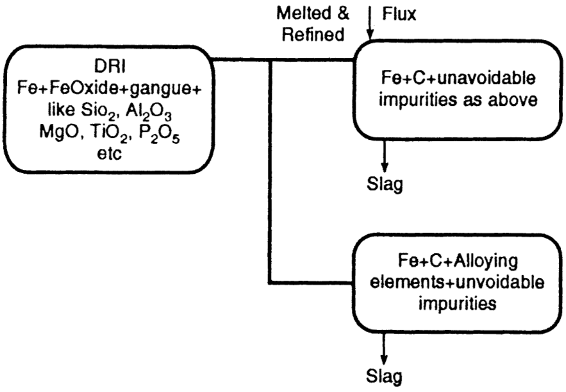

FIGURE 2.3. Alternative route of steel making.

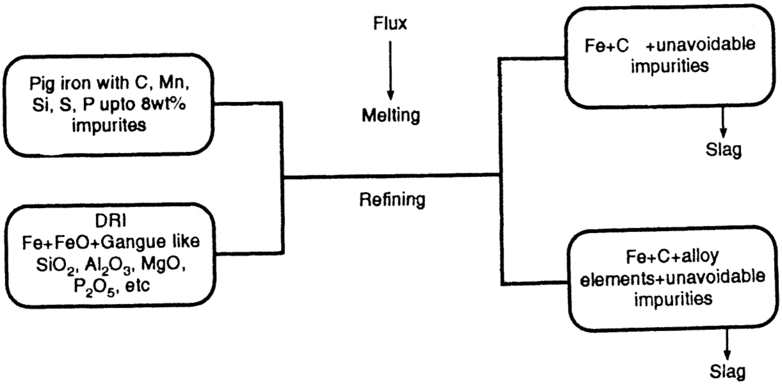

FIGURE 2.4. Modern route of steel making using HM, DRI and scrap as charge. As a combination of the traditional and the alternative routes of steel making.

The Si, Mn, P, and S are iron impurities; only carbon is the useful alloying element to make steel, and hence these are shown to be removed up to the indicated lower limits.

Molten iron from blast furnace is generally available at around 1350 to 1550°C. The melting point of iron is 1535°C, and although the melting point generally decreases because of the alloying content, steel is generally produced finally at around 1600-1700°C.

2.2. Conventional Processes

2.2.1 Basic Bessemer/Thomas

In this process molten pig iron is held in a vessel with a perforated bottom called a converter. Cold air or an oxygen-enriched blast is forced through the metal from below as shown in Figure 2.5. Refining is complete in about 15-20 minutes, and taking into account the time required for charging, tapping, repairs, etc., a tap-to-tap time of about 30-35 minutes is required. This is an autogenous process, i.e., no external heat is needed. The exothermic chemical reactions during refining provide the necessary heat in order to maintain the metal well above its melting point, in spite of the fact that cold air is blown through.

2.2.2. Open Hearth

The furnace in this process is a fairly shallow basic lined vessel as shown in Figure 2.6, hence the name. It is heated by either liquid and or gaseous

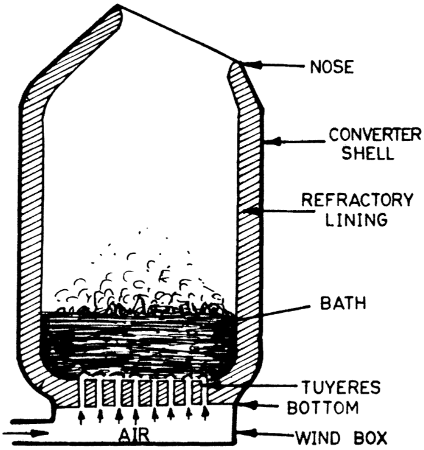

FIGURE 2.5. Basic Bessemer/Thomas converter.

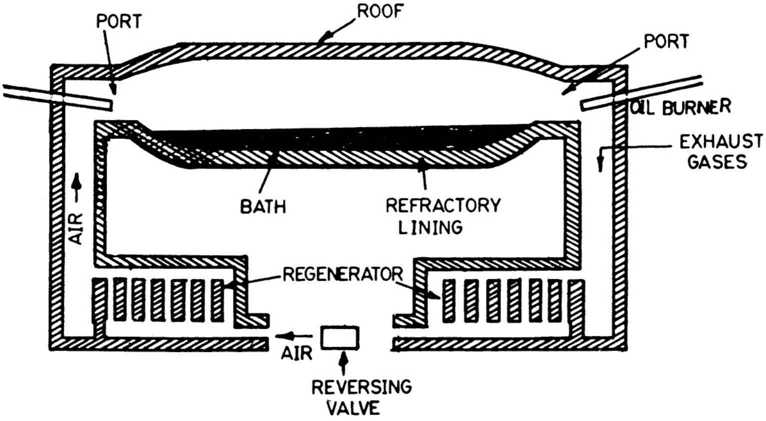

FIGURE 2.6. Open-hearth furnace.

fuels using the heat regeneration principle so as to attain steel making temperatures of about 1600-1700°C. In modern practice the charge is a mixture of scrap and molten pig iron. The scrap is initially heated to near its softening point and molten pig iron from the blast furnace is poured onto it. The charge may contain iron ore and limestone. The early practice of using iron ore for refining is now of only historical significance. Now refining is universally carried out by blowing oxygen gas through lances. This makes the process during refining an autogenous one, although heating is necessary for heating the scrap initially. The basic slag is prepared by adding lime and iron ore. The atmosphere in the furnace is therefore always oxidizing. A tap-to-tap time of about 6-10 hours is usually required although some furnaces have achieved this in as low as even four hours.

2.2.3. Electric Arc

Generally a three-electrode arc furnace, as shown in Figure 2.7, is used. The steel making temperature is maintained by an electric arc struck between the electrodes and the metallic charge. This process is very similar to open-hearth in charging and refining, and several hours are needed for one heat. This is the only process wherein either oxidizing or reducing conditions can be maintained at will during refining since the furnace does not possess its own ambient oxidizing atmosphere. The high cost of electrical energy in general makes this process costly. It is well suited for making alloy steels. It usually runs on cold charges and the furnace is of a small capacity. The modern electric arc furnaces are, however, of much bigger sizes and some of them also take hot metal as part of the charge.

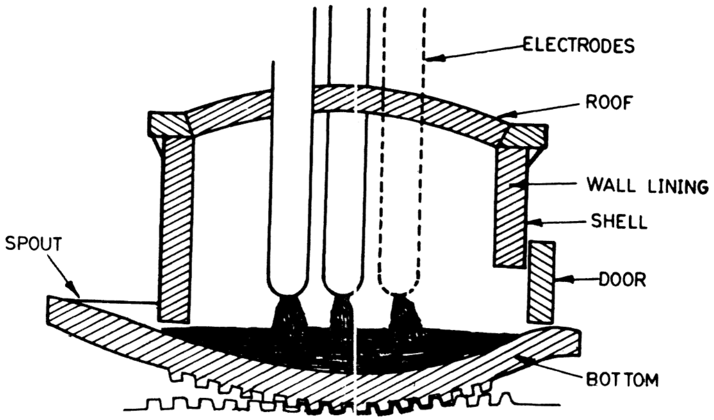

FIGURE 2.7. Electric arc furnace.

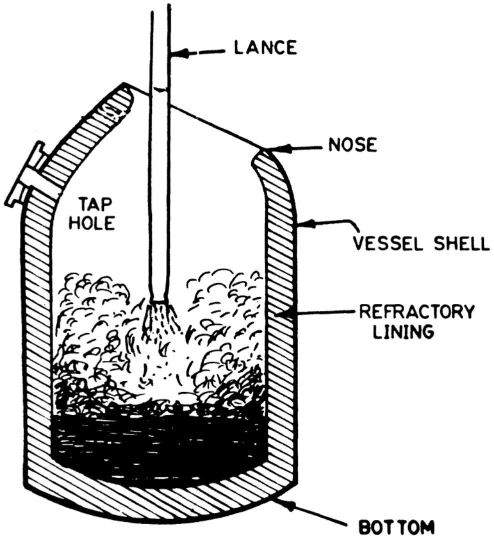

FIGURE 2.8. LD vessel.

2.3. Basic Oxygen Furnace Processes

2.3.1. LD

The furnace is very similar in design to that of the Bessemer converter, except that the bottom is solid and oxygen is blown vertically from the top as shown in Figure 2.8. All oxygen steel making processes are autogenous like Bessemer and need molten pig iron as a charge material. Steel scrap is used

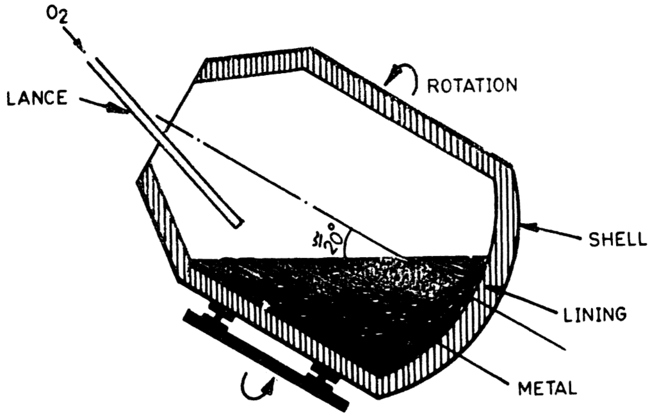

FIGURE 2.9. Kaldo vessel.

as a coolant since excess heat is available in these processes. The refining is complete in about 20-25 minutes of oxygen blowing, and a tap-to-tap time of 40-60 minutes is needed.

2.3.2. Kaldo

The furnace is quite similar to that of LD converter in design and it is kept inclined to horizontal at an angle of about 20° as shown in Figure 2.9. During refining the converter is rotated around its own axis at any desired speed up to a maximum of 30 rpm. The oxygen lance is also inclined to the horizontal and the angle is adjustable. The refining cycle is a little longer than that of LD, i.e., 60-70 minutes.

2.3.3. LDAC/OLP

This is a modified LD process developed to successfully refine pig iron containing more than 1.5% P in order to replace the old Thomas or the basic Bessemer processes. In order to hasten removal of phosphorus, lime powder is blown along with oxygen in this process. The vessel design is similar to that of LD. The lance, however, is of a different design to allow the supply of lime powder along with oxygen.

2.3.4. Q-BOP/OBM

The difficulties in top blowing of oxygen as in the LD process and the attendant problems and the fascination for the simpler design of the Bessemer process ultimately led to the development of this process in which oxygen can be blown, from the tuyeres at the bottom of the vessel, to achieve

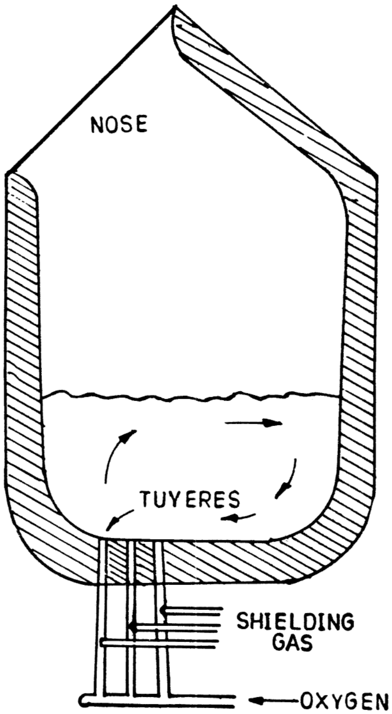

FIGURE 2.10. OBM vessel.

refining as shown in Figure 2.10. This was not possible in the days of Bessemer although he had thought of blowing oxygen through bottom tuyeres in his vessel. The developments in general engineering aspects of such designs finally could make it a commercial success. The vessel in this process is similar to the Bessemer vessel in its design. The process was originally developed to refine Thomas grade irons but it can now treat ordinary basic iron. The tap-to-tap time is nearly 40 minutes.

2.3.5. Hybrid Blowing

The difficulties and problems faced in refining using top oxygen blowing LD process and also in using bottom oxygen blowing as in OBM process have lately led to the development of a successful blowing strategy by combining the two, that is blowing from top as well as from bottom simultaneously; this came to be known as a combined process of steel making as shown in Figure 2.11. There are several such designs, commonly known as the *hybrid* processes of steel making. Now the original LD process itself,

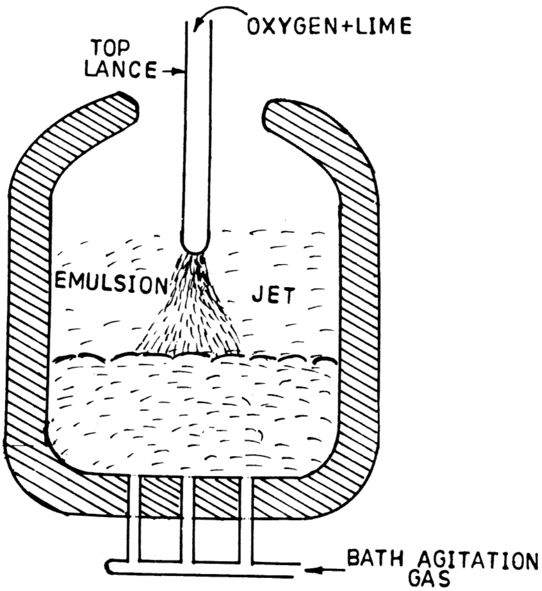

FIGURE 2.11. Hybrid blowing vessel.

in juxtaposition to the modern one, is referred to as merely a conventional oxygen process and the recent hybrid processes are currently used and classified as oxygen refining processes of steel making.

2.4. Modified Conventional Processes

Although the Bessemer process is very fast, it cannot be adopted widely because of its rigorous requirements of pig iron composition, and also because of the poor quality of the product. Soon after the Bessemer process was commercially adopted, the open-hearth process became available and was adopted widely because of its better quality product and its versatility with respect to charge material composition. The open-hearth process is very slow, but that enables the operator to effect better control on the refining reactions to obtain the desired end product of a better quality. With the introduction of the oxygen steel making processes, the existing open-hearth plants were hard put to compete with the new processes in terms of production rate coupled with the quality of the product. The conventional processes were, therefore, suitably modified to use oxygen during melting and refining to decrease the tap-to-tap time. A tap-to-tap time of about four hours has now been attained on many furnaces against the earlier 12 to 16 hours. Oxygen lancing was carried out vertically through the roof or through the end walls at an angle (Ajax process). The Bessemer blast is also enriched with oxygen to improve the quality of the product. Oxygen lancing during refining in an electric arc furnace is adopted with some distinct advantages particularly for alloy steel making.

The conventional processes were modified even to the extent of changes in the design and operational procedures to take advantage of using oxygen as a refining agent. The Tandem furnace process and the Twin Hearth are typical examples in point.

2.5. Activities in a Steel Making Shop

The steel making shop is that part of the plant where the steel making furnace(s) is located and where steel is made and cast into ingots. This was earlier known as the steel melting shop, or SMS, on the shop floor because of melting of scrap for its conversion to steel in earlier years. It is now better to call this the steel making, or SMS. In general a steel making shop is supposed to carry out the following functions:

(*i*) Preparation and storage of raw materials required for the entire shop so that these are readily available during the operation as required. This means organization of inward material flow. Orders for the materials should be placed well in advance to avoid any delays and ensure smooth operation.

(*ii*) Charging, melting, and subsequent refining of the charge to the desired chemical analysis.

(*iii*) Deoxidation of steel, either inside or outside the furnace, addition of alloying elements, if any, to obtain desired chemical composition.

(*iv*) Tapping the refined and finished steel from the furnace into a ladle; ladle additions to be made while tapping.

(*v*) Secondary refining or ladle metallurgy, if any.

(*vi*) Teeming refined steel in ingot molds to obtain sound and clean ingots, or continuous casting may be adopted instead.

(*vii*) Stripping of ingots, reconditioning of ingot molds, assembly of molds, etc.

(*viii*) Maintenance of all the equipment in the shop.

In short, all the activities in the shop are geared to produce what has been asked for, by the most economical method from the given charge materials.

CHAPTER 3

REFINING SLAGS

In This Chapter

- Structural
- Slag Properties
- Theories of Slags
- Iron and Steel Slag (USA 2017)
- Events, Trends, and Issues

3.1. Structural

The molten oxide byproduct of smelting and refining processes is known as a slag. Traditionally oxide melt containing silica as one of the constituents is called slag because it is invariably present in the slag. Strictly speaking, oxide melt containing at least two oxides of opposite chemical nature should be called a slag. The term however is used more in the conventional manner rather than according to any strict scientific definition. By convention oxides like SiO_2, P_2O_5, B_2O_3, etc., are called acid oxides, and oxides like CaO, MgO, FeO, MnO, Na_2O, K_2O, etc., are called basic oxides. Aluminum oxide, Al_2O_3, is called amphoteric oxide. This opposite chemical nature results in the formation of complex silicates, phosphates, etc., as

$$2RO + SiO_2 = 2RO \cdot SiO_2 \qquad (3.1)$$

and $$4R_2O + P_2O_5 = 4R_2O \cdot P_2O_5 \qquad (3.2)$$

(where R represents monovalent or divalent cations) in a manner analogous to the aqueous acid-base neutralization.

The basic oxides are by and large ionic solids wherein the ionic character predominates over the covalent character of the bond. On melting, these dissociate into:

$$RO = R^{2+} + O^{2-} \text{ and } R_2O = 2R^{+} + O^{2-} \qquad (3.3)$$

Although short-range order exists in the melt, no complex ions are formed and the ions behave as fairly free mobile ions. If the covalent character predominates over ionic character, in bonding a cation with oxygen anion, the tendency of complex ion formation increases on melting. The acid oxides have dominant covalent bonding and hence do not dissociate into respective simple ions as shown above.

In an orthosilicate melt the dissociation appears to be as

$$2RO + SiO_2 = 2R^{2+} + SiO_4^{4-} \qquad (3.4)$$

and which indicates as if there is no activity of oxygen ions in the melt. This composition can conveniently be referred to as a neutral silicate melt. As the basic oxide content increases, the free oxygen ions appear in the melt and it is termed a basic melt. On the other side of the neutral composition, since the structure is deficient in oxygen ions, it is termed an acid melt. The orthosilicate composition is actually not neutral. Experimental evidence indicates that there is a finite activity of oxygen. This activity can arise only if some silicate ions polymerize to form higher order complexes even at this composition. The orthosilicate melt may therefore contain ions like R^{2+}, O^{2-}, SiO_4^{4-}, $Si_2O_7^{-6}$ and so on. The presence of several acid oxides and basic oxides in a commercial slag complicates the picture still further.

The nature of ions in a basic melt is known with some certainty, whereas that in an acid melt is quite uncertain. The uncertainty increases with the increase in the acidity of the melt. The basic slags in practice may contain the following ions depending on the composition:

Simple cations:	Fe^{2+}, Ca^{2+}, Mg^{2+}, Mn^{2+}, Na^{+}, K^{+}, etc.
Simple anions:	O^{2-}, S^{2-}, F^{-}, etc.
Complex anions:	: SiO_4^{4-}, PO_4^{3-}, AlO_3^{3-}, FeO_3^{2-}, etc. and other higher order complexes.

The nature of chromium ions is uncertain and simple aluminum cation may exist. The majority of steel production comes from basic processes and it is fortunate that the structure of slag in these processes is reasonably well known. It is however, unfortunate that the acid steel making slag structure cannot be precisely stated.

3.2. Slag Properties

The main slag properties of interest to extractive metallurgists are its ability to retain oxide products (including gangue) of reactions and its oxidizing /reducing (oxidizing potential) power. The viscosity, thermal conductivity, surface tension, etc., of a slag are also important from the point of view of rate of slag-metal, slag-gas, and slag-refractory reactions.

The ability of a slag to retain oxides is generally expressed as the ratio of basic to acid oxides and is variously represented as mass concentration ratios:

$$V \text{ ratio} = \frac{\%\,\text{CaO}}{\%\,\text{SiO}_2} \tag{3.5}$$

Modified
$$V \text{ ratio} = \frac{\%\,\text{CaO}}{\sum \%\,(\text{SiO}_2 + \text{Al}_2\text{O}_3 + \text{P}_2\text{O}_5)} \tag{3.6}$$

Basicity
$$B = \frac{\sum \%\,(\text{all basic oxides})}{\sum \%\,(\text{all acid oxides})} \tag{3.7}$$

Common
$$B = \frac{\sum \%\,(\text{CaO} + \text{MgO})}{\sum \%\,(\text{SiO}_2 + \text{Al}_2\text{O}_3 + \text{P}_2\text{O}_5)} \tag{3.8}$$

Specific
$$B = \frac{\sum \%\,(\text{RO} - 4\text{P}_2\text{O}_5)}{\%\text{SiO}_2} \tag{3.9}$$

In fact in all the above representations of basicity norms, it is presumed that all the CaO, MgO, etc., added as flux, during steel making is fully dissolved in the slag to form liquid oxide melt. In reality anything up to 10wt% of each of such oxides may remain undissolved in slag even at steel making temperatures. This happens in oxygen steel refining because of its faster refining. This was not so in slower open hearths. This proportion may be higher in still faster refining operations wherein time is not available for lime assimilation in slag. If this is not corrected the basicity values calculated on the basis of overall analysis of these oxides may be unrealistic i.e., much higher than the real effective values.

3.2.1. Oxidizing Power

Thermodynamically even the acid slags have some finite free oxygen ion activity but the basic slags have a much greater value. The oxygen ion activity should not be confused with the oxidizing/reducing ability of the slag. The oxidizing/reducing power of a slag is thermodynamically equal to the partial pressure of oxygen in equilibrium with the slags. Of the common oxides generally present in slags of interest in iron and steel making, the iron oxide (FeO) has the highest equilibrium partial pressure of oxygen. This means that the p_{O_2} value in equilibrium with the iron oxide in the slag shall automatically be in equilibrium with the rest of the oxides. If $p^0{}_{O_2}$ is the equilibrium partial pressure of oxygen in contact with pure iron oxide (i.e., iron oxide held in an iron container) the equilibrium partial pressure of oxygen of a slag will be given by

$$p_{O_2}/p^0{}_{O_2} = a_{(FeO)} \tag{3.10}$$

i.e.

$$p_{O_2}\ a[a]_{(FeO)} \tag{3.11}$$

Since the value of $p^0{}_{O_2}$ is very high the oxidizing power of a slag is, therefore, conventionally expressed as the activity of iron oxide in slag. These activity data have been determined for a variety of slag compositions and have been usefully compiled in the form of iso-activity curves in the pseudo-ternary system (CaO + MgO + MnO-(SiO + P_2O_5-(FeO) as shown in Figure 3.1. Although this figure strictly holds true for slags at 1600°C it can be used to know the iron oxide activity in the normal range of steel making temperatures since the value is little affected by temperature changes. Alternative plots for the system (CaO + MgO-(SiO_2-(FeO_t) are also known. The (FeO_t) means the iron oxide content based on the total iron content of the slag and its use simplifies the analysis of the slag. The general shape of the plot is not affected by the use of (FeO_t) in place of true (FeO). The form of plots in Figure 3.1 indicates that for any given iron oxide content its activity shall be maximum for that composition which is close to the join of iron oxide apex to the orthosilicate composition. This is because of a high preferential attraction between 2RO and SiO_2. The increase in silica concentration tends to form iron silicate and increase in base concentration

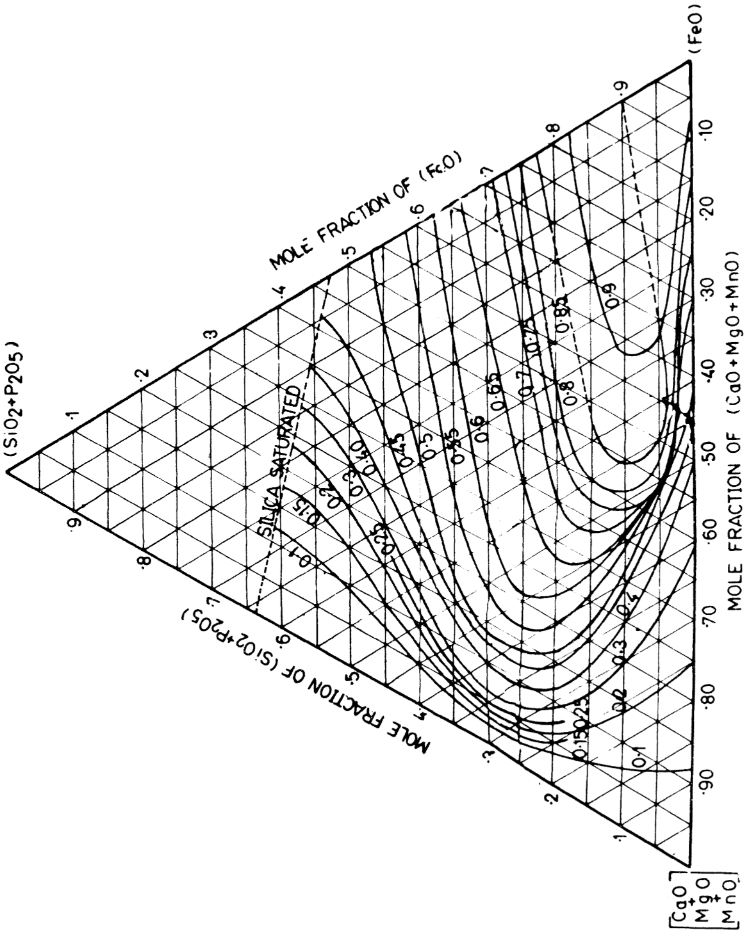

FIGURE 3.1. Activity of iron oxide as determined in laboratory slags. The iso-activity curves have been superimposed on the pseudo ternary phase diagram (after Turkdogan and Pearson[1])

tends to form respective ferrites thereby reducing the activity of iron oxide as the composition changes away from the orthosilicate composition, on either side. A slag with V-ratios equal to two has thus the maximum oxidizing power for any iron oxide content. In practice however slags with much higher V-ratios are required to effectively eliminate phosphorus and sulfur from iron during refining.

The basicity and oxidizing power of a slag are independent properties to the extent that in practice, basicity is generally related to lime concentration and the oxidizing power to the iron oxide content of the slag, although the latter is also a base. The slag, therefore, can be of any basicity and any oxidizing power provided the requirements regarding viscosity, thermal conductivity, surface tension etc., are properly met. In basic steel making, slag can be either highly basic and highly oxidizing or highly basic and quite reducing. In acid steel making the slag being silica saturated has practically no basicity, but it is highly oxidizing.

Liquid iron takes oxygen gas in solution when in contact with iron oxide. The equilibrium is represented as:

$$\underline{\mathrm{Fe}} + \underline{\mathrm{O}} = (\mathrm{FeO}) \tag{3.12}$$

associated free energy change is given by

$$G^\circ = -109000 + 45.9\ T$$

The equilibrium constant

$$\begin{aligned} K &= a_0/a(\mathrm{FeO}) \\ &= [\%\mathrm{O} \cdot f_0]/a(\mathrm{FeO}) \end{aligned} \tag{3.13}$$

The activity coefficient of oxygen dissolved in iron can be calculated as described later in Section (4.2) and which gives

$$\log f_0 = -0.1\%\ \mathrm{O} \tag{3.14}$$

and from this activity value of iron oxide in any slag can be calculated for a particular temperature, knowing the oxygen content of the metal bath.

Oxygen content of iron increases with the increase in activity of iron oxide in slag in contact.

Iron exists as various oxides and it is difficult to neither predict nor actually measure the concentrations of those oxides individually easily in any slag. Hence chemically iron is estimated and is expressed as Fe_TO, i.e., iron expressed as FeO. This is how it is always dealt with in the literature, unless otherwise specifically indicated.

During steel making if the slag acts only as a sink for oxide products and it does not take part in the refining reactions, it may be very viscous or *dry* (in acid Bessemer process) so that it can be readily separated from the steel melt. If the slag has to act as sink as well as take part in refining, it should be thin or *wet* (in all other steel making processes). A thin slag increases mass transport and thereby accelerates reactions. A thin slag in general has good thermal conductivity. In open-hearth and in Kaldo processes where heat is supplied from above, the slag acts as a thermal barrier and the rate of heat transfer to the metal bath is inversely proportional to the thickness of the slag layer. The practical slag compositions are chosen to achieve all these requirements at the optimum. The approximate iron oxide contents and the V-ratios of slags used in steel making practices are shown below:

Process	*Approx. % (FeO)*	*Type of slag*	*Approx. V-ratio*
Acid Bessemer	12-18	$FeO\text{-}MnO\text{-}SiO_2$	Nil
Basic Bessemer	12-18	$FeO\text{-}CaO\text{-}P_2O_5$	2.5-3.0
Acid open hearth	15-25	$FeO\text{-}MnO\text{-}SiO_2$	Nil
Basic open hearth	15-30	$FeO\text{-}CaO\text{-}P_2O_5$	1.5-3.0
Basic electric arc (oxidizing)	15-25	$FeO\text{-}CaO\text{-}P_2O_5$	1.5-3.0
Basic electric arc (reducing)	0.5-2.0	$CaO\text{-}CaC_2$	3.0-5.0
Basic oxygen furnace	15-40	$FeO\text{-}CaO\text{-}P_2O_5$	2.0-3.5

The slag composition varies considerably with the progress of refining in each process and the above figures and types should be used with caution.

The viscosity of any slag can be decreased by raising the temperature. The maximum steel making temperatures (average in the furnace) rarely exceed 1700°C. Highly basic slags are not thin enough even at this temperature. The viscosity of these slags is generally decreased by the addition of either silica, fluorspar, or bauxite. The use of silica alters the V-ratio and hence is not used extensively except when spar cannot be added. Even if silica is used it is used very sparingly. Spar addition simplifies the silicate network, and thereby reduces viscosity of slags.

The F^- -ions act like O^{-2} ions in modifying the complexes and thereby decrease the viscosity. It is used extensively except when P_2O_5 rich slags, of fertilizer grade, are to be made. The slag containing fluorspar cannot be used as a phosphatic fertilizer. Although Al_2O_3 itself is a very refractory oxide its addition does reduce the softening point of some slags and hence their viscosities. It is usually an acid oxide but of much less strength than silica, and its addition does not alter the V-ratio of the slag. It is, therefore, preferred if use of spar is forbidden.

The liquidus temperature is yet another important property of a slag. In general the higher the liquidus temperature, the higher will be the viscosity of that slag at the steel making temperatures. The liquidus temperatures have been determined for innumerable types and compositions of synthetic and practical slags. These data are generally plotted in the form of ternary and quaternary diagrams with iso-liquidus lines superimposed on them. The readers can refer to the standard reference books for the information. The applicability of a certain slag can be judged vis-a-vis its liquidus temperature from such diagrams readily.

3.2.2. Synthetic Slag

Nowadays synthetic slags are made and used for refining under special circumstances, particularly in secondary refining. These slags are prepared by melting the required oxides in desired proportions in a separate furnace. The earliest reference to pre-melted slag is with reference to its use in the Perrin process, which is the progenitor of secondary steel making. These are now commonly used in secondary steel making, particularly for desulphurization. Such slags have also been recommended for external dephosphorization as well. It is however not yet as commonly used as the use of synthetic slag for desulfurization. The synthetic slags for desulfurization are strictly of the binary type CaO-Al_2O_3 with liquidus temperature around 1450°C and for this alone it is preferred in steel making. However, in practice a small amount of SiO_2, which is invariably present in slag, due to many reasons, changes it to a ternary system like CaO-Al_2O_3-SiO_2 as shown by the partial ternary system in Figure 3.2. The usual composition is generally 42-48% Al_2O_3, around 52-58% CaO and balance SiO_2 and other oxides as this have the lowest liquidus temperature range in the ternary system CaO-Al_2O_3-SiO_2. The slag is however dealt with as binary system while working out the chemistry. The oxides like SiO_2, MgO, and so on, usually come from

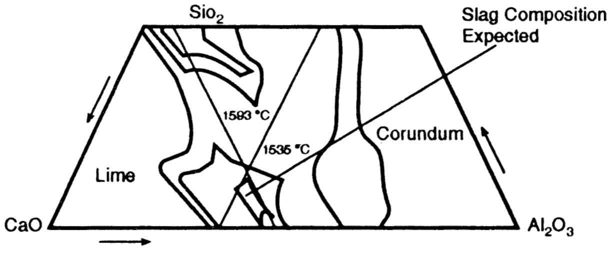

FIGURE 3.2. The relevant portion of the CaO-Al_2O_3-SiO_2 ternary system of interest for desulfurization of steel.

the carry-over slag. Such a slag should not contain any carbon, any free undissolved lime, or fluorine and should have a stable composition. In other words, these slags must be made by melting them fully and properly as is visualized in their binary equilibrium diagram, then solidified and supplied in the form of sandy powder for their use, when it will be melted again in the steel making furnace.

3.3. Theories of Slags

Besides the structure, the liquidus temperature and the thermodynamic properties of a slag, attention has also been given to understand the factors that affect the reactions of a slag with other phases. The theories of slags are nothing but a set of principles to evaluate as far as possible the true equilibrium constant of a reaction between the slag and any other phase like metal, refractory walls of a container, furnace gases, etc.

3.3.1. Molecular Theory

The molecular theory assumes the existence of neutral molecules such as various silicates, phosphates, sulfides, oxides, etc., in the molten slag. The reaction can readily be written in terms of chemical formulae of these molecules. Schenck assumed the complex molecules to be dissociated to a definite extent into the corresponding simpler molecules e.g., as

$$2(\mathrm{FeO.SiO_2}) = 2(\mathrm{FeO}) + (\mathrm{SiO_2}) \tag{3.12}$$

from which the concentration of the free oxide could be calculated.

The knowledge of the free oxide concentration in a slag is a useful parameter in the calculation of equilibrium constant of a reaction of slag with any other phase. The free oxide concentration can be calculated from the dissociation constant of such reactions as

$$D = \frac{(\%\mathrm{FeO})^2 \cdot (\%\mathrm{SiO_2})}{(\%2\mathrm{FeO} \cdot \%\mathrm{SiO_2})} \tag{3.13}$$

A series of temperature-dependent values of dissociation constants were computed by Schenck and coworkers from laboratory and work data. They also derived equilibrium constants for partitioning reactions between slag and metal. A value of zero for the dissociation constant of calcium phosphate is in agreement with its high stability and this assumption explained the phosphorus distribution satisfactorily.

Since the activity values of oxides in complex silicate slags deviate much less from ideality, the free oxide concentrations calculated by Schenck were not far off from the true activity values that are known today. This explains the widespread adoption of this approach even today, particularly by the open-hearth operators. For the conditions under which the *D* values have been estimated, the treatment should predict the equilibrium conditions accurately.

The treatment is thermodynamically quite sound and hence should not be totally discarded in spite of the fact that its basic assumptions are no longer valid in the light of the present day understanding of slag structure.

The molecular theory was quite simple in the early days since a limited number of complex compounds in slag were known. The original simplicity no longer exists since a very large number of such compounds have been found out in slag and dissociation constants for each one of them need to be known to calculate the free oxide concentrations.

3.3.2. Ionic Theory

The molecular constitution of molten slags cannot explain a number of phenomena observed in slags e.g., electrical conductivity, electrolysis, electrocapillarity, etc. From several independent observations it has been established and now universally accepted that slags are ionic liquids. A galvanic element is formed if a metal bath is in contact with molten slag. It means that the reaction between slag and any other phase is an electrochemical process and can be represented in the form of a partial electrochemical reaction such as

$$[S] + 2e = (S^{2-}) \tag{3.14}$$

$$[Fe] = [Fe^{2+}] + 2e \tag{3.15}$$

$$[O] + 2e = (O^{2-}) \tag{3.16}$$

The overall reaction can be represented as

$$[S] + [Fe] = (S^{2-}) + (Fe^{2+}) \tag{3.17}$$

so that the electrical neutrality of the phase is maintained. The equilibrium constant of the above reaction is

$$K_{(3.17)} = \frac{{}^{a\cdot}(Fe^{2+})\,{}^{\cdot a}(s^{2-})}{{}^{a\cdot}[Fe]\,{}^{\cdot a}[s]}$$

The ionic theories attempt to compute the activities of ions to determine the equilibrium constant. Various attempts have been made in this respect. A reasonably satisfactory theory has been proposed by Flood and co-workers[2] in that the slag is assumed to be a mixture of two independent regular solutions, one of cations and other of anions. On this basis the activities were evaluated as

$$a_{(\mathrm{FeS})} = N'_{\mathrm{Fe}}{}^{2+}\, N'\, \mathrm{S}^{2-} \tag{3.18}$$

where *N'* is the electrically equivalent fraction of the ion. The cations alone determine the fraction of any cation and the anions alone of any anion. If the valency of the ions is same the ionic fraction is equal to *N'* but if they differ the ion fraction is different and *N'* will have to be calculated involving permutations, Sterling's approximation and so on. For example, for a cation mixture of A^{+}, B^{2+}, C^{3+}, D^{4+} and so on, ions the electrically equivalent fraction for C^{3+} ions is given by

$$N'_{C^{3+}} = \frac{3n_{C^{3+}}}{n_{A^{+}} + 2n_{B^{2+}} + 3n_{C^{3+}}\, 4n_{D^{4+}} + \cdots} \tag{3.19}$$

which is nothing but a charge fraction of that ion with respect to total like charges. The true equilibrium constant can then be represented for an anionic equilibrium in terms of cation fractions and for cationic equilibrium in terms of anionic fractions. This is obvious because the oppositely charged particles alone will have the necessary bond for retaining it in the slag phase. The following method illustrates the method of calculation. The partitioning of sulfur between iron melt and a complex slag can be written as

$$[\mathrm{S}] + (\mathrm{O}^{2-}) = (\mathrm{S}^{2-}) + [\mathrm{O}] \tag{3.20}$$

The equilibrium constant can be represented as

$$\log \mathrm{K}_{(3.20)} = N'_{\mathrm{Ca}^{2+}} + \log k_{\mathrm{Ca}^{2+}} + N'_{\mathrm{Fe}^{2+}} + \log k_{\mathrm{Fe}^{2+}} + \cdots \tag{3.21}$$

where the $k_{\mathrm{Ca}}{}^{2+}$ refers to the equilibrium constant of a simple reaction like

$$[\mathrm{S}] + (\mathrm{CaO}) = (\mathrm{CaS}) + [\mathrm{O}]$$

and which can be experimentally determined. Similarly the $k_{\mathrm{Fe}^{2+}}$, $k_{\mathrm{Mg}^{2+}}$ $k_{\mathrm{Mn}^{4+}}$, etc., refer to the corresponding simple reactions. The ability of the slag to retain a negative ion like sulfur is thus determined solely by the various cations present in the slag, since the oppositely charged ions alone will have the necessary bond for its retention. The anions will not play any part in the anion partitioning.

Since even these values of simpler reactions are not found out without some assumptions, the true equilibrium constant of Equation (3.20) is generally replaced by equilibrium quotient as

$$\log K'_{(3.20)} = N'_{Ca^{2+}} \log k'_{Ca^{2+}} + N'_{Mn^{2+}} + \log k'_{Mn^{2+}} + \cdots \tag{3.22}$$

This is known as Flood's additive function[2] for sulfur transfer between slag and metal.

Similarly for a cationic equilibrium like

$$(Fe^{2+}) + [Mn] = (Mn^{2+}) + [Fe] \tag{3.23}$$

The equilibrium quotient is

$$\begin{aligned}\log K'_{(3.23)} = {} & N'_{SiO_4^{4-}} \cdot \log k'_{SiO_4^{4-}} + N'_{Si_2O_7^{6-}} \log k'_{Si_2O_7^{6-}} \\ & + N'_{PO_4^{3-}} \cdot \log k'_{PO_4^{3-}}\end{aligned} \tag{3.24}$$

where $k'_{Si_4O^{4-}}$ refers to a simple reaction

$$(2Fe \cdot SiO_4) + [Mn] = (2Mn.SiO_4) + [Fe] \tag{3.25}$$

and the others to the corresponding simple reactions. The evaluation of even the K' values in Equation (3.22) will also be a great achievement in itself, let alone the true values of equilibrium constants.

Since the precise nature of the anions is known only in basic slags, the cationic equilibria can be analyzed for these slags with tolerable accuracy. The majority of steel making reactions are between basic slag and metal, and can therefore, be analyzed with tolerable accuracy. The reaction equilibrium in acid steel making processes, however, cannot be assessed with equal certainty since the precise nature of the Si-O, Al-O, etc., complexes are not known in these slags.

Although the assumptions involved in the ionic theory are open for justifiable dispute, it has definitely explained many of the observations on the reactions between slag and other phases satisfactorily. The ionic interpretation, being based on the more realistic nature of slag, and also since it involves a limited number of ionic species, needs to be followed up still further to explain the reactions of slags with other phases like furnace gases and furnace refractory lining.

3.4. Iron and Steel Slag (USA 2017)

SOURCE: U.S. Geological Survey, Mineral Commodity Summaries, January 2017

(Data in million metric tons unless otherwise noted)

3.4.1 Domestic Production and Use

Iron and steel (ferrous) slags are coproducts of iron and steel manufacturing. After cooling and processing, ferrous slags are sold primarily to the construction industry. Data are unavailable on actual U.S. ferrous slag production, but it is estimated to have been in the range of 15 to 20 million tons in 2016. Domestic slag sales1 in 2016 amounted to an estimated 18 million tons, valued at about $350 million (ex-plant). Iron (blast furnace) slag accounted for about 47% of the tonnage sold and had a value of about $300 million; about 85% of this value was from sales of granulated slag. Steel slag produced from basic oxygen and electric arc furnaces accounted for the remainder.2 Slag was processed by about 29 companies servicing active iron and steel facilities or reprocessing old slag piles at about 135 processing plants in 31 States; included in this tally are some facilities that grind and sell ground granulated blast furnace slag (GGBFS) based on imported unground feed.

Prices listed in the table below are weighted averages (rounded) for iron and steel slags sold for a variety of applications. Actual prices per ton ranged widely in 2016, from a few cents for some steel slags at a few locations to about $110 for some GGBFS. Air-cooled iron slag and steel slag are used primarily as aggregates in concrete (aircooled iron slag only), asphaltic paving, fill, and road bases; both slag types also can be used as a feed for cement kilns. Almost all GGBFS is used as a partial substitute for portland cement in concrete mixes or in blended cements. Pelletized slag is generally used for lightweight aggregate but can be ground into material similar to GGBFS. Owing to low unit values, most slag types can be shipped only short distances by truck, but rail and waterborne transportation allow for greater distances. Because of much higher unit values, GGBFS can be shipped longer distances, including from overseas.

Salient Statistics—United States:	**2012**	**2013**	**2014**	**2015**	**2016[e]**
Production (sales)[1, 3]	16.0	15.5	16.6	17.7	18.0
Imports for consumption[4]	1.2	1.7	1.8	1.4	2.0
Exports	(5)	(5)	0.1	(5)	(5)
Consumption, apparent[4, 6]	16.0	15.5	16.5	17.7	18.0
Price average value, dollars per ton, f.o.b. plant[7]	17.00	17.50	19.00	19.50	19.50
Employment, number[e]	1,800	1,700	1,700	1,700	1,600
Net import reliance[8] as a percentage of apparent consumption	7	11	10	8	11

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[e] *Estimated.*

[1] *Data are from an annual survey of slag processors and pertain to the quantities of processed slag sold rather than that processed or produced during the year. The data exclude any entrained metal that may be recovered during slag processing and then sold separately or returned to iron and, especially, steel furnaces. The data are incomplete regarding slag returns to the furnaces.*

[2] *There were very minor sales of open hearth furnace steel slag from stockpiles but no domestic production of this slag type in 2012–16.*

[3] *Data include sales of imported granulated blast furnace slag, either after domestic grinding or still unground, and exclude sales of pelletized slag (proprietary but very small). Overall, actual production of blast furnace slag may be estimated as equivalent to 25% to 30% of crude (pig) iron production and steel furnace slag as about 10% to 15% of crude steel output.*

[4] *Based on official (U.S. Census Bureau) data. In some years, the official data appear to have understated the true imports; the apparent discrepancy was small for 2012, but may have been nearly 0.4 million tons in 2013 and 2014, depending on whether imports from Italy were mischaracterized as being from Spain or not. The U.S. Geological Survey canvass captures only part of the imported slag.*

[5] *Less than 0.05 million tons.*

[6] *Although definable as total sales of slag (including those from imported feed) – exports, apparent consumption of slag does not significantly differ from total sales owing to the very small export tonnages.*

[7] *Rounded to the nearest $0.50 per ton.*

[8] *Defined as imports minus exports.*

3.4.2. Recycling

Following removal of metal, slag can be returned to the blast and steel furnaces as ferrous and flux feed, but data on these returns are incomplete. Entrained metal, particularly in steel slag, is routinely recovered during slag processing for return to the furnaces, and is an important revenue source for the slag processors, but data on metal returns are unavailable.

3.4.3. Import Sources (2012–15)

The dominant imported ferrous slag type is granulated blast furnace slag (mostly unground), but official import data in recent years have included significant tonnages of nonslag materials (such as cenospheres, fly ash, and silica fume) and slags or other residues of various metallurgical industries (such as copper slag) whose unit values are outside the range expected for granulated slag. The official data appear to have underreported the granulated slag imports in some recent years, but likely not in 2011–12. Based on official data, the principal country sources for 2012–15 were Japan, 33%; Canada, 31%; Spain, 16%; Germany, 5%; and other, 15%; however, much of the tonnage from Spain in 2013–14 may in fact have been from Italy, and slag from the Netherlands and Switzerland in 2015 may have been from Germany.

3.5. Events, Trends, and Issues

The supply of blast furnace slag continues to be problematic in the United States because of the closure and (or) continued idling of a number of active U.S. blast furnaces in recent years, including one in 2015, the lack of construction of new furnaces, and the depletion of old slag piles. Only a limited quantity of locally produced granulated blast furnace slag was available. At yearend 2016, granulation cooling was available at only two active U.S. blast furnaces, down from three in 2014. Installation of granulation cooling was being evaluated at a few blast furnaces, but it was unclear if this would be economic. Pelletized blast furnace slag was in very limited supply (one site only), and it was uncertain if any additional pelletizing capacity was planned.

Basic oxygen furnace steel slag from domestic furnaces has become less available recently because of the closure of several integrated iron and steel complexes; thus, the long-term supply of steel slag will be increasingly reliant on electric arc furnaces, which now contribute the majority of U.S. steel production. Where not in short supply, slag (as aggregate) sales to the construction sector tend to fluctuate less than those of natural aggregates. Domestic- and import-supply constraints appear to have limited domestic demand for GGBFS in recent years. Although prices have increased, sales volumes have failed to match the relative increases that have characterized the overall U.S. cement market since 2010. Long-term demand for GGBFS likely will increase because its use in concrete yields a superior product in many applications and reduces the unit carbon dioxide (CO_2) emissions footprint of the concrete related to the portland cement (clinker) content. Recent regulations to restrict emissions of CO_2 and mercury by coal-fired powerplants, together with some powerplant closures or conversion of others to natural gas, have led to a reduction in the supply of fly ash in some areas, including that of material for use as cementitious additive for concrete. This has the potential to increase future demand for GGBFS, but the availability of material to supply this demand will increasingly depend on imports, either of ground or unground material. Imported slag availability may be constrained by increasing international demand for the same material and because not all granulated slag produced overseas is of high quality. New restrictions on mercury emissions by cement plants may reduce demand for fly ash as a raw material for clinker manufacture, and this could lead to use of air-cooled and steel slags as replacement raw materials.

3.5.1. World Mine Production and Reserves

Slag is not a mined material and thus the concept of reserves does not apply to this mineral commodity. Slag production data for the world are unavailable, but it is estimated that global iron slag output in 2016 was on the order of 300 to 360 million tons, and steel slag about 160 to 240 million tons, based on typical ratios of slag to crude iron and steel output.

3.5.2. Substitutes

In the construction sector, ferrous slags compete with crushed stone and sand and gravel as aggregates, but are far less widely available than the natural materials. As a cementitious additive in blended cements and concrete, GGBFS mainly competes with fly ash, metakaolin, and volcanic ash pozzolans, and to a lesser degree with silica fume. In this respect, GGBFS also competes with portland cement itself. Slags (especially steel slag) can be used as a partial substitute for limestone and some other natural raw materials for clinker (cement) manufacture. Some other metallurgical slags, such as copper slag, can compete with ferrous slags in some specialty markets but are generally in much more restricted supply than ferrous slags.

CHAPTER 4

PRINCIPLES OF STEEL MAKING

In This Chapter

- Introduction
- Thermodynamics of Refining
- Kinetics of Refining
- Thermal Principles of Refining
- Deoxidation of Steel
- Plain Carbon Steel Production
- Alloy Steel Production
- Tapping Temperature
- Production of Ingots by Casting
- Control of Refining

4.1. Introduction

As has already been stated, steel making is a refining or an oxidation process, with the exception of reducing conditions being specifically required to eliminate sulfur. The necessity of refining pig iron arises because it is quite impure. Steel making can also be viewed as an attempt to rectify the inevitable mistake made by the iron producer earlier by producing such an impure material. Being an oxidation process steel making needs a source of oxidizing agent. The obvious choice being the oxygen in the form of atmospheric air and/or iron oxide in the form of iron ore and mill scale (oxide produced during hot working of steels). The conventional methods of steel making use either or both of these reagents. The oxygen steel making processes use pure oxygen whereas the modified conventional processes use iron oxide, air and pure oxygen judiciously.

The chemistry of a steel making refining process can be simply described as

$$\underline{Fe} + \underline{O} = (FeO) \qquad \ldots(4.1)$$

$$\underline{C} + \underline{O} = (CO) \qquad \ldots(4.2)$$

$$\underline{Si} + \underline{2O} = (SiO_2) \qquad \ldots(4.3)$$

$$\underline{Mn} + \underline{O} = (MnO) \qquad \ldots(4.4)$$

$$\underline{2P} + 5\underline{O} = (P_2O_5) \qquad \ldots(4.5)$$

$$\underline{S} + (CaO) = (CaS) + \underline{O} \qquad \ldots(4.6)$$

The products in the above reactions are only those that are stable at steel making temperatures. The oxides that are not thermodynamically stable at steel making temperatures need not be considered here. The choice of molecular interpretation in the above reactions is purely for the sake of simplicity. Except the sulfur reaction all the rest are oxidation processes and are favored under the oxidizing conditions of steel making. In the following discussion, therefore, the oxidation reactions are dealt with in general and the sulfur reaction is described separately.

In the case of oxidation of carbon, the product, being a gas, passes off into the atmosphere but the rest of the oxide products shall remain in contact with the iron melt in the form of a slag phase. In steel making the reactions in Equations (4.2) to (4.6) should move to the right in preference to the oxidation of iron and that the danger of reversion of an impurity to the metal phase is as remote as possible. From the point of view of law of mass action the required conditions can be achieved by increasing the activities of the reactants and decreasing those of the products. For a given composition of iron melt, the activity of the impurity is fixed and hence cannot be increased. The oxidizing potential of an oxidizing agent can be increased. The oxidizing potential of an oxidizing agent can be increased by using atmospheric air ($a_O = 0.21$) in place of iron oxide ($a_O \cong 10^6 - 10^{-8}$ in slag phase) and pure oxygen ($a_O = 1$) in place of air. But once the nature of the oxidizing agent is chosen it cannot be increased. The activity of the product can however be decreased by combining it with oxide of opposite chemical character, i.e., an acid oxide product is mixed with basic oxide and vice versa. As far as the physical requirement of the oxide product is concerned it should be readily separable from the iron melt. This is achieved by keeping the slag and the metal both as thin liquids so that the metal being heavier settles down and the slag floats on top, in the form of two immiscible liquids that can be separated readily.

If the oxide products of iron refining reactions are examined, silicon and phosphorus form acid oxides and hence a basic flux is needed to form a suitable slag for their effective removal. The higher the proportion of base available, the lower the danger of backward reaction. For manganese elimination, since manganese oxide is basic, an acid flux will be required. The nature of the process itself has made the task somewhat simpler. During refining, being the largest bulk, iron itself gets oxidized to some extent as (FeO) which is basic in nature. It is possible to adjust the contents of silicon and manganese in pig iron such that the amounts of (FeO + MnO) formed during refining would be able to form a slag essentially of the type FeO-MnO-SiO_2 and fix up silica in it. In such a slag P_2O_5 is not stable because (FeO + MnO) together are not strong enough bases to fix it up in slag. In order to oxidize phosphorus in preference to iron, a strong external base like CaO and/or MgO is needed in sufficient proportion to form a basic slag to hold P_2O_5 without any danger of its reversion. Phosphorus is best eliminated by a slag of the type CaO-FeO-P_2O_5. It is quite interesting to note that such a slag is also capable of removing sulfur from iron melt to a certain extent.

The steel making processes can now be divided into two broad categories:

(*i*) when silicon is the chief impurity to be eliminated from iron and that phosphorus and sulfur need not be eliminated and, (*ii*) when phosphorus, and to some extent sulfur, are the chief impurities to be eliminated along with even some silicon.

The elimination of manganese will take place under both categories.

In the finished steel, with a few exceptions, phosphorus and sulfur each must be below 0.05%. If phosphorus is above this limit, steel becomes cold-short and if sulfur is more it becomes hot-short. Higher sulfur contents are recommended for free-cutting variety of steels and a slightly high phosphorus level is desirable for efficient pack rolling of steel sheets.

If the pig iron composition is such that phosphorus and sulfur both are below 0.05% and, therefore, need not be eliminated, it is possible to remove silicon along with manganese in such a way that slag of the type MnO-FeO-SiO_2 is formed without the necessity of addition of an external flux. Such a process of steel making is called acid steel making process, which is carried out in an acid brick-lined furnace. On the other hand, to eliminate phosphorus and sulfur, the reverse reaction rate can only be suppressed if the slag contains a good amount of stronger base than as is internally available in the form of FeO and MnO. External CaO (and also MgO) is used as a flux and slag of the type CaO-FeO-P_2O_5 is made. Such a process is called basic steel making process. The furnace lining in this case has to be basic in nature.

In brief the composition of pig iron is the only factor that determines the acid or the basic character of the process to be adopted for steel making. In an acid process slag is acidic and the furnace lining has to be acidic to withstand the slag. Similarly in a basic process the slag contains excess basic oxide and the furnace lining should be basic in nature. If the lining is of the opposite chemical character, slag will readily react with the lining and cause damage to the furnace. Besides the acid or the basic nature, the slag needs to possess many other physical and chemical properties to carry out refining efficiently.

The refining reactions during steel making essentially take place at the slag metal interface with the exception of carbon reaction, which takes place at the gas-metal interface.

4.2. Thermodynamics of Refining

For the generalized refining reaction

$$[X] + [O] = XO \text{ (any phase)} \qquad \ldots(4.7)$$

(where X stands for any element dissolved in iron) the equilibrium constant

$$K_{(4\cdot7)} = \frac{a_{XO}}{{}^{a}[X] \cdot {}^{a}[O]}.$$

Evaluation of $K_{(4.7)}$ needs the value of either free energy change of the reaction or the activity coefficient value of X, O, and XO. The activity coefficient values of various solute elements have been experimentally determined in binary Fe–X and ternary Fe-*X*-*i* iron melts. The effect of concentration of the third element *i* on the activity coefficient of *X* is generally represented in the form of plots between log f_X^i and the concentration of *i*. The diagrammatic representation of this type can be used to estimate graphically the value of f_X in any complex melt. These semi-log plots are generally linear at low concentrations of X and i so that they can be represented by an equation of the type

$$\log f^i_X = e^i_X\,[\%i] \qquad \ldots(4.8)$$

where e_X^i is known as the interaction coefficient expressing the effect of i on X. Knowing the interaction coefficient the Henrian activity of a solute in a complex melt can be mathematically computed as

$$\log \mathrm{h}_X = e^x_X \cdot \%X + e^{x1}_X \cdot \%X_1 + e^{x2}_X \cdot \%X_2 + \ldots \log \%X \qquad \ldots(4.9)$$

where X, X_1, X_2 etc., are solute elements. The value thus obtained is strictly true in very dilute solutions but in the absence of any more precise information it can be used for concentrations even up to a few percent of a solute element without involving much error. For higher concentrations graphical computation may be more accurate than that calculated according to Equation (4.9).

Similarly the activity of coefficient of oxides for many binary and ternary slags has also been experimentally established. The activities are generally represented in the form of iso-activity lines in the ternary diagram. Since the practical complex slags can be considered in the form of a simplified ternary system, the above data can be used without involving much error. Alternatively the complex slag can be reduced to some form of a pseudo-ternary system, e.g., $[(CaO+MgO-(SiO_2+P_2O_5)-(FeO)]$ and the available activity data can be used. Since the activity values of oxides in complex silicate slags often deviate little from ideality, the error involved in the above simplification is not significant.

The activity of the oxidizing agent can be estimated in the form of partial pressure of oxygen in the gaseous phase and as the activity of iron oxide in the slag phase, as discussed in Chapter 3. Whichever oxidizing agent is used for refining, it will result in correspondingly proportional activity of oxygen dissolved in the iron melt as represented by [O] in Equation (4.7). In the presence of other solute elements the activity coefficient of oxygen in iron melt is given by[3]:

$$\begin{aligned}\log f_{\mathrm{O}} = {} & 0.0085[\%\mathrm{W}] + 0.008[\%\mathrm{C}] + 0.006[\%\mathrm{P}] + 0.006[\%\mathrm{Ni}]\\ & + 0.0035[\%\mathrm{Mo}] - 0.021[\%\mathrm{Mn}] - 0.04[\%\mathrm{Cr}]\\ & - 0.131[\%\mathrm{Si}] - 0.133[\%\mathrm{S}] - 0.2[\%\mathrm{O}] - 0.3[\%\mathrm{V}]\\ & - 0.45[\%\mathrm{Co}] - 3.9[\%\mathrm{A1}] \qquad \ldots(4.10)\end{aligned}$$

During refining as the oxidizing agent is brought in contact with iron melt, oxygen may dissolve in iron up to its solubility limit. Any further supply of oxygen would result in oxidation, i.e., the insoluble oxide will be formed. In general the equilibrium concentration of oxygen in iron will be inversely proportional to that of the solute element in the melt. The precise nature of this relationship for each and every solute element is of great utility in knowing the equilibrium partition conditions. These relationships vary according to the values of the activities of the components involved in the reaction. For a fixed activity of the product the relationship is fixed for a fixed temperature.

4.2.1. Carbon Reaction

The carbon oxygen reaction plays a dominant role in steel making.

The activity of carbon in liquid iron-carbon alloys shows a negative deviation from ideality. The presence of carbide-forming elements like Nb, V, Cr, desulfurizing, Mo, etc., decrease; whereas the presence of non-carbide formers like Cu, Ni, Co, etc. increase the activity coefficient of carbon in iron melts. The activity coefficient is given by[3]:

$$\begin{aligned}\log f_C = {} & + 0.0076[\%Co] + 0.012[\%Ni] + 0.046[\%S] + 0.051[\%P] \\ & + 0.08[\%Si] + 0.14[\%C] - 0.0056[\%W] - 0.0083[\%Mo] \\ & - 0.012[\%Mn] - 0.016[\%Cu] - 0.024[\%Cr] \\ & - 0.06[\%Nb] - 0.077[\%V] - 0.34[\%O] \qquad \ldots(4.11)\end{aligned}$$

The equilibrium constant of carbon reaction in Equation (4.2) is

$$K_{(4.2)} = \frac{p_{CO}}{{}^{h}[c]\cdot{}^{h}[o]}\cdots$$

or

$$f_C[\%C]f_O[\%O] = \frac{p_{CO}}{K_{(4.2)}} \qquad \ldots(4.12)$$

The relationship of carbon and oxygen contents in liquid iron at various carbon monoxide pressures is shown in Figure 4.1. The product of mere weight percentages [%C], [%O] is not constant along any isobaric curve; hence the activity coefficients must be taken into account in the calculation of equilibrium constant. Temperature dependence of equilibrium constant is believed to be

$$\log K_{(4.2)} = \frac{1056}{T} + 2.131 \qquad \ldots(4.13)$$

giving a value of 494, i.e., approximately rounded off to 500 at 1600°C.

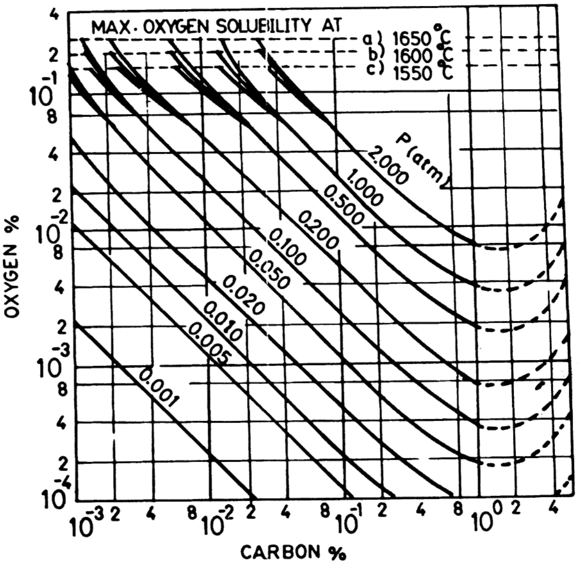


FIGURE 4.1. The equilibrium carbon and oxygen contents in molten iron with carbon mono-oxide at various pressures (after, Bauer Fleischer and Otto[4]).

It is thus possible to produce steels with less than 0.1%C using pure oxygen as an oxidizing reagent (oxygen steel making processes). The product of carbon reaction being gas, passes off into the atmosphere and the reaction (4.2) always tends to go in the forward direction. The carbon content could be decreased much below the above level by treating it under vacuum (vacuum treatment of steel) or under inert gas atmosphere.

4.2.2. Phosphorus Reaction

Although the boiling point of phosphorus is very low, a considerable amount of it still remains dissolved in liquid iron because of its strong interaction with iron. It is experimentally difficult to determine the interaction parameters of phosphorus with other solutes, thus only a few of these values are known. The activity coefficient is given by[3]

$$\begin{aligned}\log f_{P} = {} & 0.13[\%C] + 0.13[\%O] + 0.12[\%Si] + 0.062[\%P] \\ & + 0.024[\%Cu] + 0.028[\%S] + 0.006[\%Mn] \\ & 0.0002[\%Ni] - 0.03[\%Cr] \qquad \ldots(4.14)\end{aligned}$$

and the presence of other solutes is assumed to have no influence on the above equation.

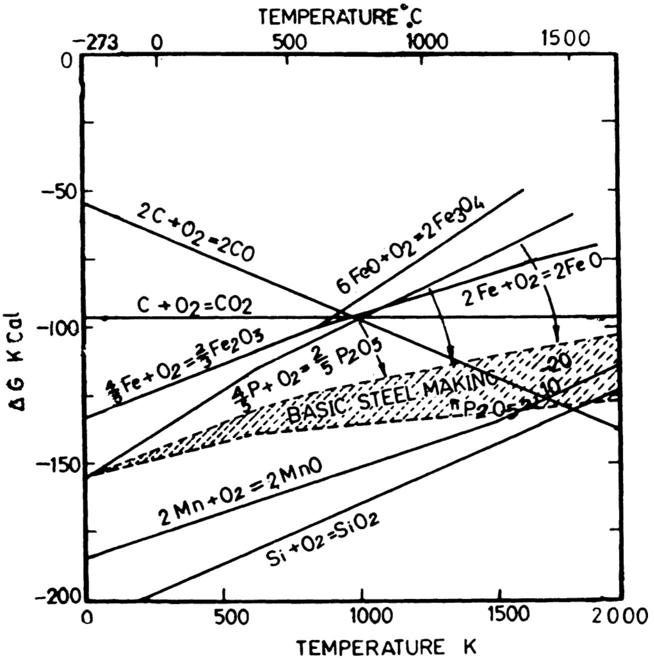


FIGURE 4.2. The oxide Ellingham diagram of interest in refining basic pig iron. Decreasing the activity of the product oxide results in rotation of the line in clockwise direction. By employing highly basic slags, the iron and the phosphorus lines are thus widely separated. The shaded area indicates the basic steel making conditions.

The iron and phosphorus lines on the Ellingham diagram are so close to each other that the entire phosphorus in the burden gets reduced along with iron in an iron blast furnace. This might have posed a serious problem in the oxidation of phosphorus but for the

The fact that the two lines are widely separated during steel making by decreasing the activity of (P_2O_5) using a strong and excess external basic flux like lime. The standard and non-standard lines on the Ellingham diagram are shown in Figure 4.2. Slag with a basicity of 2-4 decreases the activity coefficient of (P_2O_5) to $10^{-15} - 10^{-20}$. The steel making slags may contain up to 25% P_2O_5 but even then the activity of (P_2O_5) in slag remains extremely low. For effective removal of phosphorus, basic steel making processes must employ slags of high basicities. If the basicity falls, phosphorus may soon revert to the metal phase. In acid steel making processes, since the slag is nearly saturated with silica, phosphorus cannot be eliminated at all.

The partitioning of phosphorus between slag and metal can be described as:

$$2[P] + 5(FeO) + 3(CaO) = (3CaO.P_2O_5) + 5[Fe] \qquad \ldots(4.15)$$

or

$$2[P] + 5[O] + 3(O^{2-}) = 2(PO_4^{\ 3-}) \qquad \ldots(4.16)$$

and the equilibrium constants are

$$K_{(4.15)} = \frac{a_{(3CaOP2O5)} \cdot a^5_{[Fe]}}{a^2_{[P]} \cdot a^5_{(FeO)} \cdot a^3_{(CaO)}}$$

or

$$K_{(4.16)} = \frac{a_{(PO4)^{3-}}}{a^2_{[P]} \cdot a^5_{[O]} \cdot a^3_{(O^{2-})}}$$

The ratio of phosphorus content in slag to that in metal is generally referred to as dephosphorizing index D_P which has a higher value for higher basicity and higher oxidizing power of the slag.

The additivity function for phosphorus distribution between slag and metal as obtained by Flood and coworkers[2] is

$$\log K' = 21N'_{Ca^{2+}} + 18N'_{Mg^{2+}} + 13N'_{Mn^{2+}} + 12N'_{Fe^{2+}} + \ldots \qquad \ldots(4.17)$$

i.e. the dephosphoring ability of various basic oxide cations is in the ratio of

	Ca	:	Mg	:	Mn	:	Fe
	10^{21}	:	10^{18}	:	10^{13}	:	10^{12}
or approximately	30,000	:	1000	:	3	:	1

In other words, lime is 30 times more effective on molar basis and 20 times more effective on weight basis for dephosphorization than magnesia. The (FeO) is only effective as an oxidizing agent and not as a base. The interrelationships of D_P, basicity and iron oxide content found in laboratory experiments are shown in Figure 4.3. The precise nature of these curves varies with the changes in melt composition but the general nature should remain the same. The dephosphorizing index increases with increase in the (FeO) content up to 15% because of its high oxidizing ability. If (FeO) content increases still further the proportion of lime is decreased, hence

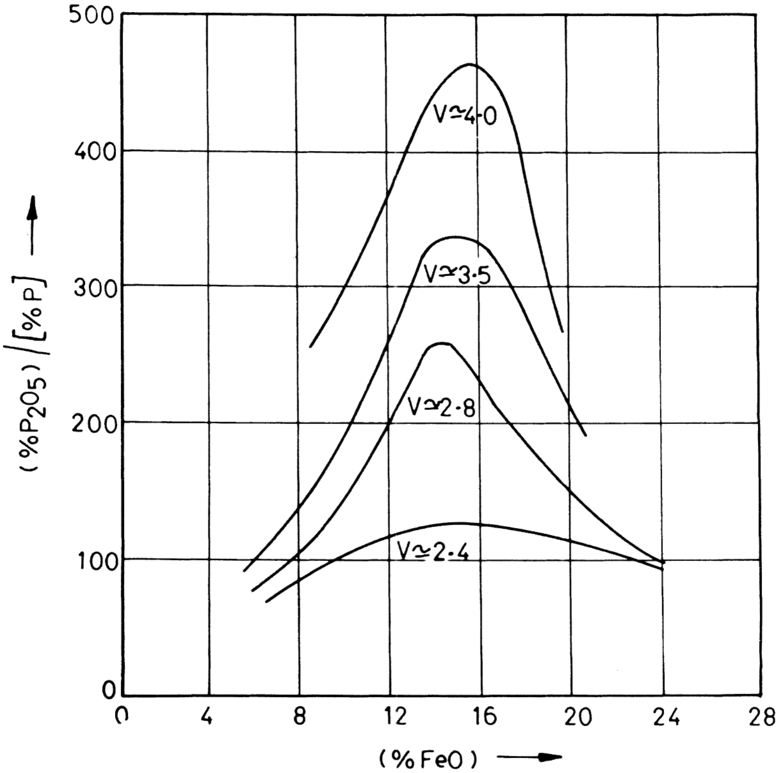


FIGURE 4.3. The variation or dephosphorizing index $(\%P_2O_5)/[\%P]$ with iron oxide content of slags of various basicities under laboratory conditions. The figures on the curves are values of V-ratios $(\%CaO)/(\%SiO_2)$. (after Balajiva, Quarrel and Vajragupta[5])

the advantage of high oxidizing ability of (FeO) is neutralized by the loss of time in the slag resulting in a maximum dephosphorizing index at around 15% (FeO). An increase in temperature tends to lower the value of D_p and vice versa, hence dephosphorization is more effective at low temperatures. It means that the activity coefficient f_{FeO} has a maximum value when the basicity is around 1.8 to 2.0, and that it depends on slag composition.

Soda ash is 100 times superior to lime, on molar basis, for dephosphorization, but its use in practice is forbidden because of its severe corrosive action on furnace refractories. Usually the magnesia content of a basic steel making slag is not under control because it reaches equilibrium with furnace lining. Similarly (MnO) content is dictated by charge material and is not very adjustable. The steelmaker is, therefore, left with the control on lime, silica, and iron oxide content of the slag to effect the required dephosphorization of the melt.

Since phosphorus oxidation is highly exothermic reaction, it is favored by lower temperatures. Excessively high temperature, raised in the course of refining, may lead to reversion of phosphorus from slag to metal during steel making in actual practice.

For practical purposes, the empirical relationship for dephosphorization that was used by McCance[6] was:

$$E_p = \frac{(\%P_2O_5)}{[\%P]\cdot(\%P_2O_5 + \%FeO)}$$

The depiction of dephosphorizing index in this way and evaluation of phosphorus removal in basic steel making was good enough for slow-hearth processes, in which the end-phosphorus levels that were required in those days were not too low. With fast steel making processes and requirements of very low-end phosphorus contents, the phosphorus partitioning in steel making must occur at a very high efficiency. For such processes it is necessary to predict end phosphorus as a function of operating conditions of the shop. From this point of view the equilibrium quotient $K_{(4,\ 16)}$ is depicted as

$$K_p = [(\%P)/\%P] \times \%O^{-2.5}$$

which Turkdogan [7] calls as phosphate capacity of slags. The phosphate capacity did not show any specific relationship with the basicity or basic oxide contents of commercial slags. But it depicts a straight-line relationship when the basic oxide contents are expressed as:

$$BO = [(\%CaO + \%CaF_2 + 0.3\ \%\ MgO)$$

as is shown in Figure 4.4. It indicates the proportionate influence of each of the basic oxides in the slag that need to be taken into account while evaluating their abilities for dephosphorization.

Likewise the phosphate capacity varies with temperature inversely and that

$$\log K_p = 21740/T - 9.87 + 0.071(\text{Basic Oxides})$$

It should be noted that the dephosphorization is essentially governed by the iron oxide content of the slag, and not by the oxygen dissolved in

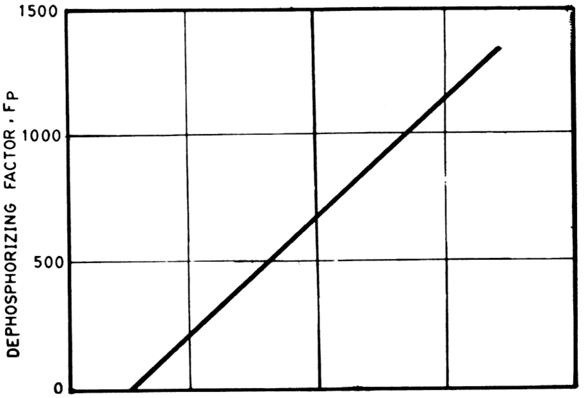


FIGURE 4.4. Depicts the phosphate capacity of the slags as a function of some specific empirical formula.

melt at a given basicity, over a wide range of carbon contents of steel. The two, though interrelated, need to be understood separately. The dephosphorizing capacity of the practical slag can be artificially increased by allowing the slag to flush out of the furnace and by preparing a fresh basic slag in actual operation.

Similarly for obtaining very low phosphorus steel, synthetic slags with high phosphate capacity may be used externally. This is called external dephosphorization.

Different formulae have been proposed by different workers including Healy, Turkdogan, Suito, Lonui, Young, and so on, to predict end phosphorus in a given practice. These propositions differ considerably [8]. Hence every plant has to develop one's own relationships for their own plant practices to predict end-phosphorus contents, particularly in faster BOF processes. The model suggested at Tat-Steel is [8].

$$\log\{(\%P)/\%P\} = A \times (/T) + B \times (\%CaO) + C \times \log(\%Fe_{Total}) + D$$

where A, B, C and D are constants and which must be empirically established in a given practice from a very large number of collected-heat-data.

One thing is certain, in general, that the higher the CaO content of the slag, the better the chances of getting correct end-phosphorus, provided adequate iron oxide is present in the slag.

In brief high basicity, low temperature and (FeO) content around 15% favor effective dephosphorization of metal by slag.

4.2.3. Silicon Reaction

Silicon content decides the grade of pig iron. It strongly interacts with iron to produce appreciable negative deviation from ideality. The activity coefficient in the presence of other elements is given by[3]:

$$\begin{aligned}\log fs_i &= 0.18[\%C] + 0.11[\%P] + 0.11[\%Si] + 0.058[\%A1]\\ &= 0.056[\%S] + 0.025[\%V] + 0.014[\%Cu]\\ &\quad + 0.005[\%Ni] + 0.002[\%Mn] - 0.0023[\%Co]\\ &\quad - 0.23[\%O] \qquad \ldots(4.18)\end{aligned}$$

Activity of silica in slags of interest in steel making is known to a large extent. Silica is invariably present in refining slags, and as such is a very important factor in determining its basicity. Silica is a very stable oxide, hence once silicon is oxidized to silica the danger of its reversion does not usually arise in refining slags.

The equilibrium constant of reaction in Equation (4.3) at unit activity of silica is given by:

$$K_{(4.3)} = \frac{1}{a_{[Si]} \cdot a^2_{[O]}}$$

or the quotient as $K'_{(4.3)} = \dfrac{1}{[\%Si] \cdot [\%O]^2}$

and $$\log \frac{1}{K'_{(4.3)}} = 11.01 - \frac{29150}{T} \qquad \ldots(4.19)$$

and the value of $K'_{(4.3)}$ has been found to be constant even up to 15% Si. It is due to the lowering of the activity of oxygen being compensated by the corresponding increase in the activity of silicon. This constancy gives rise to the linear relationship of log [%Si] *vs.* log [%O] in steel in equilibrium with silica-saturated slags.

In acid steel making processes slag is nearly saturated with silica and the above relationship should hold true. But since these slags contain (MnO) and the metal contains [Mn], the relationship of [%Si] and [%O] should also take into account the influence of [%Mn] on the above equilibria. This is further investigated during the discussion on manganese reaction.

In basic steel making the activity of silica is so low that its preferential oxidation never poses any problem like that of phosphorus removal.

The silicon content of iron in a basic process should, however, be as low as possible in order to decrease lime consumption in maintaining the basicity required for effective phosphorus elimination at minimum of slag volume. If excess silicon enters basic steel making furnace, either slag flushing may be required or more than one slag may be made. Alternatively the high silicon-bearing pig iron is treated outside the blast steel making furnace. This is known as external desiliconization of hot metal [see Sections (5.8.2) and (11.4)].

4.2.4. Manganese Reaction

Next to silica, MnO is the most stable oxide product during refining of pig iron. Iron and manganese form ideal solutions, and even in the presence of other solutes, ideal behavior is assumed. MnO has extensive solubility in slags. It is fashionable to describe manganese distribution as

$$(\mathrm{FeO}) + \mathrm{Mn} = (\mathrm{Mno}) + \mathrm{Fe} \qquad \ldots(4.20)$$

for which

$$K_{(4\cdot 20)} = \frac{a_{(\mathrm{MnO})} \cdot a_{[\mathrm{Fe}]}}{a_{(\mathrm{FeO})} \cdot a_{[\mathrm{Mn}]}}$$

$$K'_{(4\cdot 20)} = \frac{(\%\mathrm{MnO})}{(\%\mathrm{FeO})\,[\%\mathrm{Mn}]}$$

Using the laboratory and plant data the following relationship has been estimated

$$\log K_{(4\cdot 20)} = \frac{7940}{T} - 3.17$$

The dependence of $K'_{(4.20)}$ on slag composition is of interest in steel making and it is shown in Figure 4.5. The activity coefficient of manganese oxide increases with basicity, hence the ratio (%MnO)/[%Mn] decreases with basicity. For the same basicity this ratio increases with oxidizing power of slag.

The interrelationships of silicon, manganese and oxygen contents of iron in equilibrium with silica saturated slags of the type SiO_2-MnO-FeO are of direct interest in acid steel making, and are shown in Figure 4.6. It indicates that for practical slags containing up to 20% (MnO), manganese in metal shall be a maximum of 0.1% as is usually observed in practice.

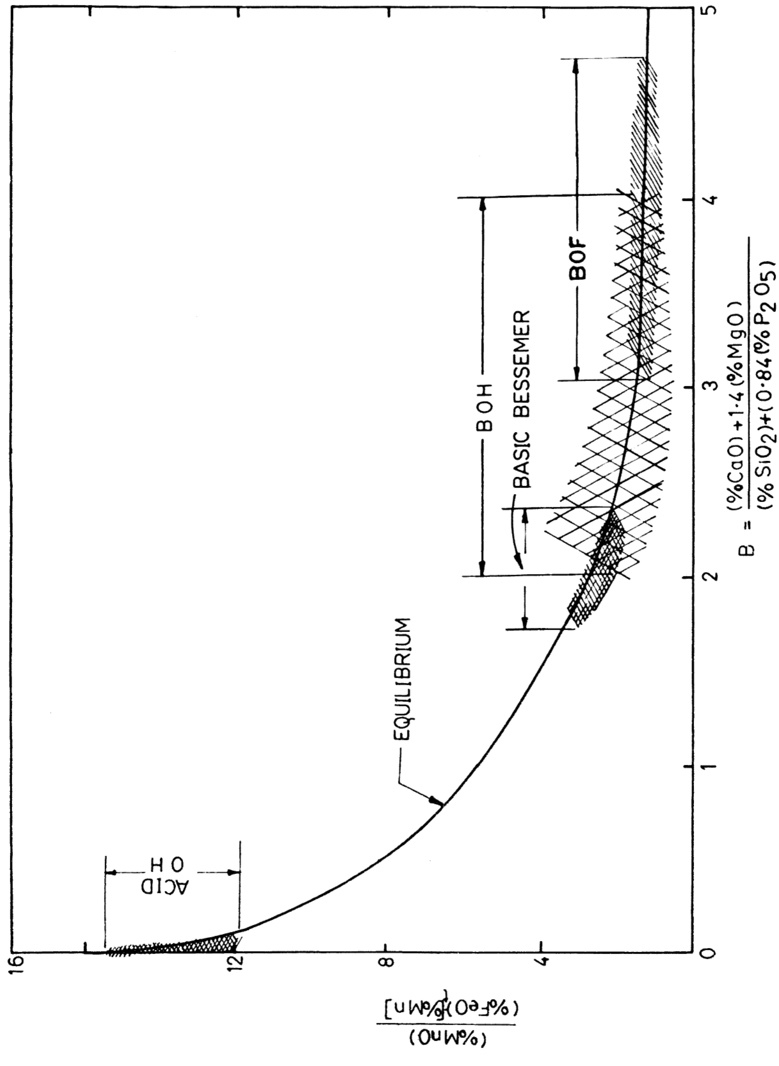


FIGURE 4.5. Variation of equilibrium quotient for manganese with basicity of slag at steel making temperatures. The shaded areas indicated variations in commercial practices.

In basic steel making practice, since the activity coefficient of (MnO) is much higher than that in acid practice, a higher equilibrium [%Mn] content is found.

For $$(Fe^{2+}) + [Mn] = (Mn^{2+}) + [Fe] \quad \ldots(4.21)$$

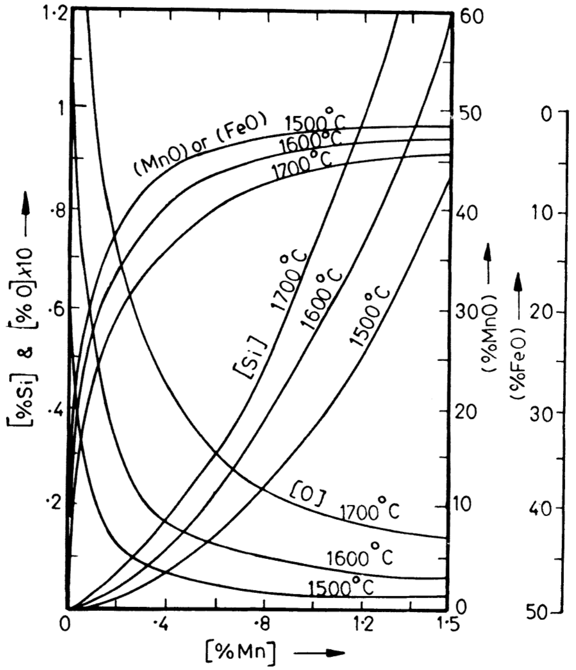


FIGURE 4.6. Variation of [Mn], [Si], and [O] contents of molten iron in equilibrium with silica saturated FeO-MnO-SiO_2 slags. (after Korber[9], adopted from[10])

for which the additive function has been found by Flood and co-workers[2] at 1600°C as

$$\log K'_{(4\cdot21)} = 2\cdot9N'_{SiO_4^{4-}} + 3\cdot0N'_{FeO_4^{5-}} + 2\cdot5N'_{PO_4^{6-}} + 2\cdot4N'_{O^{2-}} + 1\cdot5N'_{F^-} \qquad \ldots(4.22)$$

The observed value is in close agreement with the estimated value.

It has also been observed in practice that any sudden change in manganese content of the bath tends to move it in the direction of equilibrium.

4.2.5. Sulfur Reaction

Although sulfur has a very low boiling point like phosphorus, a considerable amount of it is found in liquid iron because of its strong interaction with iron. The activity coefficient of sulfur can be estimated as[3]:

$$\begin{aligned}\log fs = {} & 0.29[\%P] + 0.11[\%C] + 0.063[\%Si] + 0.035[\%A1] \\ & + 0.0097[\%W] + 0.0027[\%Mo] + 0.0026[\%Co] \\ & - 0.0984[\%Cu] - 0.011[\%Cr] - 0.013[\%Nb] \\ & - 0.016[\%V] - 0.026[\%Mn] - 0.027[\%O] \\ & - 0.28[\%S] - 0.952[\%Zr] \qquad \ldots(4.23)\end{aligned}$$

sulfur can be assumed to exist as CaS in slag up to a few percent concentration and Henrian behavior is usually assumed. Partitioning of sulfur between slag and metal can be described as

$$[FeS] + (CaO) = (CaS) + (FeO) \qquad \ldots(4.24)$$

or

$$[S] + (O^{-2}) = (S^{2-}) + (O) \qquad \ldots(4.25)$$

The equilibrium constants are

$$K_{(4\cdot24)} = \frac{a_{(CaS)} \cdot a_{(FeO)}}{a_{[FeS]} \cdot a_{(CaO)}}$$

$$\text{and } K_{(4\cdot25)} = \frac{a_{(S^{2-})} \cdot [0]}{a_{[S]} \cdot a_{(O^{2-})}}$$

Assuming Henrian behavior and rearranging the terms

$$\frac{(\%S)}{[\%S]} = K_{(4\ 25)} \cdot \frac{(\%CaO)}{(\%FeO)}$$

and also

$$= K'_{(4\cdot25)} \cdot \frac{NO^{2-}}{[\%O]} \qquad \ldots(4.26)$$

where the (%S)/[%S] is known as desulfurizing index, and its variation with basicity and oxidizing power of slag is shown in Figure 4.7. For a fixed basicity the index is inversely proportional to the iron oxide content of the slag or the [O] content of the metal, i.e.,

$$\begin{aligned}\frac{(\%S)}{[\%S]} & \propto \frac{1}{(\%FeO)} \\ & \propto \frac{1}{(\%O)} \qquad \ldots(4.27)\end{aligned}$$

Similarly for a fixed (FeO) content of slag

$$\frac{(\%S)}{[\%S]} \propto (\%CaO) \qquad \ldots(4.28)$$

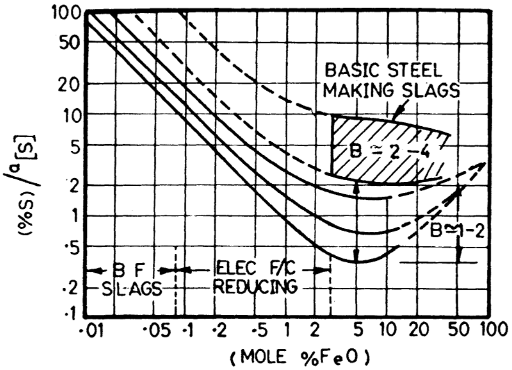


FIGURE 4.7. Desulfurizing ability of slag as affected by iron oxide content and basicity of slags. Shaded area shows the range of basic steel making slags. B values indicate the basicity of slags. (after Rocca, Grant and Chipman[11]).

For efficient removal of sulfur high basicity and low oxygen potential is, therefore, essential. The effect of changing the temperature on desulfurizing index is uncertain. Thermodynamically lower temperature is bound to improve the desulfurization index but in practice high temperature is favored from the kinetic point of view. From the activity point of view it is easier to remove sulfur from pig iron than from steel because the presence of the usual impurities in pig iron raises the activity coefficient of sulfur.

The same impurities also increase the activity coefficient of [C] and hence help to maintain a lower equilibrium [%C] in the melt and thereby help desulfurization. The partitioning of sulfur between slag and metal in laboratory experiments and in basic steel making practice can be described as

$$\begin{aligned}\log K'_{(4\cdot25)} = &-1\cdot4\,N'_{\mathrm{Ca}^{2+}} - 1\cdot9N'_{\mathrm{Fe}^{2+}} - 2\cdot0N'_{\mathrm{Mn}^{2+}} \\ &- 3\cdot5N'_{\mathrm{Mg}^{2+}} + 1\cdot63\,N'_{\mathrm{Na}^{+}} \qquad \ldots(4.29)\end{aligned}$$

(the values are corrected for 1600°C)

This leads to the relative desulfurizing ability of basic cations as follows:

Na	:	Ca	:	Fe	:	Mn	:	Mg
1070	:	1.00	:	0.325	:	0.25	:	0.0075

The soda ash cannot be used in practical furnaces because of its extremely corrosive nature, hence the operator is left with control predominantly of lime content to effect desulfurization. Although (FeO) may help desulfurization as a basic oxide, the attendant increase in oxidizing potential of the slag more than offsets this and desulfurization becomes more effective only with a decrease in (FeO) content of the slag.

Since steel making is carried out under oxidizing conditions (except in electric arc furnace reducing slag period) the efficiency of desulfurization is very low. The value of the index ranges from 50 to 100 under blast furnace conditions and drops to -10 under basic steel making conditions.

It is generally possible to eliminate about 50% of the sulfur present in the charge in slow processes like open hearth. Therefore, to produce finished steel with less than 0.05%S a maximum of about 0.08-0.10%S can be tolerated in the molten pig iron charge in such slow processes. In faster processes like Bessemer, LD, or OBM not much of desulfurization of metal can be achieved because of lack of time for sulfur transfer from metal to slag to take place. The sulfur content of the molten metal charge therefore must itself be below 0.05%. Since the sulfur specifications for finished steel are getting more and more stringent, the sulfur content of charge metal should preferably be below 0.04%. In an electric arc furnace under reducing conditions sulfur can however be eliminated efficiently but the proportion of steel so produced in electric furnaces is still very small.

It is therefore, desirable and economical to remove sulfur, as far as possible, in the blast furnace itself, where the conditions are more conducive for desulfurization. There is however a limit to which this can be achieved in the blast furnace. It has been possible and economical to remove sulfur from pig iron outside the blast furnace and before it enters the steel making furnace. This is known as external desulfurization; of hot metal. It is being extensively practiced, particularly when the sulfur specifications are of the order of 0.005, i.e., 50 ppm or much lower in final steel. This is required for flat products or for smooth operation of continuous casting of thin slabs.

In actual practice, highly basic slags are employed for effective dephosphorization and such slags are quite capable of removing some part of sulfur (may be around 20%) from the metal as well. This is however not adequate to achieve the desired lower sulfur specifications of the present day. Hence every attempt is made to keep the sulfur content of the charge in steel making itself below the specification level, thereby eliminating any necessity of desulfurization in steel making for meeting the specifications.

In spite of this, if it becomes necessary to eliminate sulfur to meet the lower specifications, then the ladle furnace process or secondary steel making has to be adopted, using synthetic slag, with high sulfur capacities, for its effective removal.

Turkdogan calls [%S] × [%0]/[%S] as the sulfide capacity of slags and correlates it with the acid oxide contents like silica and phosphorus pentoxide of slags [12] to indicate the level of equilibrium achieved with respect to sulfur partition between slag and metal.

It has been found that the sulfide capacities of slags are better represented as a function of acid oxide contents rather than the basic oxide contents of the slags. Refer here again to the ionic theory of slags given in Section 3.3.2. It shows good dependence on [% SiO_2 + 0.84 x% P_2O_6]. This is shown in Figure 4.8. as the concentration of SiO_2 and P_3O_5 in the slag increases the sulphide capacity decreases and vice versa.

In brief sulfur transfer from metal to slag is favored by high basicity, high temperature and very low oxidizing potential (reducing conditions, i.e., very low FeO content which is usually less than 1%) of the slags. Although with the high basicity slags that are employed in basic steel making, some desulfurization is feasible along with dephosphorization; this route is not resorted to consciously for sulfur removal. The desulfurization that may take place in this way is treated as only a byproduct.

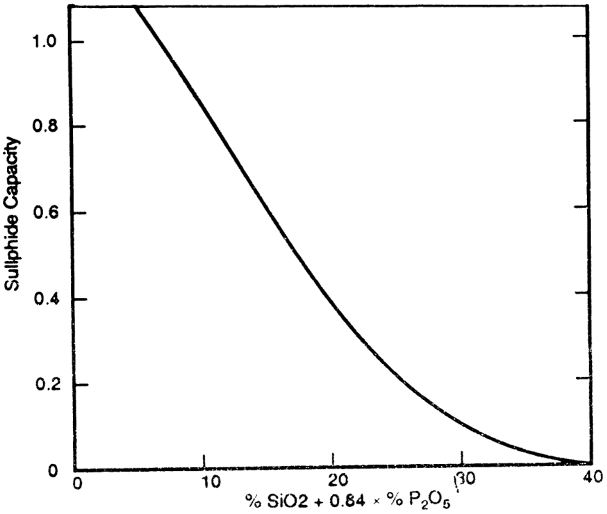


FIGURE 4.8. The variation of sulphide capacity of slags with its composition

4.3. Kinetics of Refining

The partitioning of solute elements during refining of molten pig iron takes place at the slag-metal and gas-metal interfaces. The structure of the interface is broadly described as:

$$\text{Interface} = \begin{array}{c} \textit{Bulk phase No. 1} \\ \hline \textit{Stagnant boundary layer of phase 1} \\ \hline \textit{Stagnant boundary layer of phase 2} \\ \hline \textit{Bulk phase No. 2} \end{array}$$

The mechanism of reactions in such a heterogeneous system is generally described in the form of the following three steps:

(*i*) Diffusion of reactants across the stagnant boundary layer(s) to the interface.

(*ii*) Chemical reaction at the interface (including sub-steps like adsorption of reactants and desorption of products).

(*iii*) Diffusion of products across the corresponding stagnant boundary layer(s) into the bulk phase.

The overall rate of the reaction is controlled by the slowest step out of the above three steps, i.e., either by the mass transport rates across the stagnant boundary layers or by the chemical reaction at the interface. In either case the individual atoms participating in the overall reaction must acquire enough energy to overcome the energy barrier before the reaction can occur. Such processes are called activated processes and their rates are related to the temperature of the reaction by the Arrhenius relationship:

$$\text{Rate} = \text{A} \cdot \exp\left(-\frac{Q}{RT}\right)$$

where A and Q do not vary sensibly with temperature and R is the gas content. In brief A is the number of particles correctly oriented for reaction and A is the activation energy to overcome the barrier resisting the reaction. The detailed interpretation of A and Q can be seen in the absolute reaction rate theory. It is not yet possible to obtain the reaction rates in steel making using the theory of absolute reaction rates. The Arrhenius relationship can be experimentally established in the form of useful plots of log rate *vs* 1/T. These are linear relationships with slope equal to – Q/R and from which the activation energy Q of the process can be calculated. The slope (i.e., activation energy) changes with change in the rate controlling mechanism. In the case of diffusion-controlled processes, since the diffusion coefficient also varies with temperature, the observed value is the apparent activation energy of the process. Apparent activation energies up to 10-15 kcal/g mole is a strong indication of metal boundary diffusion being the slowest step, and in the range of 10-60 kcal/g mole is a strong indication of slag boundary diffusion being the slowest step in the overall steel making reactions. The apparent activation energy for chemically controlled processes is also in the range of 10-60 kcal/g mole. Both the diffusion as well as chemically controlled processes improve their rates with rise in temperature. The two, however, can be differentiated by stirring the phases. A process under diffusion control is accelerated by stirring that phase, whereas a chemically controlled process remains unaffected. Any higher value of activation energy than 60 kcal/g mole is definitely indicative of chemical control.

In general the chemical reaction step (No. *ii*) at steel making temperature is bound to be very fast and the refining reactions are usually controlled by one of the two mass transport processes. The rate of mass transport across a stagnant boundary layer is given by

$$J = D_i \frac{C_r - C_i}{x} A \text{ g/sec.}$$

where D_i is the diffusion coefficient, C_r and C_i are the concentrations in the bulk and at the interface respectively, x is the thickness of the stagnant boundary layer and A is the interfacial area. The D_i values for various solute elements at steel making temperature in slags as well as in metal phases are known. The average value of x for molten iron is 0.003 cm in still condition. The value of x in a slag phase varies considerably with slag composition; it increases with increase in viscosity. For the usual thin (wet) refining slags in steel making a value of the order of 0.01-0.03 cm may be assumed for x. Increased turbulence caused by stirring, decreases the value of x and thereby hastens mass transport resulting in improved rates of diffusion controlled processes.

The following account illustrates the interfaces where the refining reactions predominantly take place in commercial processes:

1. Slag metal interface controlled — Open hearth and electric without oxygen lancing, Kaldo, etc.

2. Gas metal interface controlled — Acid Bessemer.
3. Mixed controlled — Open hearth and electric with oxygen lancing, Thomas, LD, OLP, OBM, etc.

4.3.1. Reactions at Slag-Metal Interface

Except the oxidation of carbon the rest of the refining reactions take place at the slag-metal interface. In general these reactions are found to be under diffusion control, generally more so, across the stagnant boundary of slag rather than that of the metal phase. Vigorous stirring of the slag-metal system is known to accelerate these reactions. In practice vigorous stirring is caused in the Bessemer process by bottom blowing, in an open hearth and in an electric furnace by the carbon boil, in Kaldo process by rotation of the vessel and, in LD and Rotor processes by the supersonic oxygen jet. Stirring reduces the thickness of the stagnant boundary and increases the interfacial area. In addition every attempt is made to keep the slag as fluid as possible by either raising the temperature of the furnace and/or by adding suitable fluxes to hasten mass transport in the slag phase.

Kinetically the removal of P, Si, Mn, and S from iron by slag, therefore, does not pose any serious problem of being slow in practice provided the thermodynamic and physical requirements of the slag are met. The observation is amply substantiated in practice by the fact that P, S, and Mn are very prone to reversion to metal phase if slag composition changes in unfavorable direction even for a short while.

The situation about the removal of phosphorus has changed considerably because of very fast refining in BOFs. It has become imperative to achieve at the turndown of a BOF the correct %P along with desired % C. Hence the kinetics of P removal has acquired a lot of significance to effect economy. The highly basic, thin and oxidizing slag should be prepared as early as possible during refining to improve upon the kinetics of dephosphorization. Equally important is to prevent P-reversion by not allowing the temperature to rise unnecessarily, since lower temperature favors dephosphorization.

4.3.2. Carbon Reaction

Carbon forms the single largest impurity in pig iron to be eliminated during refining. It is characteristically different from the rest of the impurities in that the oxide product (Equation 4.2) is a gas at steel making temperatures.

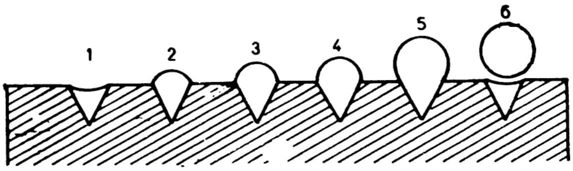
The reaction of oxidation of carbon practically does not take place at the slag-metal interface because of the difficulty in nucleating gas bubbles there. The reaction takes place as the gas metal interface since it eliminates the necessity of nucleation of gas bubble. The decarburization rates in pneumatic processes of steel making in general are, therefore, higher than those in hearth processes. The decarburization rate $\left(\frac{dC}{dt}\right)$ is controlled by the rates of diffusion of either carbon or oxygen to the gas metal interface. In the bottom-blown Bessemer converters a continuous series of *fresh* air bubbles pass through the melt. It results in increasing the total area of gas-metal interface enormously. The turbulence caused by the bottom blowing also favors mass transport. The net result is that decarburization rates up to 10% per hour are observed in practice. In the basic oxygen furnace processes the formation of slag-gas-metal emulsion helps in providing an ever-fresh gas-metal interface and thereby decarburization rates closer to those in the Bessemer process are obtained. The carbon reaction is, however, very slow in a hearth process wherein no gaseous refining medium is used. In order to understand this differential behavior let us first consider the physics of formation of a carbon monoxide bubble.

It is known that the pressure inside a bubble of radius *r* in molten metal of surface tension a is given by,

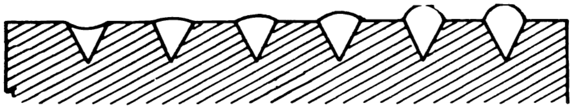
$$p_b = p_o + \frac{2\sigma}{r}$$

where p_o is the static pressure due to the heads of atmosphere, slag and metal compressing the bubble and, the $2\sigma/r$ is the pressure term due to surface tension opposing the growth of the bubble. If a carbon monoxide bubble is to form homogeneously in the melt, for the average molecular size of the bubble nucleus of *r* equal to nearly 5-8 A°, the pressure within the bubble should have a value of 10^4-10^5 atoms. The equilibrium partial pressure of CO gas at the gas-metal interface is given by Equation (4.12). Under no circumstances in practice can the bath be supersaturated with carbon and oxygen contents to the extent that the equilibrium p_{CO} will have a value as high as 10^4-10^5 atoms at steel making temperatures. In other words the possibility of homogeneous nucleation of carbon monoxide bubble in the melt during refining should be entirely ruled out.

(*a*) *Active sites.* The bubble attains hemispherical shape before the partial pressure of CO attains its equilibrium value, hence the bubbles are



Active sites



Inactive sites

FIGURE 4.9. The effect of crevice size on the uncleation and growth of CO bubbles at the metal-refractory interface

formed and, being mechanically unstable, are separated.

(*b*) *Inactive sites.* The CO partial pressure inside the bubble reaches its equilibrium value before the hemispherical shape is attained, hence no separation is possible.

Alternatively a carbon monoxide bubble can nucleate at the slag-metal or refractory-metal interfaces. The extent of nucleation at the liquid slag-metal boundary in negligibly smaller than that at the solid refractory-metal interface. The molten metal would always be resting on certain areas of porous solid refractory. The high surface tension of molten iron prevents it from wetting these pores and the gas-metal interfaces such as these provide the necessary interface at which the carbon reacts with oxygen and the gaseous product is desorbed in the pore space. The carbon reaction can proceed here until the p_{CO} inside the bubble is less than the equilibrium p_{CO}. With the progress of the reaction the bubble p_{CO} increases and thereby the bubble tends to grow into a hemispherical shape, i.e., the bubble radius equals the pore radius. Any further rise in bubble p_{CO} after the attainment of hemispherical shape results in increasing the bubble radius and hence the pressure term $2\sigma/r$ decreases. This amounts to a decrease in the internal pressure of the bubble, and as a result, the carbon reaction moves further to the right. Beyond the hemispherical shape, mechanically the bubble, as it grows, becomes more and more unstable until finally it detaches itself from the refractory wall and rises through the melt. While the bubble ascends the p_o term decreases and hence the bubble size increases. This also helps in pushing the carbon reaction to the right. The same process repeats itself at the pores and the carbon oxidation continues provided the other requirements are met. If the pore size is smaller than a certain critical value, the bubble p_{CO} may equal equilibrium p_{CO} value before the bubble attains a hemispherical shape. No further growth of the bubble is possible and the pore does not actively participate in the nucleation of bubble. For a fixed value of the product of the activities of carbon and oxygen there is thus a minimum unwetted pore size at which nucleation of CO bubble is possible. The unwetted pores of radius greater than the minimum size act as active sites, and the smaller pores are inactive sites for carbon reaction as shown in Figure 4.8. For the typical condition in open-hearth it can be quantitatively shown that unwetted pores greater than about 0.01 mm in radius are the most probable nucleation sites for carbon reaction to take place.

In an interesting experiment, molten iron was held under slag layer in a nicely glazed silica crucible. The value of the product of the activities of carbon and oxygen in the melt could be built up manyfold beyond the equilibrium value. The moment the inner glazed surface was scratched with a silica rod the carbon-oxygen reaction started vigorously and the bath soon attained the equilibrium carbon and oxygen contents. A similar gas evolution at some of the discontinuities in the wall surface can be observed in carbonated beverages. When water boils the bubbles form at such discontinuities in the walls of the container. The bubbles in all these cases form one after the other at these preferential sites.

Even if an inert gas is bubbled through the iron melt, the carbon-oxygen reaction takes place at the gas-metal interface and CO is unloaded in the inert gas bubble. Any floating solid pieces (e.g., lime, ore, etc.) at the slag-metal interface also provide a situation very similar to the refractory-metal interface. The CO does nucleate here but does not get a chance to grow as is possible when it forms at the bottom. These bubbles generally cause foaming of slags and should be avoided as it acts as an insulating blanket and restricts heat transfer to the metal in hearth processes, but the same has been considered as beneficial during oxygen blowing in electric arc furnaces.

4.3.3. Mechanism of Oxygen Transport and Kinetics of Carbon-Oxygen Reaction

In the pneumatic processes, oxygen is readily available at the gas-metal interface. The rate of reaction is affected by the diffusion of carbon to the interface. The stirring of the bath caused in these processes helps to hasten

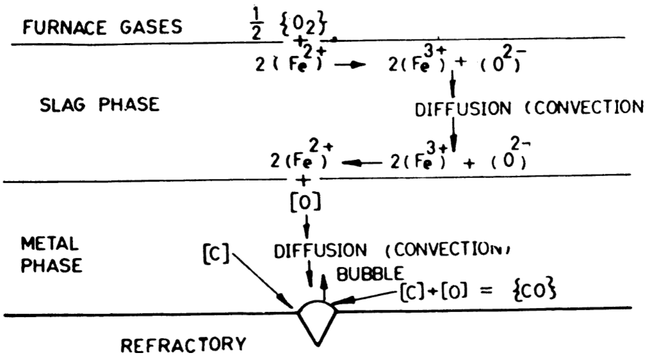


FIGURE 4.10. The mode of oxygen transfer from gas to metal via slag in a hearth processes so as to form CO bubbles at the furnace bottom.

carbon transport and thus decarburization rates up to 10% per hour are obtainable in these processes.

In hearth processes oxygen from the furnace atmosphere has to diffuse across the slag and the metal layers to reach the pore-metal interface. The physical solution of oxygen in slag is negligible. It must dissolve in slag in ionic form. The mechanism of oxygen transport from the gas phase is shown in Figure 4.10.

At the gas-slag interface oxygen dissolves as:

$$1/2\{O_2\} + 2e \rightarrow (O^{2-}) \qquad \ldots(4.32)$$

and the iron in the slag gets oxidized as:

$$2(Fe^{+}) \rightarrow 2(Fe^{3+}) + 2e^{-} \qquad \ldots(4.33)$$

so that the overall reaction is

$$2(Fe^{2+}) + \tfrac{1}{2}(O_2) = 2(Fe^{3+}) + (O^{2-}) \qquad \ldots(4.34)$$

Due to thermal diffusion these migrate from gas-slag to slag-metal interface and a reverse reaction as:

$$2(Fe3+) + (O2-) = 2(Fe2+) + [O] \qquad \ldots(4.35)$$

takes place. The ferrous ions again migrate to the gas-slag interface and complete the cycle, which repeats itself. The oxygen dissolved in metal diffuses to the pore-metal interface and combines with carbon.

The process of diffusion of oxygen can be hastened by eliminating the diffusion path across the slag layer by adding lumps of ferric oxide which, being heavier than slag but lighter than metal, settle at the slag-metal interface. It supplies oxygen as

$$\text{<Fe2O3>} \rightarrow 2(\text{FeO}) + [\text{O}] \qquad \ldots(4.36)$$

which is an endothermic reaction. It is known as oreing of the slag. In the absence of oreing when oxygen solely comes from furnace atmosphere, under the usual conditions of open-hearth practice, a maximum dC/dt of 0.12-0.18% C/hr is attainable. If oreing is carried out to its maximum extent the rate can increase up to 0.6% C/hr. The limit is due to the inadequate heat transfer across the slag layer since the overall reaction

$$\frac{2}{3}\,\text{<}Fe_2O_3\text{>} + 2[\text{C}] \rightarrow 4/3[\text{Fe}] + 2\{\text{CO}\}$$

$$\Delta H = +65 \text{ kcal} \qquad \ldots(4.37)$$

is endothermic in nature. If low-pressure oxygen is sprayed on the slag surface the above endothermic reaction is replaced by

$$[\text{C}] + [\text{O}] = \{\text{CO}\}$$

$$\Delta H = -89 \text{ kcal}$$

with a net heat gain and hence dC/dt up to 3% C/hr have been obtained. Higher dC/dt values can be obtained in hearth processes by oxygen lancing of the bath during the refining period. This is the principle of the modified hearth process.

4.3.4. Importance of Decarburization Reaction

In steel making processes, suitable slag of the right chemical and physical characteristics is always the aim, to ensure smooth and efficient oxidation of impurities like Si, Mn, and P. Out of these three in a normal basic process of steel making, Si and Mn join the slag readily without any effort on the part of the steelmaker except control of flux additions, which is very easy. Dephosphorization however poses various problems and requires time for its required and effective control. Stirring the slag and metal system does wonders but stirring in hearth processes has to depend on carbon boil alone, which is not fast enough to cause required stirring. On the contrary when oxygen is blown, i.e., in pneumatic refining, stirring caused by the blowing action does help dephosphorization. Pneumatic refining also results in decarburization and therefore the rate of decarburization is controlled in steel making to control the overall process refining. The relative rates of decarburization and dephosphorization are adjusted in all steel making processes to arrive at the desired end conditions when refining is over, only through control of decarburization rate, provided proper slag is available.

4.4. Thermal Principles of Refining

The invention of the Bessemer process was more of a chance rather than the result of a logical approach based on sound understanding of the underlying scientific principles. The open-hearth furnace was designed, relatively more scientifically, for melting steel scrap. It developed into a full-scale steel making process without understanding the principles of steel making. A cumbersome construction and a thermally most inefficient design like open hearth would never have survived as a steel making unit for over one hundred years, but for the ignorance of the principles underlying steel making. The vast amount of heat available from the exothermic reactions of steel making became apparent from the study of thermal balance of steel making processes, and this finally laid the basic foundations of oxygen steel making. This is discussed below.

4.4.1. Thermal Efficiency of Steel Making Processes

During steel making the impurities are oxidized to their respective oxides as in Equations (4.1) to (4.5) all of which are exothermic reactions. Similarly the dissolution of oxides in each other in the formation of a slag is also exothermic processes. The actual amount of heat liberated during refining will have to be found out in relation to the compositions of the bath and the slag, the initial temperature of the metal, the nature and temperature of the oxidizing agent used and so on. A precise value of this is not easy to obtain. As a standard guide the temperature rise attainable by oxidation of 0.01% of each of the element dissolved in liquid iron at 1400°C by oxygen at 25°C is calculated assuming that no heat is lost to the surroundings and such data are shown in below.

Element	Fe	Cr	Mn	P	Si	C	**c**
Temp. rise °C	0.5	1.5	0.7	2.3	3.1	1.2	3.8
Product	(FeO)	(Cr_2O_3)	(MnO)	(P_2O_5)	(SiO_2)	{CO}	$\{CO_2\}$

An additional rise of 0.5°C will be due to the dissolution of silica in slag and rise of 1.5°C due to the dissolution of phosphorus pentoxide in slag. The dominant exothermic nature of P, C, and Si oxidation reactions is very evident. The major portion of heat from carbon oxidation comes out only if the end product is carbon dioxide. For the typical pig iron composition under Indian conditions, i.e., iron containing:

C, 4.0%, Si, 1.0–1.2%, P, 0.35% and Mn, 8.0%

and assuming the usual combinations of SiO_2 and P_2O_5 as basic silicates and phosphates the rise in temperature is expected to be in the range of 900-1000°C. It can also be shown that an almost equal additional rise can be obtained by utilizing the heat generated by burning CO to CO_2 alone. During steel making this exothermic heat is required to accomplish the following:

1. To heat molten iron from its usual temperature of charging (1250-1350°C) to the tapping temperature level of about 1600°C or even higher.
2. To melt as much of the home scrap or reduce iron ore, if any, charged as coolants.
3. To heat any other cold charge materials (including gaseous).
4. To compensate for heat losses to the surroundings.

Out of the above four items, the items 1, 3, and 4 must be met from the exothermic heat to make the process autogenous. If any excess heat is still left over it can be used for item 2. Depending upon the amount of excess heat available the percentage share of item 2 shall vary. The extent to which the heat obtained from exothermic reactions is utilized in the conventional process of steel making is described in the following section.

4.4.2. Conventional Pneumatic Processes

In the conventional pneumatic processes (using air as an oxidizing agent) oxidation of impurities does produce enough heat to make process autogenous. The initial charge should be molten iron. A study of thermal balance of the Bessemer process reveals that as much as 25% of the total thermal energy goes to waste in the form of sensible heat of the exhaust gases. This is because 79% of the cold blast, by volume, being nitrogen, has to be unnecessarily heated from room temperature to the steel making temperature. As if this were not enough, a part of this unwanted material remains dissolved in the finished steel thereby making it strain brittle (not suitable for deep drawing purposes). In spite of the loss of so much heat, the correct tapping temperature can still be obtained if only hot metal of suitable composition (see Chapter 8), which is readily obtained from a blast furnace, is used. There is however no extra heat left in the process to take any significant amount of steel scrap or ore as coolants. The tapping temperature is just adequate for teeming, hence no alloying additions except the usual deoxidizers and recarburizers can be added in the ladle.

All the CO formed during refining burns at the mouth of the converter using oxygen from atmospheric air and thereby radiates all the heat to the surroundings. If this heat is taken into account as much as 60-70% of the theoretical exothermic heat is wasted in the Bessemer operation.

4.4.3. Conventional Open-Hearth Process

Cold charge in an open-hearth furnace needs to be melted-using external fuel. The use of hot metal in the charge minimizes the fuel requirements to the extent of heat required for melting and superheating the cold iron or steel scrap. But once the charge is molten what factors prevent it from becoming an autogenous process like the Bessemer process? The answer to this question has to be seen in the refining reactions. In the conventional open-hearth practice, the refining operation cannot be made autogenous for two reasons:

1. The open-hearth operation takes a much longer time for refining than Bessemer, hence the heat losses by radiation, conduction etc., to the surroundings are greater in an open hearth than in the Bessemer.
2. The oxidation reactions using solid iron oxide as an oxidizing agent like

$$<Fe_2O_3> + 3\,[X] = 3(XO) + 2\,[Fe] \qquad \ldots(4.38)$$

$$<Fe_2O_3> + [X] = (XO) + 2\,[FeO] \qquad \ldots(4.39)$$

(where X stands for any impurity in the bath) are both net endothermic processes. If the energy required to heat the cold oxide to the reaction temperature and that required to take solid lime in solution are taken into account, there is overall deficiency of heat even while silicon (which is used as the primary heat producer in an acid Bessemer process) and phosphorus (which is used as the heat producer in basic Bessemer process) are being oxidized. The deficiency is more pronounced in the case of carbon and manganese oxidation. The refining can therefore, continue only if heat is supplied to the bath from an external source. It means fuel supply is needed in an open-hearth process even during refining. The rate of refining is in fact conditioned by the rate of heat flow from the furnace gases to the metal bath across the insulating slag layer.

In a conventional open-hearth practice, if additional pre-heated air is supplied on the slag surface, the carbon monoxide formed during refining can be burned inside the furnace chamber and the heat thus released can be used in the process. In a well designed furnace as much as 70% of the total energy required during refining can be met by the CO to CO_2 oxidation reaction. The process however still requires external heating during refining.

Generally the same arguments can be made about the electric furnace processes as far as the refining by iron oxide addition is concerned.

4.4.4. Refining by Oxygen

The use of oxygen in place of iron oxide or air results in the following beneficial changes:

1. From thermodynamic point of view oxygen is the best of all the oxidizing agents since the impurity contents in iron, in equilibrium with oxygen, will be least as compared to those values when air or iron oxide is used instead. Dead soft steels can be easily produced. The fact was qualitatively realized by Bessemer himself, but since oxygen was not available on a large scale it could not even be tried experimentally.
2. The volume of waste gases in a Bessemer process would be reduced by 60-70%. The heat loss, as sensible heat, is also proportionately reduced by using oxygen in place of air.
3. The problem of nitrogen pick-up in the finished steel is very much minimized.
4. The net endothermic refining reactions are replaced by the net exothermic reactions if oxygen is used in place of iron oxide in hearth processes.
5. The sluggishness of carbon boil is eliminated if oxygen gas is used in place of iron oxide in hearth processes (difficulty in bubble nucleation is eliminated) thereby making the refining operation an autogenous one.

All these observations derived from the fundamental understanding of the steel making processes clearly established the superiority of refining iron by oxygen over that by air and/or iron oxide. It could not however be commercially adopted on wide scale because of the high cost of oxygen. Oxygen was used during steel making whenever economics permitted e.g., for the decarburization of bath in open hearth to obtain steels containing lower than 0.25%C. The introduction of the Linde-Frankle process in 1942 for the manufacture of tonnage oxygen solved the problem of oxygen required for steel making. Immediately after World War II, pilot plant trials were undertaken to refine iron by blowing oxygen. The persistent investigations finally led to the development of three different oxygen steel making processes, namely the LD, the Kaldo, and the Rotor. These were designed almost concurrently at three different places in the period 1947-1954 to exploit the above-mentioned advantages with a view to cater to the individual set of requirements at these places. Broadly the processes aim to obtain:

1. Maximum production rate with minimum of capital and running cost.
2. Better quality product, i.e., the least nitrogen and hydrogen contents in steel.
3. High thermal efficiency with maximum amount of coolant consumed at a minimum of refractory consumption. The overall process being an autogenous one.
4. Maximum maneuverability to produce a wide range of products from varied charge materials easily, particularly the production of soft steels.
5. Maximum cleanliness of the product.

The thermal efficiency was improved by burning part or all of the CO evolved during refining to CO_2 inside the vessel. It may contribute up to 25-35% of the theoretical heat input in the furnace besides the nearly 50-60% available by way of usual exothermic reactions. All the heat available by way of CO to CO_2 reaction could not be utilized because of the limitations imposed by the life of the available refractory materials of furnace construction. A certain limitation was also imposed by the necessity to conduct heat through the slag layer, which is always present above the metal bath in a steel making furnace. The more this heat is utilized inside the steel making furnace, the shorter the lining life.

The reasons that the use of pure oxygen as an oxidizing agent has made such an outstanding difference to the practice of steel making, going on for nearly 100 years, and making it now fully a autogenous one, should be apparent from the information above.

In addition to the new oxygen steel making processes, the use of oxygen modernized even the conventional processes of steel making and a maximum amount of oxygen was used in these processes to obtain the above-mentioned advantages. The conventional open-hearth designs had to be altered to adopt oxygen refining. In some cases the alterations had been drastic enough to call these as different processes of steel making, e.g., Ajay, Tandem Furnace, SIP, Twin-Hearth, etc. Similar drastic alterations of Bessemer led to the development of OBM process, Phoenix Lance process, etc. But with all these modifications they could not stand competition from BOF processes and ultimately, on economic grounds and due to inferior quality products, they were all replaced in due course by the BOF processes.

4.4.5. Preheating of Charge in Steel Making

The thermal efficiency of steel making processes did improve considerably after the adoption of oxygen refining, so much so that the process becomes fully autogenous if molten iron forms the charge in required proportion. In fact impurities like Si, Mn, C, P, etc., act as fuels in steel making during refining. The all-solid charge steel making however requires heating and melting, even if it is made autogenous afterwards by oxygen refining. The energy consumption for melting solid scrap can be decreased by preheating the scrap using the sensible heat in the hot gases generated during refining. This aspect of preheating has been fully exploited in making the open-hearth process fully autogenous by converting the age-old design of open hearth into the Twin-Hearth furnace design or the Tandem-furnace design, in which the refining and preheating go simultaneously in two different halves of the furnace, and is reversed periodically. Preheating of scrap using refining hot gases is also intelligently used in designing the Energy Optimized Furnace or EOF.

Where preheating is not possible, oxygen refining is begun without waiting until all the charge is molten. Oxygen refining and melting are concurrently carried out after melting of a small lot of charge, thereby the refining heat is utilized concurrently for melting as well. Once a small pool of metal is made, the process can continue autogenously, using oxygen.

If electricity is costly, oxygen-fuel burners have also been used for preheating solid charge in an otherwise electric arc furnace.

Thermal efficiency of steel making, particularly in an electric arc furnace, has been improved by shielding the arc using foamy slag during early melting-refining.

The use of freshly burned lime instead of limestone is recommended only to conserve furnace heat which otherwise would be used for its calcination.

Preheating of solid scrap in a steel making operation is, however, not a simple task. It requires fairly advanced engineering.

4.5. Deoxidation of Steel

Since steel making is carried out under oxidizing conditions, oxygen is bound to dissolve in iron melt. The solubility of oxygen in pure iron at 1600°C is 0.23% and rise to 0.48% at 1800°C. The oxygen content in iron varies inversely with the impurity contents in general, and with carbon content in particular. As refining progresses, the oxygen content in the melt increases, and at the end of refining a considerable amount of oxygen (0.05-0.10%) is left in liquid steel. If such steel is cast, since the solid solubility of oxygen in pure iron is only 0.003%, the excess oxygen is evolved in the form of gases leading to an unsound casting. A very carefully refined steel can thus be totally ruined unless the excess oxygen is eliminated before its casting. The removal of residual oxygen content of refined steel is known as deoxidation or killing of steel. It can be brought about in two different ways:

(1) **Diffusion deoxidation**—The dissolved oxygen content is reduced by inducing it to diffuse into the slag inside the steel making furnace itself or under vacuum separately outside. There is a limit to which this can be made use of effectively.

(2) **Precipitation deoxidation**—Elements having a higher affinity for oxygen than iron has for oxygen are added to the melt. A reaction of the same type as in Equation (4.7) leads to the formation of oxide product. This method is generally adopted as it is very effective in decreasing the oxygen content of steel. The choice of a deoxidizer depends on a number of factors.

4.5.1. Thermodynamics of Deoxidation

Thermodynamically the best deoxidizer should have the minimum %[O] in equilibrium with its own minimum content in steel. The equilibrium states for the common deoxidation reactions pertaining to steel making

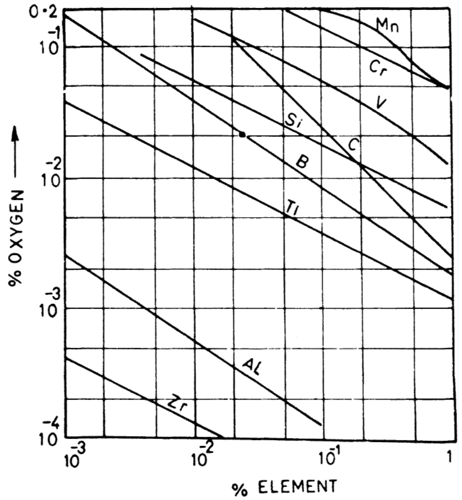


FIGURE 4.11. Comparison of deoxidizing powers of various elements at 1600°C. The products of deoxidation being respective oxides[13].

conditions have been established reasonably well. The equilibrium oxygen contents for various deoxidation reactions giving pure oxide product are shown in Figure 4.11. It indicates that aluminum is a very effective deoxidizer and is quite extensively used in practice. Silicon is also reasonably effective. Al, Si, Mn and C are commonly used as deoxidizers, since they are reasonably cheap. It may be noted that Mn, Si, and C are the same elements that were oxidized earlier as impurities from iron and the same elements are used later to remove excess oxygen from the refined steel. Occasionally elements like Zr, Ti, B, V, Nb, etc., may also be used but these are costlier than common deoxidizers.

Chemically the residual deoxidizer content in steel should not lead to any adverse effects on the properties of steel, i.e., these contents must be within the range of chemical specifications at the end of deoxidation.

4.5.2. Kinetics of Deoxidation

Kinetically the deoxidizer should be quick in action to obtain its high percentage utilization for deoxidation reaction. Such data are however largely unknown, and thermodynamic rather than kinetic considerations dictate the choice of a deoxidizer.

Besides, the physics of the process is also equally important. Oxide product of the deoxidation reaction should not remain as mechanically entrapped particles in steel. These entrapped particles are known as nonmetallic inclusions that impair the mechanical properties of steel. The mechanical properties of steel vary with the number, size, shape, distribution, and composition of these inclusions. Every attempt is therefore made to eliminate the product of deoxidation from the melt and thereby obtain clean steel. The term cleanliness refers to the relative freedom of steel from inclusions.

A gaseous product of deoxidation is, therefore, ideal. Carbon alone produces a gaseous product of deoxidation reaction. At atmospheric pressure it is not a very effective deoxidizer as can be seen in Figure 4.1. It is however very effective under reduced pressures. The vacuum treatment of steel can be adopted to obtain cleaner steels providing the economics otherwise permit. The physics of carbon monoxide bubble formation as discussed in Section (4.3.2) may be referred to again to understand deoxidation of steel by carbon.

The kinetics of the deoxidation reaction, per se, is not so much important as is the kinetics of elimination of product of deoxidation to produce relatively cleaner steels. This requires the understanding of the mechanism of precipitation deoxidation, hereinafter referred to only as deoxidation, since almost the entire deoxidation is invariably achieved through precipitation deoxidation alone.

4.5.3. Mechanism of Deoxidation

The deoxidation process is a very complex one. It can broadly be described in the form of four consecutive steps:

1. Dissolution and homogenization of the deoxidizer in the steel melt so as to make the deoxidation reaction move in the direction of oxide formation.
2. Formation of critical nuclei of the deoxidation product in a homogeneous medium since it involves formation of a new phase.
3. Progress of deoxidation resulting in growth of the reaction products.
4. Separation of the product of the deoxidation reaction by way of their flotation from the steel melt to improve cleanliness.

This in turn imposes certain requirements on the nature and quality of the deoxidizer used for obtaining cleaner steels for a given end use.

The deoxidizer should be in a form such that its dissolution in steel should pose no problems. Virgin elements like silicon, aluminum, titanium, etc., may not dissolve in steel readily because of the adherent oxide layer on their surfaces. But if these are used in the form of their ferro-alloys, dissolution poses no problem since iron therein acts as the carrier. Hence most deoxidizers are used in the form of their ferro-alloys.

The process of nucleation of deoxidation product becomes easier if an interface is available. Prior deoxidation by aluminum resulting in the formation of an alumina/steel interface is useful in this respect. Other inhomogeneities are also beneficial.

Growth of the reaction product is directly dependent on its nature. Liquid particles can coalesce more readily than solid particles. Deoxidation is, therefore, frequently carried out to obtain liquid product as far as possible. Deoxidation by silicon alone is quite effective as it produces a solid product (SiO_2). Deoxidation by manganese alone does produce a liquid product but it is not effective. When the two are used together the FeO-MnO slag produced earlier by manganese reaction readily takes SiO_2 in solution when silicon deoxidizes the bath. The resulting product is a slag of the type FeO-MnO-SiO_2 wherein the activity of silica as well as MnO are much lower than when these are used individually. This fact also increases their effectiveness in reducing the oxygen content of steel when used together in the correct proportion and added suitably. Manganese and silicon are used in the ratio of 7 : 1 to 4 : 1 to obtain a thin liquid slag as product. Fe-Mn is added first and then Fe-Si. With 0.5% [Mn] the equilibrium oxygen level at 1600°C is approximately 0.1% but if 0.1% [Si] is also present along with the oxygen, the level drops to 0.015%. This is shown in Figure 4.12.

Aluminum is a very effective deoxidizer since Al_2O_3 is a far more stable oxide than SiO_2, MnO, etc. But Al_2O_3 is solid even at steel making temperatures and hence cannot be used alone to deoxidize steel completely. It is generally used along with Mn and Si when alumina has a chance to join the thin liquid slag product of deoxidation. Boron, titanium, zirconium, etc., are also quite effective deoxidizers. The level of deoxidation obtainable by 8% Si can be achieved either by about 0.7% B, or 0.1% Ti, or 0.002% Al or 0.00003% Zr. There are occasions when deoxidation has to be carried out by Al alone. Then the problem of alumina particles floating in the bath has to be tackled and failing that, particularly in continuous casting, troubles may arise by the way alumina streaks in solidified steel and casting problems.

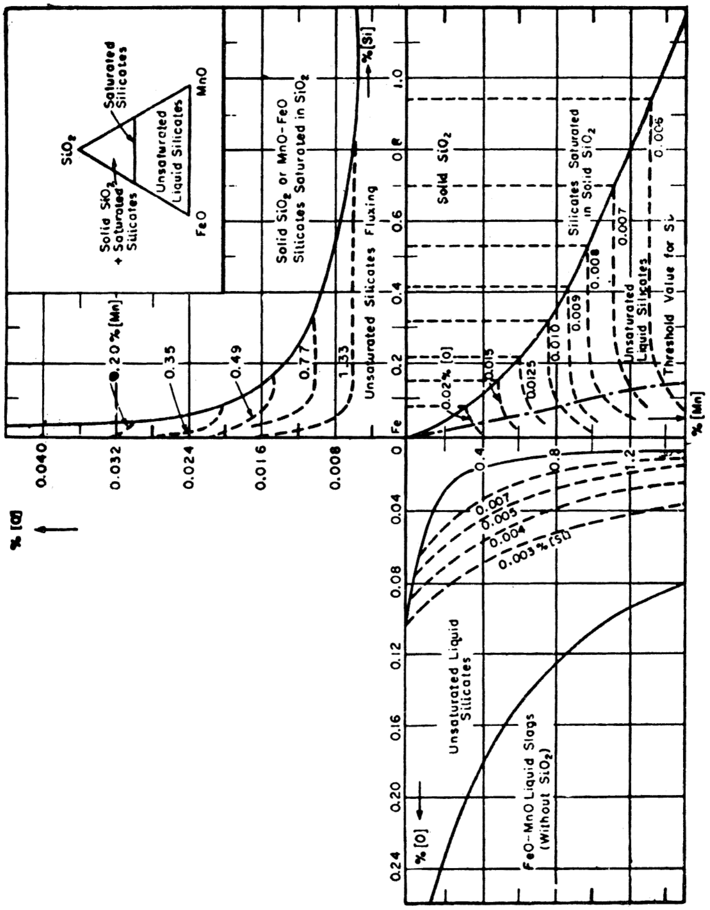


FIGURE 4.12. The equilibrium relationships of interest during deoxidation of steel by silicon and manganese in contact with solid and liquid slags at 1600°C (after Kalling[10]).

The use of deoxidizers other than carbon leads to the formation of liquid or a solid product in the form of a dispersed phase in steel melt. Being lighter than steel the oxide product can rise to the surface and can be eliminated at the top of the melt. In a quiet steel bath the rate of rise of oxide particles can be assessed from Stoke's Law. Using typical data it can be shown that generally particles smaller than 10^{-3} cm in radius shall not be able to rise to the surface in a usual ladle, before commencing teeming after tapping. On the other hand, particles with more than 10^{-2} cm radius shall almost completely be eliminated. For effective removal, therefore, the particles must coalesce into a bigger one.

In practice the liquid melt is never quiet and convection currents of appreciable strengths are always present. The inclusions, therefore, no longer move vertically in accordance with Stoke's law but are carried along the currents in the melt. It also prevents layering of conclusions as in the quiescent bath. At any time some inclusions are brought by these currents to the surface while some others are carried downwards into the bulk metal. A fraction of those brought to the surface get assimilated and thereby separated from the melt. The size of inclusions in the bulk grow because of collision and coalescence and when they get chance to come to the surface they may get eliminated. The larger is the size of product particles the larger is its chance to get eliminated by way of rising upwards and joining the slag layer at the surface of the melt.

Sometimes the reaction products are beneficial if these remains entrapped in a very finely dispersed form. Aluminum forms finely dispersed Al_2O_3 particles, which do not readily coagulate, and the surface of these particles act as the possible nucleation sites during solidification of steel. The enormously large number of these nucleation sites result in fine grain structure of the resulting steel. Zr is added to avoid segregation of sulphides in free cutting steels. Use of Zr and Ti fix up the nitrogen dissolved in steel as harmless nitrides which otherwise would lead to intergranular fracture.

The deoxidizers are added in the form of their ferro-alloys or virgin metals. Aluminum is added in the form of shots, small blocks, cubes, etc., and carbon as graphite or anthracite.

4.5.4. Deoxidation and Final Chemistry

The chemical reactions of the deoxidation process are the same as refining reactions i.e., the deoxidizing element combining with dissolved oxygen to give its oxide. The typical reactions are given below:

$$\underline{Si} + 2\underline{O} = (SiO_2) \qquad \ldots(4.40)$$

$$2\underline{Al} + 3\underline{O} = (Al_2O_3) \qquad \ldots(4.41)$$

representing the element by the general term X, the equilibrium shall be represented as

$$K = a_{(XO)}/(a_X * a_O)$$

It means that at the end of the deoxidation process, depending upon the leftover dissolved oxygen in the bath, there will be some leftover deoxidizing agent concentration also in the steel bath. This cannot be avoided.

This is the reason that, when liquid steel is deoxidized by silicon, the end chemistry of steel would have around 0.15-0.3% Si left in the bath. Similarly if manganese is used for deoxidation and since it is a weaker deoxidizing agent, more manganese, being cheaper, is used as deoxidizing agents for the production of common variety of steels. The steel specifications in this case will need to have silicon around 0.2-0.3% and manganese around 0.4-0.7% as unavoidable leftover impurities, with leftover oxygen levies of around 30-60 ppm in the final analysis. Even though these are classified as impurities, in general, and are removed to a very low level during refining, these cannot be finally removed below these levels; they are to be tolerated as part of the final chemistry, and therefore as final chemical specifications for such steels.

In fact, the silicon and manganese are removed far below to 0.1% or so in the refining process but the subsequent deoxidation by the same elements leads inevitably to their final chemistry at the above-mentioned high levels. For cheaper and common steel production there is no alternative. When the use of relatively more costly aluminum is permissible then only is it possible to contain the levels of both silicon and manganese to very low levels of the order of less of 0.02-0.10. Any extra content unnecessarily hardens the steel. This is the essential requirement when softer flat steel products of specific grades are produced.

When more effective deoxidation is required, aluminum is used as a more powerful deoxidizer to aim at around less than 5 ppm as leftover dissolved oxygen in steel. The equilibrium aluminum content in this steel shall be less than 0.1% or so, more probably around 0.04%.

Steel cannot be made with lower than the above values as specifications using Si/Al/Mn as deoxidizers. In case the specifications insist on very low oxygen and, very low silicon, manganese and aluminum, then the required chemistry can only be achieved by vacuum deoxidation with carbon alone. In this case the product of deoxidation is desorbed into the atmosphere, and the leftover deoxidizer in the form of carbon is also very low. This is the practice that is essential for producing very soft steels like the deep-drawing (DD) or extra deep-drawing (EDD) or the interstitial-free (IF) qualities. These grades are in great demand these days and this is the only technique that can produce them.

4.5.5. Deoxidation Practice

At the end of the refining steel bath can be deoxidized either inside the furnace or while being tapped in a ladle. In the former case it is called furnace deoxidation, and in the latter case it is called ladle deoxidation. A small part of the total deoxidation may be carried out in the ingot molds if fine-grained steel is to be produced, or if earlier deoxidation has not been adequate. The extent of deoxidation carried out at each of the possible stages is governed by the nature of the process and the end product required. The deoxidation operation decreases the oxidizing potential of the bath. If the oxidizing refining slag is present in contact with the metal, the deoxidation may result in reversion of the refining reactions in Equations (4.1) to (4.5). In an acid steel making process silica and manganese oxide, being stable oxides, are not prone to reversion, but in a basic process P_2O_5 from the slag, is very readily reduced back to the metal bath if the oxygen potential drops. In an acid process, therefore, deoxidation of the bath can be fully carried out inside the furnace even if the refining slag is in contact with the bath. In a basic process, however, the refining slag containing P_2O_5 should be removed as far as possible before deoxidizing the bath inside the furnace. In a basic process, the deoxidation can alternatively be carried out in the ladle. At times a majority of the refining slag is flushed-off in a basic process and the deoxidation may be carried out partly (minor extent) in the furnace and the major part in the ladle. Deoxidation inside the furnace is beneficial in obtaining a more clean steel because the products of deoxidation have more time and chance to rise to the surface of the bath.

Depending upon the extent of deoxidation three different types of steel ingots are teemed viz. the killed or fully deoxidized, the semi-killed or balanced and, the rimming steels. If the carbon content is less than 0.15% the liquid steel contains enough of dissolved oxygen (see Figure 4.1), and a rimming variety can be produced. The rimming steels are not at all deoxidized because dissolved oxygen is essential for the rimming action during solidification of steel in ingot molds. If carbon is more than 0.16%, the level of dissolved oxygen rapidly falls with increasing carbon content. The low carbon steels may be partially deoxidized to produce semi-killed or balanced variety of steel ingots. The others may be fully deoxidized or killed steel ingots e.g., high carbon steels.

Alloy steels are invariably fully-killed steel to obtain maximum recovery of alloying additions.

4.6. Plain Carbon Steel Production

In the production of plain carbon steels the bath is always decarburized during refining, to a level lower than the specifications. The deficit is met by adding recarburizer in the furnace, if possible, or alternatively in the ladle while tapping. A very minor part is also made up by the carbon contained in the deoxidizers if these are added. An attempt should always be made to finish the heat such that a minimum amount of recarburizer is needed. The deoxidizers and recarburizers are usually added together in the ladle while tapping in basic steel making processes. Excessive decarburization of refined steel in the ladle is not recommended because it tends to make steel dirty due to the entrapment of the ash resulting therefrom. It also tends to increase the nitrogen content of steel.

4.7. Alloy Steel Production

Basically, alloy steel can be produced in three different ways.

1. Entire amount of allowing specification is accomplished by the addition of virgin metal and/or ferro-alloys to the molten, refined and properly deoxidized (as required) steel bath.
2. Clean and classified alloy steel scrap is carefully melted to avoid any significant loss of allowing elements on melting. The precise specification level is finally obtained by adding a very small amount of requisite elements or ferro-alloys. It is essentially a process wherein what goes in must come out.
3. A part of the total alloying content is derived from the scrap charge (being the cheap source). The remainder is met by virgin metal or ferro-alloy additions to meet the specifications.

Alloying elements can be classified into four different categories from the point of view of steel making viz.:

1. Volatile elements like Zd, Cd, Pb, etc. These cannot be added to steel easily. The recovery is very poor.
2. Non-oxidizable elements like Cu, Ni, Mo, Co, As, desulfurizing, etc. These can be added at any stage during steel making, even inside the furnace. The recovery is nearly 100 percent.

3. Partially oxidizable elements like P, Mn, Cr, etc. These can be added to a steel bath which has been partially deoxidized and with proper care, good alloy recovery can be obtained.
4. Completely oxidizable elements like Al, Si, Ti, Zr, V, etc. These elements should be added only after adequate deoxidation of the bath, or else the alloy recovery goes down because part of the alloy is lost in deoxidation itself of the bath. Alloying elements of category (3) and (4) are best added under vacuum during secondary steel making.

If partially or fully oxidizable alloying elements are to be added inside the steel making furnace, it must be possible to carry out deoxidation of the bath inside the furnace. These can be added in any acid process and basic electric process of steel making under reducing conditions. Any amount of alloying can be accomplished in these processes, hence these, particularly the acid open hearth, acid and basic electric arc processes, are most suited for medium and high alloy steel production.

Partially and fully oxidizable alloying elements can be added to the ladle during tapping to produce low alloy steels.

4.8. Tapping Temperature

The tapping temperature of any refined steel, that is, the amount of superheat in the steel melt is a fairly critical value and depends on several factors such as:

1. Chemical composition of the melt.
2. Amount of additions like recarburizer, deoxidizers and alloying additions to be made in the ladle.
3. Holding time of the melt in the ladle before teeming starts.
4. Teeming practice as such.

Basically steel melt in the ladle should be at the required teeming temperature, which is at least 30-50°C above its liquids temperature even while teeming the last ingot. If this temperature is found for the given chemical composition from the relevant equilibrium diagram, the tapping temperature can be computed by regression analysis, taking into account the temperature loss at each stage in the reverse order until the point of tapping. This final temperature is shown in Figure 4.13.

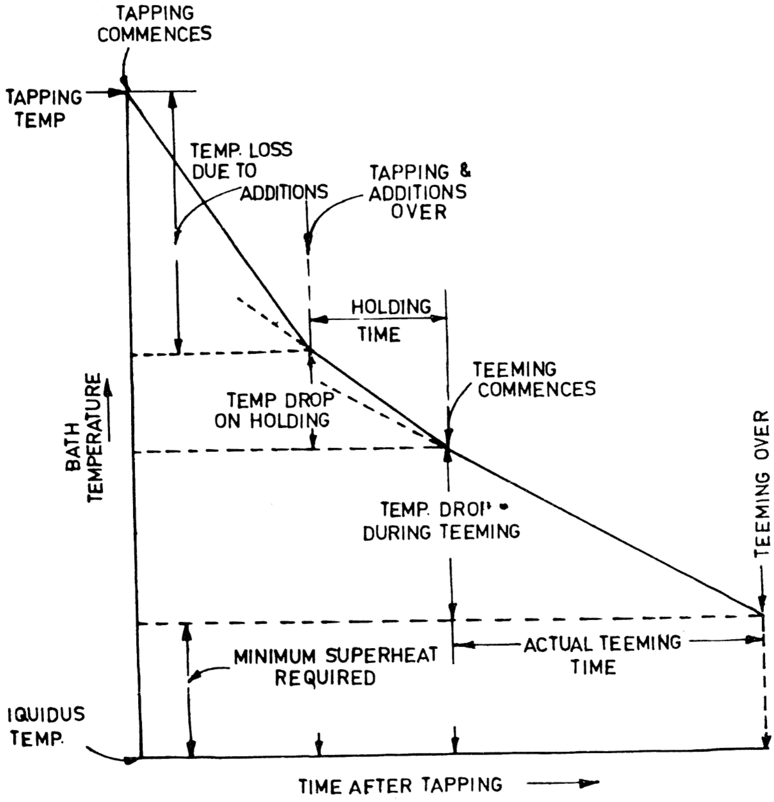


FIGURE 4.13. Temperature drop in liquid steel after tapping until it solidifies as ingot.

Figure 4.15. The loss in super heat due to the addition of cold alloying additions to molten iron at 1600°C. The bath is assumed to lose no heat to the surroundings. (Reproduced by permission from AMIE.)

This temperature drop during the entire teeming period is worked out by computing the temperature drop during the holding period, during the entire teeming period as such, and due to the total additions in the ladle. The way the tapping temperature is computed is illustrated in Figure 4.13. On the higher side the tapping temperature is conditioned by the process of refining and the refractories used in the construction of the furnace.

The tapping and teeming temperature ranges for plain carbon steels are shown in Figure 4.14 as an illustration.

Similarly the temperature loss due to cold alloying additions for the usual variety of additions is shown in Figure 4.15 under idealized adiabatic

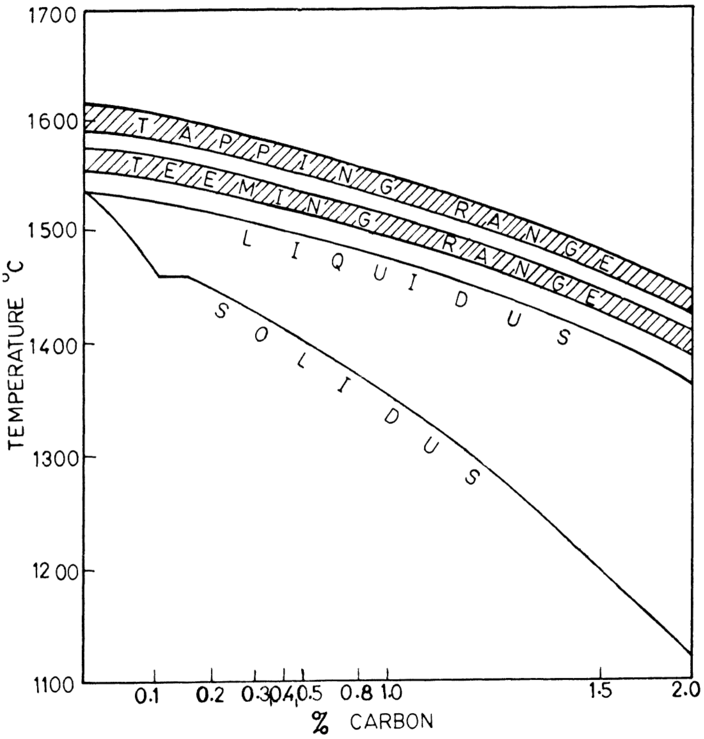


FIGURE 4.14. Tapping temperature ranges for plain carbon steels as a function of their carbon contents.

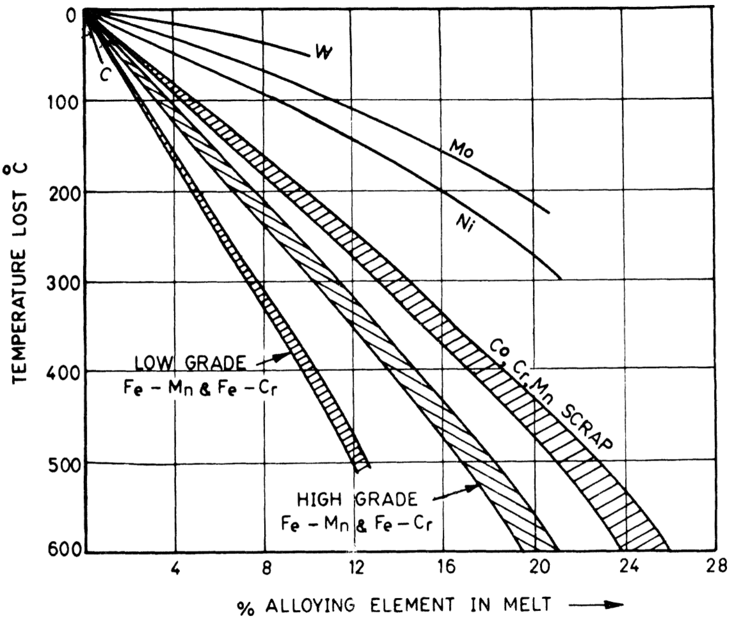


FIGURE 4.15. The loss in super heat due to the addition of cold alloying additions to molten iron at 1600°C. The bath is assumed to lose no heat to the surroundings. (Reproduced by permission from AMIE.)

conditions. The actual loss shall be more than this because of additional heat loss to the surroundings. It can therefore be understood that only a small amount of alloying is thus permissible in the ladle or else the bath will be unnecessarily quenched. In other words, only low alloy steels can be satisfactorily made by ladle additions alone. If extra superheat is not available even for small additions, preheated additions can be considered as a way out. However, even if extra superheat is available or even if preheated ferro-alloys are added, excessive alloying in the ladle is not recommended, as the cleanliness of steel may be adversely affected.

4.9. Production of Ingots by Casting

From the furnace liquid refined steel is tapped in ladles called teeming ladles. Additions, if any, are made while tapping steel, to adjust the chemical composition as close to the specifications as possible. Liquid steel is held in the teeming ladle for nearly twenty minutes for homogenization of chemical composition and temperature to be achieved. It is then ready for casting. The process of pouring liquid steel in a mold is known as teeming. In the conventional method of casting, liquid steel is teemed into ingot molds either directly, via tundish or indirectly as shown in Figure 4.16 (*a*). The indirect pouring is also called bottom pouring. Alternatively continuous casting is adopted in which steel is teemed in a small bottomless mold continuously and the solidified ingot is pulled out from the bottom at a matching rate to obtain a continuous length of the same cross- section. It is then cut into desired lengths. This is shown in Figure 4.15 (*b*).

In either case the red-hot ingots are stored in soaking pits for their subsequent rolling to produce the semis. This practice results in the generation of a certain amount of scrap to be recycled into the steel making furnace. The ingots therefore yield less than one hundred per cent ingot weight as the salable semis.

In the conventional method of ingot production, the scrap generation may go to 20-45% of the ingot weight, depending upon the product, the mold design, teeming practice, teeming design, and so on. All care should be taken to obtain the ingot as sound as possible and thereby to generate less scrap during rolling. Figure 4.13 indicates the principle of designing the teeming practice to achieve better soundness in ingots. Whatever the practice, the teeming temperature at the end of the teeming must still be adequate. The teeming design depends on the method of pouring, the ladle

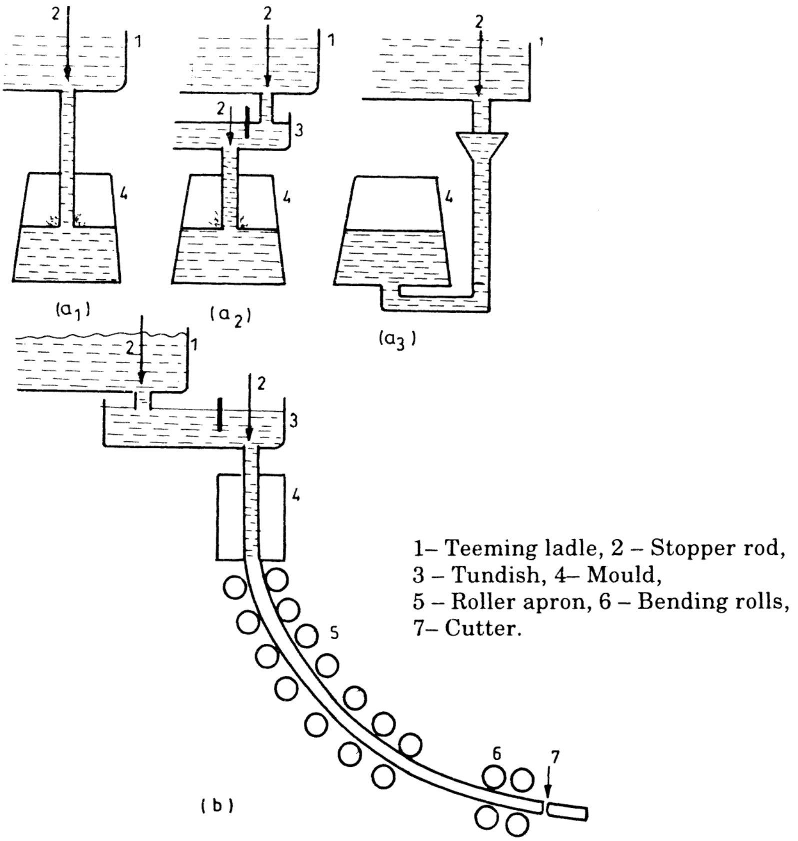


FIGURE 4.16. Casting of liquid steel into regular shapes. In the conventional casting method, ingots are cast in three different ways shown in (a) a_1 shows the direct teeming, a_2 shows the tundish teeming and the a_3 shows the indirect or bottom teeming. Continuous casting is shown in (b).

nozzle size, number of ingots simultaneously teemed, ingot size, ladle size, ingot type, and so on.

In conventional ingot casting the ingots are stripped off the molds in the stripper yard where the molds are reconditioned for their reuse. In conventional casting the ingot is rolled in primary mills to blooms, slabs, and in secondary mills to billets which are further rolled into the semis. In continuous casting the product itself is blooms, slabs, billets, etc., and hence part of the rolling is eliminated in producing the semis. This obviously results in less scrap generation, and the ingot yield increases. The ingot yield in this case may go up to 87-93% as compared to much less in conventional casting practice. The continuous casting is therefore almost universally adopted now.

Especially in large integrated steel plants, the old ingot mold practice has almost completely been replaced by the continuous casting technology. Much of the rolling has also consequently been eliminated by casting thin slabs around 50 mm thick, which are then rolled into thin sheets. The latest technology has now been developed to directly cast thin sheets by pouring liquid steel between two rolls and pulling the solidified sheet from below, in near net shape casting technology. This way fine-grained-structure-sheets have been produced. These are given only either the finishing pass or are cold rolled into the desired thinness.

In brief, the tapping temperature, the alloy additions, teeming equipment and teeming practice have to be decided in relation to each other to ultimately produce good quality acceptable semis or sound ingots as a product.

4.10. Control of Refining

From a given mix of charge of scrap and iron, a given steel is produced by refining the charge to the desired chemical specifications. The refining process must be properly controlled to achieve this within the shortest possible time and with minimum consumption of reagents and additions. In general, in slow processes like hearth processes there is little time available to assess the state of the refining during refining. However in fast processes like BOF this control part has always been a challenge. In general, knowing the charge composition, the final slag required is assessed and whatever is required to make such a slag is added from outside at appropriate times to make the desired slag. Such a slag is supposed to oxidize Si and Mn without any problem as is the case. The oxidation of phosphorus takes time and requires sufficient stirring of the slag and the metal bath. The carbon reaction is provided by Mother Nature in steel making to achieve the conditions desired for dephosphorization. Depending upon the kinetics of dephosphorization, the required carbon oxidation reaction has to be carried out. In other words that much equivalent carbon will have to be removed to obtain the desired dephosphorization. There is always more than enough carbon to be removed during refining. Hence the decarburization rate has to be adjusted to match the required dephosphorization rate.

In fact in steel making, from experience, it is the decarburization rate that is primarily controlled to control the overall process of refining, the other conditions being otherwise adjusted by slag adjustments to synchronize their removal with carbon as established from known data.

Carbon reaction is therefore the crux of refining. In all processes decarburization rates as affected by various parameters of steel making have been, and are being, established to whatever extent possible to establish the kinetics of decarburization. This information is fully utilized in the development of strategies for controlling the refining process.

CHAPTER 5

RAW MATERIALS

The chief raw materials for steel making are as follows:

- Sources of Metallic Iron
- Oxidizing Agents
- Fluxes
- Sources of Heat
- Deoxidizers and Alloying Additions
- Furnace Refractories

5.1. Sources of Metallic Iron/Metallics

The steel making process requires iron units or metallics, as it is referred to in the parlance of iron and steel making, in other words the basic raw material of steel making. In modern steel making it is met by either liquid pig iron, DRI/HBI, and the recirculated scrap of iron and steel. The following Table 5.1 shows the global demand and supply of metallics.

The primary source of metallics is in the form of molten iron, essentially from a blast furnace. It is also known as hot metal on the shop floor. Sometimes solid pig iron, known as pigs, is also used, but only occasionally for some specific reason. Molten iron is bound to improve the efficiency of steel making as it can make the process of steel making an autogenous

Table 5.1
Global demand and supply of metallics (Mt)

Year	*2000*	*2005*	*2010*
Total metallics needed	1010	1195	1370
Hot metal	480	505	540
DRI/HBI availability	43	60	75
Scrap availability	400	500	570
Shortfall in metallics	87	130	185

one. In an integrated steel plant, therefore, the blast furnace and steel making furnace are located together. The DRI/HBI or some form of sponge iron is an alternative source of metallics now available for steel making; it requires melting facilities to convert it into steel. The recently developed smelting-reduction processes also produce hot metal, as alternative sources of metallics. Hence either a blast furnace or some SR process, essentially coupled with oxygen steel making, is responsible for a major amount of steel production. A significant proportion of steel nowadays is produced via the DRI/HBI and electric furnace route. The share of different metallics in steel production is shown in Table 5.2.

It clearly shows the decreasing dominance of the blast furnace, and increasing adoption of alternative methods for the production of metallics to finally produce steel, as alternatives becoming more economical.

5.1.1. Pig Iron

The composition of pig iron produced in an integrated steel plant is always controlled, as otherwise permissible, to suit the steel making process. In other words, the choice of the steel making process is made, besides considering other factors, in relation to the quality of iron that would be produced. If the iron contains less than 0.05% phosphorus, acid steel making process could be adopted. This is known as acid, or hematite, or Swedish iron. This type of iron can be produced only at some exceptional places. By and large, the iron produced all over the world contains more than 0.05% phosphorus, generally around 0.2-0.4%. Iron containing more than 0.05% P is called basic iron, and basic steel making process is needed to refine this iron. A special category of basic iron that contains more than 1.5% P is known as Thomas iron, and special basic steel making processes have been developed to refine this iron.

Table 5.2
Share of different metallics are a percentage of the total used for steel making [14]

Iron source	*1995*	*2000*	*2005*	*2010*
Hot metal	58.9	57.2	55.0	52.5
DRI/HBI	3.4	5.2	6.6	8.1
Scrap (recirculating)	12.8	9.6	8.80	8.0
Scrap (collected)	24.9	28.0	29.5	31.4

5.1.2. Steel Scrap

The secondary source of metallic iron is steel scrap. The steel plant itself generates scrap in the form of spillage, sheared ends, rejected material, etc. to the tune of about 30-45% of its ingot production. This is variously called new scrap, home scrap, home returns, plant returns, circulating scrap, etc., which must be recycled into the steel making.

Although it is feasible to use entirely hot metal charge in a steel making furnace, it is absolutely necessary and positively economical to use the home scrap as part of the charge in an integrated steel plant. In an integrated steel plant, the steel making process(s) is chosen to fully reutilize the circulating scrap. The actual percentage of home scrap varies with the nature of product and the possible control of the process. Adoption of continuous casting in place of conventional casting may decrease the circulating scrap percentage by about 5-10%.

The average recirculating scrap under Indian conditions in integrated steel plants is as follows:

Table 5.3
Average recirculating scrap load

Product	*Scrap (% of ingots)*
Rails	-25
Plates	-50
Wheel, tire, axles, etc.	35-45
Strip	15-30
Merchant products	15-20
Forgings	30-45

Besides circulating scrap, scrap is also generated at the customer's end while making finished products. Its percentage varies considerably with the nature of the product and may be around 10 to 20% of the equivalent ingot weight. This is returned to steel works either assorted or in classified forms. The capital scrap arising from obsolete articles is also returned to steel making units. Its share is as high as 15% of the equivalent ingot weight in developed countries.

A certain fraction of the total scrap is used by foundries and for assorted reuses. On the whole it can be said that about 50% of each hot metal and return scrap goes in the steel furnaces to produce the total ingot production. In integrated steel plants where blast furnaces and steel making furnace are located together, the proportion of hot metal in the charge is generally more than 50% (60-70%), whereas the steel furnaces located near the centers of scrap generation use entirely cold charges—mainly scrap and a small fraction as pigs or cast iron machinings.

Commercial pneumatic steel making processes must have hot metal in the charge. If such steel making is situated away from a blast furnace, melting furnaces like cupola, electric furnace, rotary oil-fired furnace, etc., are required to produce hot metal by melting pig and scrap. In special cases the pneumatic steel making furnace itself has successfully been used as a melting unit prior to its refining e.g., Stokes converter, but it has not proved to be useful in commercial practice.

5.1.3. Sponge Iron or DRI

Wherever the metallurgical coal of the right quality is not available, or if the reserves are inadequate, or if other cheaper coals of a non-coking variety are available, alternative routes of ironmaking have been developed. Some of these use non-coking coals, some others use natural gas ,and so on, to reduce iron ore in solid state to produce nearly reduced iron ore. These are variously called sponge, DRI, briquetted iron (HBI), and so on. The material contains the gangue impurities of the ore and carbon as impurities. Steel can be produced by melting this sponge/DRI/HBI and refining mainly the carbon in an electric arc furnace. The gangue oxide impurities join the slag. The economics of these processes is quite precarious. It was not favorable everywhere in the 1960s and 1970s, hence not many installations came up during that time. But the situation changed and steel plants adopting the route described in Figure 1.1 (c) have been set up at many places in the world including India, and are giving good commercial results. There are several plants that produce only the sponge/DRI/HBI and sell them as raw material for substituting scrap in steel plants employing only electric steel making facilities.

Sponge iron is very porous and hence is very susceptible to oxidation in steel making. Due care must therefore be taken while using this material for steel making. Some special techniques have been developed for its effective use without serious oxidation in steel making.

Many applications of steel, in particular tin cans for food and medicines, involve plating of steel sheets by copper, tin, etc. When this is scrapped, it contains these as impurities. No known process of steel making can remove these impurities during steel making. Repetitive use and scrapping tend to build up the level of these impurities in steel. This is a very serious problem in advanced countries where the cycle time for such scrap may be only a few days. By their nature these impurities are called residuals. These are, in general, harmful to properties of steel, and generally copper above 0.05-0.1% and tin above 0.05%, and are not acceptable in steels as residuals. Steel produced from scrap may contain these residuals beyond safe levels unless specifically controlled by suitable pre-treatment of such scrap. The sponge iron is free from such residuals because it is made from virgin raw materials. It is thus best used as a substitute for scrap from the point of view of final residual contents.

5.1.4. Scrap Proportion in Charge

The proportion of scrap in the charge varies from process to process depending upon the available excess heat or input energy. The scrap consumption in such processes is shown in Table 5.2.

Table 5.2
The proportion of scrap consumption in various steel making processes

1.	Conventional processes	Bessemer	Up to 8%
		Open hearth	75%
		Electric	100%
2.	Oxygen steel making processes	LD (with	25%
		modification)	45%
		Kaldo	45%
		Rotor	45%
3.	Modified conventional processes	Bessemer	12%
		Open hearth n/a	
		Electric n/a	

The open-hearth and the electric furnace processes of steel making were developed chiefly for remelting the available scrap. Until the advent of these processes, there was no way to melt steel scrap because of the limitations of the furnace being capable of reaching the steel making temperature, i.e., of the order of 1550-1600°C. Steel scrap was then plentiful at low cost.

The open-hearth and the electric furnaces accepted even 100 % steel scrap charge for its conversion to steel. Although it is easier to deal with all scrap charge in electric furnaces, it is rather difficult to deal with all steel scrap charge in an open-hearth furnace because of its inherent oxidizing conditions. The practice was soon developed to take part of the charge as solid pigs, and later on the molten metal itself. For open-hearth the proportion was mainly determined by the availability of steel scrap and its economics. A 40-50% scrap in the charge is considered good.

For electric furnace it is economical to use 80% or more of steel scrap in the charge. Nowadays it is more a mixture of steel scrap and DRI in various proportions that is adjusted to make it more economical.

The LD process does not take more scrap because of its thermal restraints. However 25-30% is considered good. In fact, a steel plant using only LD process finds it difficult to consume all the circulating scrap in an integrated plant. Some other process has to be adopted along with it to balance this leftover scrap proportion. This will be clear from the following example:

The circulating scrap load in an integrated steel plant averages 35-45% of the ingot-producing capacity. It has been decreased by nearly 10% by adoption of 100% continuous casting in place of ingot mold casting under otherwise similar conditions. The Rourkela Steel Plant was originally designed to produce 75% of its ingot steel by LD process and the remaining 25% by the open-hearth process. This was so even though LD was preferable it could take only 15% of the circulating scrap in its charge, and the leftover nearly 25% had to be dealt with by the open hearth. This combination was necessary. This pattern has, however, changed in the later expansion with additional LD shop. The modern LWS process has been developed specially to take 45% scrap load as a modified version of the LD process.

5.1.5. Scrap Preparation

Usually the circulating and process scrap is returned to the steel furnace without any deleterious contamination. The capital scrap, in particular, may be contaminated with non-ferrous metals like tin, copper, nickel etc. Since it is not possible to oxidize these during refining, they remain as residuals in steel. As scrap gets recycled again and again, these residuals build up. The properties of steel are adversely affected by the presence of these residuals. The specifications of steel do recommend their maximum allowable limits. With modern living the circulation frequency of scrap is increasing, hence the problem of residuals is becoming serious. Detinning of scrap is carried out to remove tin from scrap. The price of the scrap therefore goes up. Copper, however, cannot be removed easily. A proper scrap blend is necessary to minimize this problem during steel making.

Scrap is classified according to its physical and chemical composition. In physical classification, large pieces need to be cut to suitable sizes to be accommodated in the charging boxes and subsequently charged in the furnace and accommodated inside the furnace easily without causing any inconvenience. Light scrap may be compressed into block-like bundles in bailing presses. The scrap particularly from the consumer durables, these days, is shredded and is freed of non-ferrous materials.

Chemical classification, if carried out, leads to segregation of scrap in terms of alloying contents, which can be suitably and better used in steel making, preferably for making the same type of alloy steel; such scrap is a cheap source of that alloying element.

5.2. Oxidizing Agents

The oxidizing agents used for steel making are iron oxide, air, and oxygen gas.

Iron oxide is used in the form of lumpy hematite ore and mill scale. Mill scale is the oxide of iron produced during hot fabrication of steel and is readily available in an integrated steel plant. The usual analyses of both of these are as follows:

	% Fe	*%S*	*%Volatiles*	*Others*
Iron ore	60-69	0.03	0.5	Gangue oxides (should be low in silica)
Mill scale	71	0.10	4.5	Minor impurity oxides

Both of these contain nearly 25% of oxygen by weight. The ores are generally low in sulfur and high in gangue oxide, whereas the mill scale is a nearly pure form of oxide but it may contain a high proportion of sulfur. Both are used in practice in suitable proportion. The use of iron oxide as an oxidizing agent results in improving the yield of the process, but it needs thermal energy to dissociate itself [see Equation (3.36)] and make oxygen available for refining reactions.

Atmospheric oxygen in the form of air is used in the Bessemer process, but nitrogen present in air gets dissolved in steel and makes it strain brittle. With the adoption of the Linde-Frankle process for the production of tonnage oxygen (purity more than 99.5% oxygen) at cheap rate, it has now become a useful oxidizing agent in the BOF processes and also during refining in hearth processes. The purity of oxygen should at least be 99.5%. The nitrogen content of refined steel increases with decreasing purity of oxygen.

5.3. Fluxes

A flux is a substance added during smelting and refining to bring down the softening point of the gangue materials, to reduce the viscosity of slag, and to decrease the activity of some component(s) to make it stable in the slag phase. Lime/limestone is generally added during steel making to make the slag basic enough to retain phosphorus and sulfur. Fluorspar and bauxite are added to decrease the viscosity of refining slags. The typical analyses of these are as follows:

The silica content of limestone decreases its value since the available base [%CaO – (SiO_2 × Basicity)] decreases with increasing silica content. The presence of MgO in limestone is also detrimental to its quality because MgO

Table 5.3
Typical analysis of fluxes used in steel making

	%CaO	*%S*	*%MgO*	*%SiO_2*	*LOI*
Lime	90-95	0.1-0.2	2-3	1.5	< 3-5
Limestone	50	0.01	1	1	≅ 40-45
Calcined dolomite	55	0.1	34-38	3-4	< 3.5
Ganister	0.5	-	0.1	94	≅ 5

(LOI Means loss on ignition)

is not as effective as CaO in retaining phosphorus and sulfur in slag. MgO and SiO_2 contents of the stone must be as low as possible. During steel making, limestone gets calcined, and CaO thus freed acts as a flux. It needs heat, hence its use should be made with due care. Generally calcined lime, i.e., CaO rather than limestone, is used as a flux if furnace heat is to be conserved.

The analysis of bauxite is:

54-56% Al_2O_3, 11-14% Fe_2O_3, 1-2% SiO_2, 1-2% TiO_2 and that of fluorspar is

75-85%CaF_2, 10% max. SiO_2, 0.8% max S.

5.4. Sources of Heat

Pneumatic processes like the Bessemer and the BOFs are autogenous since enough heat is generated during refining by the exothermic oxidation of impurities. These processes need hot metal, preferably of a suitable composition, as a charge to run the process. The amount of heat generated is always more than necessary so that scrap and/or iron ore is required to keep the bath temperature within required limits. These processes cannot be run with cold charges alone. As against this, hearth processes like the open hearth and the electric arc provide a source of heat to melt the solid charges to maintain steel making temperatures inside the furnace. These take up to 100% cold charges.

The sources of heat are:

Chemical: (*i*) Solid —pulverized fuel.
(*ii*) Liquid —oils, tar, etc.
(*iii*) Gas —producer, water, coke oven, natural and blast furnace gases.

For economic reasons the chemical fuels should be burned with excess air and thus the furnace atmosphere would always be oxidizing.

Electrical: (*i*) Induction heating.
(*ii*) Resistance heating.
(*iii*) Arc heating.

The electric furnaces have no atmosphere of their own, hence oxidizing as well as reducing conditions can be maintained in the furnaces with the help of suitable slags.

5.5. Deoxidizers and Alloying Additions

Elements like Al, Si, Mn, etc., are added primarily as common deoxidizers. Elements like Zr, B, Ti, etc., are added for deoxidation in special cases, and elements like Cr, izi, Mo, Ni, V, Nb, etc., are added generally as alloying additions. Carbon is added to recarburize steel or as a deoxidizer under vacuum. Excess additions of Al, Si, Mn, Ti, etc., above what is consumed for deoxidation, can also remain in the melt as alloying additions. Common ferro-alloy compositions are shown in Table 5.4.

Silicon. It is used as a primary deoxidizing agent in a furnace where a reducing slag is to be made. It is also used as a common deoxidizer in the furnace or in the ladle when killed or semi-killed steels are being made. This is a very effective common deoxidizer and is used in the form of ferro-silicon. Silicon is also added as alloying element for better strength, hardenability and electrical properties. A typical use is for making transformer grade steels.

Manganese. It is a weaker deoxidizer than silicon. As an alloying element it gives strength and toughness. It is added as ferromanganese of various grades. A typical use is for making Hadfield and structural steels. Where silicon and manganese are added together it may be added as a mixture of individual ferro-alloys or as silico-manganese.

Aluminum. It is a very effective deoxidizer and is used in the form of stars, rods, pellets, wires, powder, etc. Chemically it is used as ferro-aluminum or aluminum-silicon compounds with 90-97% purity. It is an alloying addition in heat resistant steels, e.g., Kanthal.

Nickel. It does not act as a deoxidizer. It is added as an alloying addition in stainless steels. It can be added any time during the heat. Carbonyl nickel is preferred to electrode nickel for its low adsorbed hydrogen content.

Chromium. It is used as an alloying addition for stainless and heat resistant steels. It is available in the form of ferro-chrome of several grades. It can act as a deoxidizer but is costly to be used for this purpose. It increases hardness, strength, yield point, and elasticity of steels.

Titanium. It is a strong deoxidizer and neutralizes the effect of nitrogen on steel by bonding it into stable insoluble nitride compounds. Titanium steels are useful for aircraft industry for their lightness coupled with strength. Ferro-titanium contains 23-25% Ti, 0.15%C and Si and, Al in certain proportion to the titanium content.

Table 5.4
Analysis of some ferro-alloys of standard grades

Ferro-alloy Type	*Grade*	*% Composition*				
		C	*Si*	*Mn*	*Cr*	*Fe*
Ferro-silicon	From 50–75% regular grades, even 85–90% grade is also available	1–2	50–90 as per grade	—	—	—
Ferro-managanese	Standard HC	7	—	78–82	—	
	Low carbon	0.07 max. 0.10 max. 0.50 max.	6.5	85–90	—	10–15
Silico manganese	FM	0.08 max.	28–32	56–61	—	—
	standard	1	22	66		5
Ferro-chrome	High carbon	7 max.	3 max.	—	58–65	—
	Low carbon	0.025 max 0.05 max. 0.75 max.	2 max.	—	67–73	—
	Low carbon 65/5	0.50 max.	5 max.	—	64–68	—
	Simplex low carbon	0.01 to 0.02	2 max.	—	68–71	—

Vanadium. It is a strong deoxidizer. It increases the strength, plasticity, and resistance to attrition and impact. It is usually added in structural, tool, and spring steels. It is added as ferro-vanadium. A typical composition is V-35% minimum, 0.75-1.00%C, 2-3.5% Si and 1-2% Al.

Tungsten. It is a strong deoxidizer. As an alloying addition it increases hardness, strength, and elasticity of steel. It is used in tool steels, high-speed steels, cemented carbide alloys, etc. It is added as ferro-tungsten. A typical composition is izi—70%, C—02—0.4%.

Molybdenum. It is only an alloying addition to improve the mechanical properties. Its presence ensures uniform microcrystalline structure, augments hardenability, and eliminates post-temper brittleness. It is used for shafts, gears, rolls, etc. Like Ni it can also be added at any time during refining. A typical composition is 55% Mo and 0.1-0.2% C.

Zirconium. It is used as a deoxidizer. It decreases the deleterious effects of nitrogen and sulfur in steel. It is added in the form of zirconium-ferrosilicon having minimum 40% Zr. It has a ratio Si/Zr of not over 0.55 and Al/Ze equals to 0.20.

Boron. It is used as a deoxidizer. As an alloying element it enhances the mechanical properties and hardenability of steel. It is added to steel in almost negligible amounts (0.0025-0.0030%) in the form of ferroboron and ferroboral. The ferroboron must have a minimum of 5% B and maximum of 3% Si, 5% Al and 0.55% C.

Niobium. The addition of niobium to stainless and heat-resisting steels enhances their plasticity and corrosion resistance. It improves welding properties of structural steels and has a stabilizing influence on stainless steels. It is added in the form of ferro-niobiumtantalum with a minimum of 50% (Nb + Ta), 0.1-0.2% C, 11-12% Si, 7% Al and 7% Ti.

Cobalt. It is used as an alloying addition for magnetic steels, and like Ni or Mo, it can also be added at any time during the heat. It is added in elemental form.

Carbon. Coke, graphite, and anthracite are used as recarburizers and their analyses are shown in Table 5.5.

Coke is also used to prepare reducing slags in an electric arc furnace.

Table 5.5
Analyses of recarburizers in percent.

	Ash	*S*	*Moisture*
Coke	8-11	1-2	5-10
Graphite	1	0.1-0.2	0-1
Anthracite	5	5.0-1.0	2-3

Besides the above elements, many others such as Cu, Sn, N, S, Ta, Pb, etc., are also added in some special varieties of steels for their beneficial effects on the specific properties of steels.

5.6. Furnace Refractories

A steel making furnace is lined with suitable refractory materials. The lining is eroded during steel making, hence the material of the lining is also required as a recurring consumable raw material. The lining is often repaired to maintain it in proper shape and state. The lining is made either by laying bricks (or blocks) or by shaping the required contour in situ using a refractory mix. The bricks are generally made in the following forms:

1. Chemically bonded, unfired, and reinforced.
2. Fired at usual or at higher temperatures.
3. Directly bonded (high strength).
4. Made from presintered (or prefused) and crushed mix.
5. Melted, cast, and cut to required shapes.

The materials used for lining acid furnaces and acid roofs of basic furnaces are essentially silica-sand, ganister, etc. Since the bulk steel production comes from basic process, dolomite and dolomite enriched with magnesite are extensively used as lining materials for basic furnaces. Chrome-magnesite was common on old basic open-hearth and basic electric furnaces. Magnesite is increasingly being used in basic furnaces, particularly if oxygen lancing of the bath is adopted. Similarly magnesite-chrome is being adopted in place of chrome-magnesite for its better performance.

Fire bricks for insulation and chromite as neutral bricks are also used in steel making furnaces.

All the basic refractory materials for lining BOFs essentially belong to the CaO-MgO system, and that the % (CaO + MgO) together should not be less than 90%. The typical compositions are shown in Table 5.6.

Freshly mixed, warm tarred dolomite or magnesite is used for making furnace bottom in situ in hearth furnaces. The hearths are hot repaired during the compaign using granular refractory material of the same type as that used earlier for preparing the bottom. Hearth furnaces should, therefore, be provided with storage facilities for granular refractory material for hot repairs.

Table 5.6
Analysis of basic refractory materials

Constituents → *Materials ↓*	*CaO*	*MgO*	*Other oxides*	*Loss on ignition*
Raw dolomite	30	20	1.5	46
Burnt dolomite	55	36	7	2
Burnt magnesite	3	90	7	—
Synthetic burnt magnesite	0.8	98	1	—
Dolomite enriched with magnesite	16	up to 80	3	—

5.7. Storage Facilities

A steel making shop may have an individual, a few, or a set of several furnaces, depending upon the output required. The storage facilities are generally common for all the furnaces in the shop and are located at suitable places. The materials need to be transported to the furnace, as and when required, with the help of an efficient transport system. The space provided for their storage is generally proportional to their volume-wise use. Bins, bunkers, etc., are provided to store the solid raw materials. The calcined lime and dolomite should be stored in closed bins to prevent moisture from changing their chemical composition. Scrap is generally brought to an adjacent yard, sorted out into various grades as per standard specifications for use, and supplied from the yard.

5.7.1. Mixer (inactive)

The hot metal from blast furnace needs to be stored in a vessel called a mixer because the blast furnace tap and steel making need not necessarily be synchronized with respect to its time and amount. The mixer acts as a buffer to store whatever metal comes from the blast furnace, and to supply metal to the steel making furnace as and when required. In doing so, it also evens out the hot metal composition, which varies from tap to tap and delivers it at a predetermined temperature. A mixer, doing only the above job, is called an inactive mixer. It is a fireclay-lined cylindrical vessel with a small fuel burner fitted in to compensate for the heat losses and to maintain the metal at the required temperature during its storage. One such mixer is shown in Figure 5.1. It is mounted on rollers and is capable of rotation

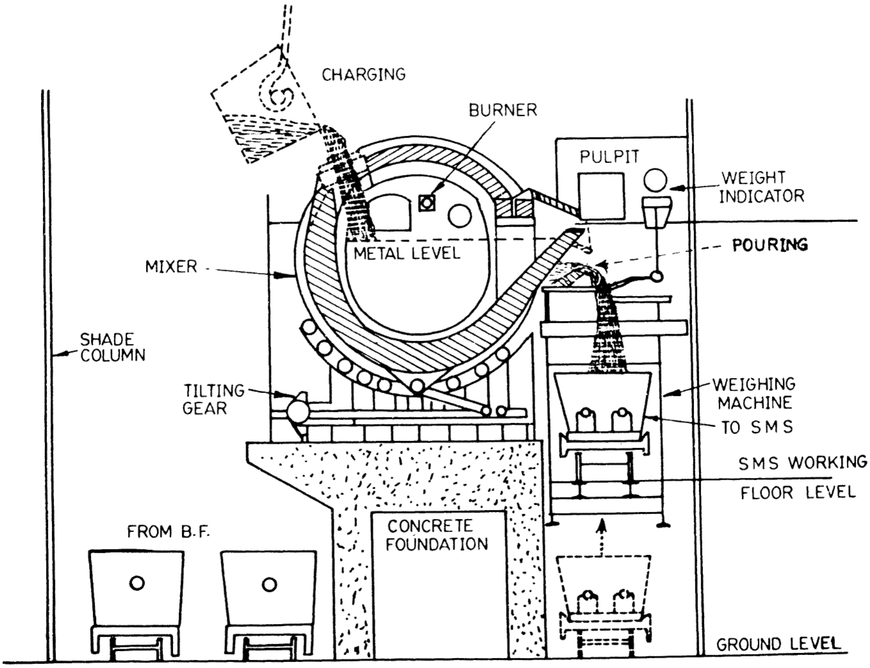


FIGURE 5.1. Inactive mixer.

around its horizontal axis. A hot metal car carries the metal from the mixer to the steel making furnace. The hot metal car is weighed with an empty ladle and with a full ladle to obtain metal weight by difference.

The use of a mixer adds to the cost of steel making by increased handling charges, extra establishment cost, and compensation of heat losses during storage. The temperature of hot metal held in an inactive mixer varies with the composition of pig iron and the process adopted for its conversion to steel. For example, TISCO maintains it at around 1300°C. The temperature is, however, maintained around 1300-1350°C at the Rourkela steel plant where LD process is used.

Blast furnace slag should not flow in along with the metal in the mixer, or it may find its way into the steel making furnace where flux consumption may rise and cause attendant difficulties because of its acidic nature.

Capacity of an inactive mixer varies considerably but it may be equivalent to 50-70% of the hot metal throughput per day. This means that a 1000-1400 t mixer is required for a million metric ton steel plant. The mixer capacities in Indian plants are shown in Table 5.7.

Table 5.7
Details of mixer capacities used in Indian steel plants

Name of plant	*Mixer units*	*Mixer capacity metric ton*	*Steel making furnaces*
Bhilai (BSP)	1	1300	3 × 500 t Twin hearth
	1	1300	3 × (250 + 250) t twin hearth
Rourkela (RSP)	2	1100 each	3 × 50 t LD and 4 × 80 t BOH (2 × 150 t LD)
	1	1100	2 × 70 t LD
Durgapur (DSP)	1	800	3 × 100 t LD and 1 × 200 BOH
Tatal Steel (TISCO)	1	1300	2 × 130 t LD
	2	800 each	2 × 130 t LD, 1 × 80 EOF and 2 × 200 t BOH
Bokaro (BSL)	2	1300 each	5 × 100/130 t and 2 × 300 t LD
Vizag (VSP)	-	-	2 × 150 t LD (+ 1 × 150 t)

It is advisable to use two mixers instead of one since one can be repaired while other is still being used. It takes several weeks to wreck old lining, reline, and commission a mixer again. In using two, one mixer is being filled while the other is being emptied out so that each one gets a sufficient chance to homogenize the composition and temperature at a predetermined level. When only one is used, the minimum level to which it is allowed to empty out is fixed. It is usually 20% of the capacity. An inactive mixer does not always remain full to its capacity; a 50-70% full mixer is considered adequate.

5.8. Pretreatment of Hot Metal

It has been shown that, except the electric arc furnace, the other steel making processes generally tolerate a maximum of 0.05% sulfur in pig iron to produce finally steel containing less than 0.05% sulfur. In fact the upper limit is only 0.04% S in the case of the LD process. If the pig iron is off-grade with respect to the sulfur content, it should be desulfurized before putting it in the steel making furnace. This is called external desulfurization. Similarly, if silicon is on the higher side (more than about 1.0%) in the basic pig iron, the lime consumption in a basic process will increase in order to maintain the required basicity. The single slag practice (in which the slag is removed at the end of refining only) may have to be changed to a two- or three-slag practice in order to maintain the slag volume within limits at any time. Both of these factors affect the production rate and yield, and thereby adversely affect the economy. The excess silicon is, therefore, removed before the metal enters the steel making furnace by what is called external desiliconization.

Any additional operation such as these adds to the cost of steel production. Therefore, whether and how the pig iron is to be treated externally has to be viewed in terms of the overall economy of iron and steel making processes. First, every attempt must be made to obtain both sulfur and silicon below the required levels, but if the blast furnace burden and its practice are not conducive in obtaining the desired quality of pig iron one of the following two alternatives is adopted.

5.8.1. Acid Burdening of Blast Furnace and External Desulfurization

The blast furnace is operated for maximum production rate at minimum coke rate, disregarding the composition of the pig iron produced. The temperature in the blast furnace in this case is low and the rate of silica reduction is decreased, leading to more viscous slag; as a result the metal desulfurization is very inefficient. The net result is that the pig iron contains a small amount of silicon but a high percentage of sulfur. Thus iron is then desulfurized externally. This practice may not be possible if the alumina content of the burden is high. It cannot be adopted for blast furnaces in India, where the slag-alumina reaches a 25% level. Externally, desulfurization can be carried out efficiency by intimately mixing good desulfurizing reagents like CaO, CaC_2, Na_2O, etc., with pig iron. At the Domnarfvet plant in Sweden, powdered lime is mixed with pig iron in a rotary furnace to eliminate over 90% of the sulfur. Reagents like CaC_2 and Na_2O are used in foundries to desulfurize cast iron very efficiently on a small scale. Under these circumstances, the blast furnace operator keeps the production rate high at minimal coke rate and does not bother about its sulfur content, which is the worry of the steel maker.

5.8.2. Basic Burdening of Blast Furnace and External Desiliconization

When the alumina is high in the burden, the blast furnace temperature has to be increased to keep the slag free-flowing since alumina raises the softening point of slag. The basicity, as well as the temperature, is raised, to make the slag thin and basic to effectively desulfurize the metal inside the furnace itself to a level less than 0.06%. The high temperature, however, leads to a high silicon content in the pig iron, since silica reduction is increased at high temperature. Sulfur and silicon contents in pig iron in a blast furnace vary inversely with respect to each other. The excess silicon is then either removed externally or by suitably modifying the steel making process. Hot metal can be externally desiliconized by any of the following ways:

1. Ladle desiliconization by (using consumable mild steel pipe) oxygen lancing in a transfer ladle.
2. In an active mixer by maintaining a suitable basic slag.
3. In an acid Bessemer process by partial blowing.

Alternatively excess silicon can also be dealt with by employing more than one slag or by flush-slag practices during steel making.

The blast furnace operator in this case aims to produce metal at high temperature, with a low sulfur content; the silicon content is the worry of the steel maker.

The chief advantage of either of these two ways is that the blast furnace operator has to concentrate on one thing only, hence his job becomes easy and economical.

The progress in external desulfurization has made it comparatively cheaper than using costly coke and running the blast furnace to obtain required desulfurization inside the blast furnace. The acid burdening of the blast furnace is almost universally adopted now to produce low silicon and high sulfur iron at low cost followed by external desulphurization.

5.8.3. Modern Approach to Pretreatment of Hot Metal

The acid and the basic burdening of blast furnace as alternatives for subsequent adoption of process for pretreatment of hot metal was all right when the final sulfur levels in steels were in the range of 0.04% to 0.05%. The present requirements are much more stringent. Since continuous casting is invariably adopted, the sulfur requirement is around 0.02% to avoid transverse cracking. In special steels the sulfur requested is around 0.01% and there are demands for 0.001% or 10 ppm. The blast furnace cannot produce these sulfur levels no matter what modification is adopted in its practice. It has been universally accepted that these very low sulfur contents can only be produced by the modern and efficient external desulfurization techniques. These have thus become essential features of modern integrated steel plants.

Once this mode of external desulphurization is adopted, the latter part of the steel making process for quality steel production with very low levels of impurities (as required, for example, for IF (interstitial free) grade) can readily be made.

Calcium carbide and magnesium-bearing reagents are commonly used either separately or as mixtures in certain proportion for external desulfurization. These are injected along with a carrier gas and have been found to reduce the sulfur contents to extremely low levels.

5.9. Raw Material Trends (USA and Global)

SOURCE: USGS and U.S. Geological Survey, Mineral Commodity Summaries, January 2017

5.9.1. Events, Trends, and Issues:

The expansion or contraction of gross domestic product (GDP) may be considered a predictor of the health of the steel making and steel manufacturing industries, worldwide and domestically. The World Bank's forecast of global GDP growth rates for 2016, 2017, and 2018 was 2.4%, 2.8%, and 3.0%, respectively. The World Bank's forecast for the U.S. 2016, 2017, and 2018 GDP growth rates was 1.9%, 2.2%, and 2.1%, respectively.

The U.S. Government's Corporate Average Fuel Economy standards will nearly double by 2025 with the average mileage for light-duty vehicles increasing to more than 54 miles per gallon of fuel. About two-thirds, by weight, of every vehicle operating today contains steel in sheet metal structural components, deck lids, doors, fenders, and hoods. The U.S. steel industry will continue to introduce a wide variety of advanced automotive high-strength lightweight steels to replace mild steel to satisfy these new requirements.

The *Organization for Economic Co-operation and Development* reported a massive global steel overcapacity estimated at nearly 544 million tons in 2015 and continuing into 2016, as a result of foreign government subsidies and other steel market-distorting policies. Overcapacity, along with depressed global steel demand and import barriers in other markets, had resulted in high levels of steel imports entering into the U.S. market in recent years; however, increased enforcement of duties and increased import tariffs placed on steel imports from select countries reduced imports in 2016. The increase in enforcement and duty rates followed allegations of illegal dumping from many countries in which steel production is thought to be heavily subsidized, including Brazil, China, India, the Republic of Korea, Turkey, and others. In China, the world's leading producer and consumer of steel, significant overcapacity is expected to continue until 2020, while consumption is unlikely to exceed 700 million tons.

Near year-end 2016, the U.S. Department of Commerce was expected to begin a formal investigation to determine whether steel companies in China have been shipping steel through Vietnam to avoid U.S. import tariffs. Trade data show steel shipments from Vietnam to the United States have increased, while shipments from China to Vietnam also increased. In addition, Canada's parliamentary trade committee held its first meeting to study dumping of foreign steel into Canada. The study began, in part, because a large steel producer in Canada was going through insolvency proceedings.

World Production:	Pig Iron		Raw steel	
	2015	2016^{e}	2015	2016^{e}
United States	26	23	79	80
Brazil	28	25	33	30
China	691	685	804	800
France	10	11	15	17
Germany	28	28	43	44
India	58	62	89	83
Japan	81	81	105	105
Korea, Republic of	48	45	70	67
Russia	53	52	71	70
Taiwan	14	14	21	21
Turkey	10	10	32	32
Ukraine	22	24	23	25
United Kingdom	9	9	11	10

Other countries	80	80	218	213
World total (rounded)	1,160	1,150	1,610	1,600

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[e] Estimated.

[1] Production and shipments data source is the American Iron and Steel Institute; see also Iron Ore and Iron and Steel Scrap.

[2] More than 95% of iron made is transported in molten form to steel making furnaces located at the same site.

[3] Source: U.S. Census Bureau. North American Industry Classification System: 3311, 331511, 331512, and 331513.

[4] Defined as steel shipments + imports – exports + adjustments for industry stock changes – semifinished steel product imports.

[5] U.S. Department of Labor, Bureau of Labor Statistics.

[6] Metals Service Center Institute.

[7] Defined as imports – exports + adjustments for industry stock changes.

5.10. Iron and Steel Scrap[1]

SOURCE: U.S. Geological Survey, Mineral Commodity Summaries, January 2017

(Data in million metric tons of metal unless otherwise noted)

5.10.1. Domestic Production and Use:

In 2016, the total value of domestic purchases (receipts of ferrous scrap by all domestic consumers from brokers, dealers, and other outside sources) and exports was estimated to be $14.2 billion, approximately 15% less than that of 2015. U.S. apparent steel consumption, an indicator of economic growth, increased to about 105 million tons in 2016. Manufacturers of pig iron, raw steel, and steel castings accounted for about 88% of scrap consumption by the domestic steel industry, using scrap together with pig iron and direct-reduced iron to produce steel products for the appliance, construction, container, machinery, oil and gas, transportation, and various other consumer industries. The ferrous castings industry consumed most of the remaining 12% to produce cast iron and steel products, such as machinery parts, motor blocks, and pipe. Relatively small quantities of steel scrap were used for producing ferroalloys, for the precipitation of copper, and by the chemical industry; these uses collectively, totaled less than 1 million tons.

During 2016, raw steel production was 80 million tons, up slightly from 78.8 million tons in 2015; annual steel mill capacity utilization was about 72% compared with 70% for 2015. Net shipments of steel mill products were 78 million tons, about the same as those in 2015.

Salient Statistics—United States:	*2012*	*2013*	*2014*	*2015*	*2016*[e]
Production:					
Home Scrap	10	8.5	7.1	6.3	6
Purchased scrap[2]	70	77	62	67	65
Imports for consumption[3]	3.7	3.9	4.2	3.5	4.1
Exports[3]	21	18	15	13	12
Consumption, reported	63	59	59	53	47
Consumption, apparent	63	71	59	64	63

Price, average, dollars per metric ton delivered, No. 1 Heavy Melting composite price, Iron Age Average, Pittsburgh, Philadelphia, Chicago	367	365	351	213	192
Stocks, consumer, yearend	4.2	4.2	4.3	4.4	4.4
Employment, dealers, brokers, processors, number[4]	30,000	30,000	30,000	30,000	30,000
Net import reliance[5] as a percentage of reported consumption	E	E	E	E	E

5.10.2. Recycling:

Recycled iron and steel scrap is a vital raw material for the production of new steel and cast iron products. The steel and foundry industries in the United States have been structured to recycle scrap, and, as a result, are highly dependent upon scrap.

In the United States, the primary source of old steel scrap was automobiles. The recycling rate for automobiles in 2013, the latest year for which statistics were available, was about 85%. In 2013, the last year that data were available, the automotive recycling industry recycled more than 14 million tons of steel from end-of-life vehicles through nearly 300 car shredders, the equivalent of nearly 12 million automobiles. More than 7,000 vehicle dismantlers throughout North America resell parts.

The recycling rates for appliances and steel cans in 2014 were 89% and 70%, respectively; this was the latest year for which statistics were available. Recycling rates for construction materials in 2014 were about 98% for plates and beams and 71% for rebar and other materials. The recycling rates for appliance, can, and construction steel are expected to increase not only in the United States, but also in emerging industrial countries at an even greater rate. Public interest in recycling continues, and recycling is becoming more profitable and convenient as environmental regulations for primary production increase.

Recycling of scrap plays an important role in the conservation of energy because the re-melting of scrap requires much less energy than the production of iron or steel products from iron ore. Also, consumption of iron and steel scrap by remelting reduces the burden on landfill disposal facilities and prevents the accumulation of abandoned steel products in the environment. Recycled scrap consists of approximately 61% post-consumer (old, obsolete) scrap, 23% prompt scrap (produced in steel-product manufacturing plants), and 16% home scrap (recirculating scrap from current operations).

5.10.3. Import Sources (2012–15):

Canada, 70%; United Kingdom, 10%; Sweden, 7%; Netherlands, 6%; and other, 7%.

5.10.4 Events, Trends, and Issues:

Steel mill production capacity utilization peaked at 80.9% in April 2012 and reached 75.1% in June 2016. Scrap prices fluctuated during the first 8 months of 2016, between about \$152 and \$237 per ton. Composite prices published by Scrap Price Bulletin for No. 1 Heavy Melting steel scrap delivered to purchasers in Chicago, IL, Philadelphia, PA, and Pittsburgh, PA, averaged about \$196 per ton during the first 8 months of 2016. Exports of ferrous scrap decreased in 2016 to an estimated 12 million tons from 13 million tons during 2015, primarily to Turkey, Mexico, and Taiwan, in descending order of export tonnage. The value of exported scrap decreased from \$6.1 billion in 2015 to an estimated \$3.3 billion in 2016. World steel consumption was expected to increase slightly from 1.48 billion tons in 2016 to 1.49 billion tons in 2017.

5.10.5. Substitutes:

About 4.8 million tons of direct-reduced iron was used in the United States in 2016 as a substitute for iron and steel scrap, up from 4.1 million tons in 2015.

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[e] Estimated. E Net exporter.

[1] See also Iron and Steel and Iron Ore.

[2] Receipts – shipments by consumers + exports – imports.

[3] Excludes used rails for rerolling and other uses, and ships, boats, and other vessels for scrapping.

[4] Estimated, based on 2002 Census of Wholesale Trade for 2010 through 2014.

[5] Defined as imports – exports + adjustments for industry stock changes.

5.11. Iron and Steel Slag

SOURCE: U.S. Geological Survey, Mineral Commodity Summaries, January 2017

(Data in million metric tons unless otherwise noted)

5.11.1. Domestic Production and Use:

Iron and steel (ferrous) slags are coproducts of iron and steel manufacturing. After cooling and processing, ferrous slags are sold primarily to the construction industry. Data are unavailable on actual U.S. ferrous slag production, but it is estimated to have been in the range of 15 to 20 million tons in 2016. Domestic slag sales[1] in 2016 amounted to an estimated 18 million tons, valued at about $350 million (ex-plant). Iron (blast furnace) slag accounted for about 47% of the tonnage sold and had a value of about $300 million; about 85% of this value was from sales of granulated slag. Steel slag produced from basic oxygen and electric arc furnaces accounted for the remainder.2 Slag was processed by about 29 companies servicing active iron and steel facilities or reprocessing old slag piles at about 135 processing plants in 31 States; included in this tally are some facilities that grind and sell ground granulated blast furnace slag (GGBFS) based on imported unground feed.

Prices listed in the table below are weighted averages (rounded) for iron and steel slags sold for a variety of applications. Actual prices per ton ranged widely in 2016, from a few cents for some steel slags at a few locations to about $110 for some GGBFS. Air-cooled iron slag and steel slag are used primarily as aggregates in concrete (aircooled iron slag only), asphaltic paving, fill, and road bases; both slag types also can be used as a feed for cement kilns. Almost all GGBFS is used as a partial substitute for Portland cement in concrete mixes or in blended cements. Pelletized slag is generally used for lightweight aggregate but can be ground into material similar to GGBFS. Owing to low unit values, most slag types can be shipped only short distances by truck, but rail and waterborne transportation allow for greater distances. Because of much higher unit values, GGBFS can be shipped longer distances, including from overseas.

Salient Statistics—Unites States:	*2012*	*2013*	*2014*	*2015*	*2016*[e]
Production (sales)[1, 3]	16.0	15.5	16.6	17.7	18.0
Imports for consumption[4]	1.2	1.7	1.8	1.4	2.0
Exports	(5)	(5)	0.1	(5)	(5)

Consumption, apparent[4, 6]	16.0	15.5	16.5	17.7	18.0
Price average value, dollars per ton, f.o.b. plant[7]	17.00	17.50	19.00	19.50	19.50
Employment, number[e]	1,800	1,700	1,700	1,700	1,600
Net import reliance[8] as a percentage of Apparent consumption	7	11	10	8	11

5.11.2. Recycling.

Following removal of metal, slag can be returned to the blast and steel furnaces as ferrous and flux feed, but data on these returns are incomplete. Entrained metal, particularly in steel slag, is routinely recovered during slag processing for return to the furnaces, and is an important revenue source for the slag processors, but data on metal returns are unavailable.

5.11.3. Import Sources (2012–15).

The dominant imported ferrous slag type is granulated blast furnace slag (mostly unground), but official import data in recent years have included significant tonnages of nonslag materials (such as cenospheres, fly ash, and silica fume) and slags or other residues of various metallurgical industries (such as copper slag) whose unit values are outside the range expected for granulated slag. The official data appear to have underreported the granulated slag imports in some recent years, but likely not in 2011–12. Based on official data, the principal country sources for 2012–15 were Japan, 33%; Canada, 31%; Spain, 16%; Germany, 5%; and other, 15%; however, much of the tonnage from Spain in 2013–14 may in fact have been from Italy, and slag from the Netherlands and Switzerland in 2015 may have been from Germany.

5.11.4. Events, Trends, and Issues.

The supply of blast furnace slag continues to be problematic in the United States because of the closure and (or) continued idling of a number of active U.S. blast furnaces in recent years, including one in 2015, the lack of construction of new furnaces, and the depletion of old slag piles. Only a limited quantity of locally produced granulated blast furnace slag was available. At yearend 2016, granulation cooling was available at only two active U.S. blast furnaces, down from three in 2014. Installation of granulation cooling was being evaluated at a few blast furnaces, but it was unclear if this would be economic. Pelletized blast furnace slag was in very limited supply (one site only), and it was uncertain if any additional pelletizing capacity was planned.

Basic oxygen furnace steel slag from domestic furnaces has become less available recently because of the closure of several integrated iron and steel complexes; thus, the long-term supply of steel slag will be increasingly reliant on electric arc furnaces, which now contribute the majority of U.S. steel production. Where not in short supply, slag (as aggregate) sales to the construction sector tend to fluctuate less than those of natural aggregates. Domestic- and import-supply constraints appear to have limited domestic demand for GGBFS in recent years. Although prices have increased, sales volumes have failed to match the relative increases that have characterized the overall U.S. cement market since 2010. Long-term demand for GGBFS likely will increase because its use in concrete yields a superior product in many applications and reduces the unit carbon dioxide (CO2) emissions footprint of the concrete related to the portland cement (clinker) content.

Recent regulations to restrict emissions of CO2 and mercury by coal-fired power plants, together with some powerplant closures or conversion of others to natural gas, have led to a reduction in the supply of fly ash in some areas, including that of material for use as cementitious additive for concrete. This has the potential to increase future demand for GGBFS, but the availability of material to supply this demand will increasingly depend on imports, either of ground or unground material. Imported slag availability may be constrained by increasing international demand for the same material and because not all granulated slag produced overseas is of high quality. New restrictions on mercury emissions by cement plants may reduce demand for fly ash as a raw material for clinker manufacture, and this could lead to use of air-cooled and steel slags as replacement raw materials.

5.11.5. World Mine Production and Reserves.

Slag is not a mined material and thus the concept of reserves does not apply to this mineral commodity. Slag production data for the world are unavailable, but it is estimated that global iron slag output in 2016 was on the order of 300 to 360 million tons, and steel slag about 160 to 240 million tons, based on typical ratios of slag to crude iron and steel output.

5.11.6. Substitutes.

In the construction sector, ferrous slags compete with crushed stone and sand and gravel as aggregates, but are far less widely available than the natural materials. As a cementitious additive in blended cements and concrete, GGBFS mainly competes with fly ash, metakaolin, and volcanic ash pozzolans, and to a lesser degree with silica fume. In this respect, GGBFS also competes with Portland cement itself. Slags (especially steel slag) can be used as a partial substitute for limestone and some other natural raw materials for clinker (cement) manufacture. Some other metallurgical slags, such as copper slag, can compete with ferrous slags in some specialty markets but are generally in much more restricted supply than ferrous slags.

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[e] Estimated.

[1] Data are from an annual survey of slag processors and pertain to the quantities of processed slag sold rather than that processed or produced during the year. The data exclude any entrained metal that may be recovered during slag processing and then sold separately or returned to iron and, especially, steel furnaces. The data are incomplete regarding slag returns to the furnaces.

[2] There were very minor sales of open hearth furnace steel slag from stockpiles but no domestic production of this slag type in 2012–16.

[3] Data include sales of imported granulated blast furnace slag, either after domestic grinding or still unground, and exclude sales of pelletized slag (proprietary but very small). Overall, actual production of blast furnace slag may be estimated as equivalent to 25% to 30% of crude (pig) iron production and steel furnace slag as about 10% to 15% of crude steel output.

[4] Based on official (U.S. Census Bureau) data. In some years, the official data appear to have understated the true imports; the apparent discrepancy was small for 2012, but may have been nearly 0.4 million tons in 2013 and 2014, depending on whether imports from Italy were mischaracterized as being from Spain or not. The U.S. Geological Survey canvass captures only part of the imported slag.

[5] Less than 0.05 million tons.

[6] Although definable as total sales of slag (including those from imported feed) – exports, apparent consumption of slag does not significantly differ from total sales owing to the very small export tonnages.

[7] Rounded to the nearest $0.50 per ton.

[8] Defined as imports minus exports.

CHAPTER 6

EFFICIENCY OF STEEL MAKING PROCESSES

In This Chapter

- Refractory Consumption
- Yield
- Production Rate
- Uninterrupted Production
- Quality of Product
- Other Factors
- Concluding Remarks

Before describing the details of various commercial processes of steel making, it would be worthwhile at this stage to know the criteria to assess the efficiency of a steel making shop; this information would be useful in order to obtain the relative cost of production, and to compare its performance with similar shops elsewhere. The efficiency can be broadly assessed in terms of the following parameters:

1. Refractory consumption (per metric ton of steel produced).
2. Yield of the process.
3. Production rate of the shop.
4. Capability for uninterrupted production.

5. Quality of the product.

6. Specific energy consumption.

6.1. Refractory Consumption

Refractory consumption in steel making is one of the most important factors affecting the cost of steel production in any process. Every attempt is made to keep the refractory consumption during steel making, in terms of the overall cost of production, to a minimum.

The refractory lining is expected to withstand the chemical action of molten slag and metal, hot flame and gases laden with corrosive oxides. It must stand the mechanical stresses due to solid and liquid chargings, liquid and gas movements, and furnace movements, if any. It is also subject to high temperatures, thermal shocks and spalling because of the frequent changes in the state of the furnace. The life of the refractory lining primarily depends upon its ability to withstand chemical attack at working temperatures. Since it is not easy to assess the resistance of a refractory to chemical attack, it is usually measured in terms of porosity of the brick or of the rammed or the fritted mass. Broadly, the more porosity, the more the refractory will be prone to chemical attack.

As described in Section 5.6, bricks are made in any one of the following conditions:

(*i*) Chemically bonded, reinforced and unfired.

(*ii*) Fired at usual or at high temperatures.

(*iii*) Direct bonded (requires high firing temperature).

(*iv*) Made from presintered or pre-fused and crushed mix.

(*v*) Melted, cast and cut to required shape.

In general the porosity of the bricks decreases in the order (*i*) to (*v*) as mentioned above, but the cost of manufacture increases in the same order.

The ability of any refractory material to stand high temperatures depends upon the quality of the material, i.e., its chemical composition. The impurities that decrease the softening point of the refractory material should be kept to a minimum. It may involve processing of the raw material to improve its quality and hence may add to the cost of brick making. In other words, better refractory lining is bound to be relatively costly.

In recent years with the working conditions in steel making furnaces getting more and more severe, due to the use of oxygen as an oxidizing agent, and when the furnaces are operated at faster production rates, the quality of the lining in the furnaces ought to be quite good. A costly lining is not necessarily uneconomical in this context as it may, being superior in properties, make the overall operation more economical.

The life of the lining can be prolonged by carrying out hot repairs of the worn out areas. This is known as *fettling or gunning.* Improvement in the lining life by adopting intermediate hot repairs is economical only if the loss in production due to repairs and cost of the repair operation are more than offset by the improved life of the lining. Hot repairs are, therefore, common features of the slow processes like open-hearth and electric arc furnaces and not of fast basic oxygen processes. In the case of faster processes, the refractory lining ought to be of superior quality.

At the end of its useful life, the lining must be replaced by stopping the campaign. The furnace is cooled, the faulty lining wrecked, new lining rebuilt, and finally it is brought back to the steel making condition with respect to its temperature and surface composition. The short lining life not only involves the additional cost of relining, but it makes the furnace idle, so that the availability of the furnace for producing steel decreases; thus, it reduces the level of production. The more often the relining is done, the less the furnace will be available, resulting in reduced production and a higher cost of steel making.

The decisive factor in the choice of a refractory material is not only the actual cost of the lining, but also its life (in terms of steel production), the relining time, the number of furnaces in the shop actually working at any time, the utilization of the used lining, and so on.

Magnesite has definitely proved to be much better than dolomite in resisting the attack of basic slag and metal at steel making temperatures, but it is invariably costlier than dolomite. Use of costly materials must always be coupled with high rates of production and longer campaigns to achieve more favorable refractory cost per metric ton of steel produced.

The techniques of bricklaying, in-situ ramming and fritting etc., are rigorously being standardized to improve the compactness of the lining at working temperatures. The fundamental behavior of the lining material is of great importance in this regard. The accelerated burning of an open-hearth bottom has been possible only through such basic understanding of the behavior. The development of super silica bricks or synthetically prepared magnesite bricks are other typical examples wherein the understanding of the fundamental behavior of lining material has been fully exploited for practical uses. The importance of lining life can be understood thus:

The lining life of an LD vessel in India was once generally around 150-200° using tarred dolomite bricks as against the 800-1200° heat in advanced countries using fused magnesite bricks. This one single factor, notwithstanding any compulsions under India conditions, is a great hurdle in achieving the required economy of steel production vis-a-vis that in the world. The situation was gloomy enough until as late as 1993 when the accepted detailed project report for LD shop No. 2 of Tata Steel had accepted only 160-180°as the LD lining made from tarred dolomite. But Tata Steel's efforts led to improve the life of the LD converter lining initially beyond 1200°, quite unheard of anywhere in the world, for tarred dolomite lining. It has been improved still further, beyond imagination, to several thousand degrees and even 12000° has been claimed to have been achieved. What the fundamental understanding of the steel making process can lead to, needs no further emphasis.

6.2. Yield

It is expressed as the percentage of the input material finally produced in useful form in a steel making process. It is variously described as:

1. Metallic yield $= \dfrac{\text{Wt. of refined steel in the ladle}}{\text{Total wt. of all metallic charge}}$

2. Ingot yield $= \dfrac{\text{Total wt. of ingots in acceptable form}}{\text{Total wt. of metallic charge}}$

3. Fe yield $= \dfrac{\text{Wt. of Fe in steel in ladle}}{\text{Total wt. of Fe in the charge}}$

4. Fe yield in ingots $= \dfrac{\text{Wt. of Fe in acceptable ingot form}}{\text{Total wt. of Fe in the charge}}$

The metallic charge consists of

(*i*) Molten iron.
(*ii*) Cold iron and steel scrap.
(*iii*) Fe content of iron oxides charged.
(*iv*) Finishings added to the furnace or the ladle in any form.

In order to calculate the weight of the metallic charge or Fe content of the charge, many operators take all the above four items into account, whereas others leave out the third or both the third and fourth. At times the recarburizer, being non-metallic, is excluded from the purview. The casting pit practice influences the ingot yield to the extent it produces scrap that includes:

(*i*) Pit and skull scrap.
(*ii*) Spillage.
(*iii*) Ingots rejected due to unacceptable specifications or incorrect sizes.
(*iv*) Scrap from runners of the bottom poured ingots.

In order to assess the relative yields into steel making shops that differ in casting-pit practices, it is better to compare the metallic yield or the total Fe yield in the ladle.

During steel making, metallic loss is due to:

(*i*) Oxidation of metalloids.
(*ii*) Loss of iron in slag (chemical and mechanical).
(*iii*) Ejections as slopping and sparking.
(*iv*) Skull formation, if any, on the mouth of the furnace.
(*v*) Loss of iron as fume and dust.
(*vi*) Loss of iron entrained in slag-flush.

Therefore, the more impure the iron charge, the less the yield. Increase in the steel scrap proportion of the charge will improve the yield, as the scrap contains hardly any impurity as compared to pig iron. More iron is lost via slag phase if the slag volume is more than necessary, if more than one slags are made, an intermediate slag is removed without allowing clean slag metal separation, and so on. Use of more than one steel making process (e.g., duplexing) also results in more total volume of slag; thus the yield decreases. High oxidizing potential in the furnace at the time of removal of slag would result in high iron oxide content in the slag, hence less of yield. The rest of the factors, namely the (*iii*) and (*iv*), are related more to faulty practices; number (*v*) is more serious in oxygen steel making processes, and the last one is due to careless practices.

Yield in any form is an important indicator of the efficiency of converting a given raw material into finished steel in any process of steel making. An improvement of even a 1% yield increases the yearly production by 10,000 t in a one-million metric ton plant, meaning additional earnings per metric ton of steel.

In an integrated steel plant producing iron from iron ores, converting iron to steel, continuously casting the steel into slabs, blooms or billets and rolling these into long or flat semi-finished products, yield is often calculated of the semi-finished product as a percentage of ore, fuel, etc., inputs in terms of iron units. In all these activities the yield in steel making is far more susceptible to operating practices than any other process, and in which maximum loss of iron to waste may take place out of all the stages of ore to steel operations. Hence yield is often talked about in steel making more than in any other operations of processing like casting, rolling, etc. It is equally true that this yield in steel making can also be improved by improving the faulty practices.

6.3. Production Rate

It is a parameter to assess the utility of the capital invested in the steel making shop and the efficiency of the staff in using the equipment provided therein for steel making. Since a steel making shop is a capital-intensive unit it must be fully utilized to produce steel. Every effort must be made to improve the production rate without losing yield and increasing running cost. The production rate is a function of the following variables:

1. Heat time or tap-to-tap time.
2. Campaign time.
3. Downtime.

Heat time or tap-to-tap time is the total time required for converting a batch of raw materials into steel including handling time. It also includes the time required for various repairs e.g., fettling of the furnace. The smaller the heat time, the faster will be the production rate of a given furnace. The campaign time means the duration before the furnace is due again for refining. It indicates the number of heat that would be processed before the furnace is again down for refining. The longer the campaign, the longer will be the period of uninterrupted working of the furnace and greater in general, will be the production rate. The downtime of a furnace means the interval between two consecutive campaigns, and is required for relining and bringing the furnace to shape to commence steel making again.

Shorter downtime means increased availability of the furnace for steel making, thereby higher production rates.

Productivity can be assessed either as tons/unit time (hour, day, year) or metric tons/unit mantime (hour, year) or in arc furnaces as metric tons per available transformer power multiplied by time or metric tons per total energy input multiplied by the time and so on.

The various equipment and services provided in a shop and the process technology should, therefore, be periodically reviewed and if possible, rectified to improve upon these various times in the direction of increased production rates. The production rates of many an old open-hearth and electric furnace shops have thus been improved. This has necessitated the provision of more efficient services and material handling systems along with introduction of new process technologies.

The importance of reducing the heat and downtimes and increasing the campaign duration can be readily ascertained in the fast basic oxygen processes. A saving of even a minute in an LD heat may mean an increase in production by over 2%, thus even seconds are well worth saving in these processes. The use of in-line and off-line computers to efficiently carry out the material and thermal balance calculations has almost become a necessity in the oxygen steel making processes. Increasing use of such aids is bound to avoid delays and improve upon production rates in any steel making process, in particular, in oxygen steel making processes.

6.4. Uninterrupted Production

The importance of uninterrupted production of steel is very important in a steel plant wherein the production line of iron to steel to semis is to be maintained at the designed speed to obtain optimum production efficiency as a whole. An iron and steel plant consists of a large amount of huge mechanical equipment operated through an equally large number of transformers, motors, pumps, etc. All this electrical and mechanical equipment needs very elaborate maintenance, periodic replacements of worn-out parts, up-grading of facilities, and so on. There are therefore regular *shutdowns* to deliberately stop working of the equipment, one or more at a time, and carrying out the repairs/replacements of parts. Very elaborate electrical and mechanical maintenance divisions are therefore an inseparable adjunct of any modern steel plant. If this maintenance is not carried out regularly, *breakdowns* may occur, with additional costs of repairs and consequently the inevitable loss of production. An optimum schedule of maintenance is worked out and every effort is made to follow that schedule so that no undue and unforeseen loss of production takes place on account of breakdowns. In fact, a well-maintained plant should not lead to any breakdown whatsoever. The German or the Japanese plants are often run on these principles. All the maintenance schedules are taken as acceptable interruptions in the schedule of production. Any unforeseen breakdown of any equipment is likely to affect production adversely and in a haphazard manner, since the iron and steel production is a systematic linear activity and all sub-steps intimately depend upon each other in the forward and backward direction. Any improvement in the performance of any equipment in the production line is beneficial in this respect. The dominant factor in this respect is the steel production. In an LD shop wherein one vessel works at a time, all repairs must be over by the time the other's life is over. With passing years, and due to wear and tear, the maintenance problems and periods grow, and here lies the danger of interruption in steel production. A substantial increase in the life of the vessel can leave one vessel free for even abnormal maintenance, and yet continue steel production via the other. If this happens the indirect benefits of uninterrupted steel production are far greater, and lead to far better economy.

In the management of a complex unit such as a steel plant, the uninterrupted production of steel, particularly by very fast steel making processes, definitely improves the efficiency as a whole.

6.5. Quality of Product

The quality of the product is a parameter to assess the ability and the skill of the staff in a steel making shop to utilize the equipment provided therein. The term quality indicates how closely the specifications are met in the product. The specifications for steel may include its:

1. Chemical composition.
2. Gas content.
3. Cleanliness.
4. Mechanical and physical properties.

The chemical composition in the specifications indicates a range of desirable elements and maximum allowable impurity contents. It may also include macro-segregation limits for various constituents e.g., sulfur. Although many times the gas content and the cleanliness (the nature, number, size, shape, and distribution of inclusions) are not specified, the gas content must be below a certain safe level, and the steel should similarly be safely clean. The mechanical properties and other requirements must also be met fully.

The quality of steel can be improved by controlling the process at every stage to obtain more closely the desired chemical specifications and homogeneity with the least gas content and, as clean a product as possible. This achievement varies from shop to shop and furnace to furnace depending upon the skill and ability of the operators to utilize the equipment to produce the desired product. This is also known as quality control of the process. It is easy to control the quality in hearth processes because these are slow. But it is more difficult to control the quality in fast pneumatic processes. In general, better quality control means less off-grade heat; thus an increased output of the shop.

Proper quality control ensures the customer his requirements. This however may not work satisfactorily every time. In such cases it should be possible to divert the off-grade heat to some other nearby (chemistry-wise) acceptable specification and marketed accordingly without loss to the plant. This diversion may result in losing some value as diversion cost. This is a commonly adopted practice in steel making and marketing, particularly when a large number of grades with closer specifications are produced and sold.

6.6. Other Factors

Besides the above-mentioned factors, the efficiency of a steel making operation is also affected by the energy consumption and cost of raw materials used in the process.

Energy is consumed in steel making for melting, refining and/or operating the various equipment in a steel making shop. In autogenous processes like Bessemer, LD, etc., energy is consumed only in operating the equipment; as such it is not a substantial factor in the overall cost of steel making. However, in open-hearth and electric furnace processes, energy is consumed in melting and refining, hence its cost forms a substantial item in the overall cost of steel production. Every attempt should, therefore, be made to minimize energy consumption in these processes. Proper furnace design and efficient operation can reduce energy consumption by reducing the heat losses from a furnace. Technology improvements in refining, e.g., the use of oxygen lancing and the exothermic energy of refining reactions, can reduce the external energy requirement of the process.

The cost of steel making can also be reduced by using cheaper quality raw materials without adversely affecting the quality of the products. The possibility of using sponge iron and cheaper scrap could be assessed in this respect. This could be very important in electric furnace processes. Similarly any reduction in the cost of raw material preparation, if any, can also affect process economy.

6.7. Concluding Remarks

The cost incurred in steel making can be categorized into the following main factors:

(*i*) *Material costs.* Charge, alloys, oxygen, fluxes and such consumables.

(*ii*) *Conversion costs.* Energy, refractories, electrodes, etc.

(*iii*) *Fixed costs.* Wages, depreciation, interest, services, etc.

When similar plants are to be compared, many aspects must be assumed to be identical. The wage-bill is to be related to number and quality of personnel provided, and that in turn, to be related to their achievement in the form of production rate, quality of the product, yield, etc. Since other material costs are to be assumed to be identical for comparison, only refractory consumption and yield have to be compared for performance assessment of similar shops.

Thus, on the whole, the efficiency of a steel making shop can be broadly assessed in terms of production rate, yield, quality of product, and refractory consumption.

CHAPTER 7

LAYOUT OF A STEEL MAKING SHOP

In This Chapter

- Location
- Size
- Layout

Fundamentally, the layout of any plant or shop means a rational arrangement of equipment, material movements, energy network, plant services, etc., for achieving an efficiently functioning floor plan for the manufacture of a given product or the economical supply of a given service. In the case of steel making the necessity to develop a plant layout may arise under the following circumstances:

(*i*) To establish a new plant on an entirely new site. (Green field installation).

(*ii*) To install an additional steel making shop in a steel plant to augment its production.

(*iii*) To expand an existing shop by adding a new furnace(s) and its auxiliaries.

(*iv*) To improve efficiency and economics of the existing plant through changes in the process, additions of auxiliary equipment, introduction of a new technical development and so on.

(*v*) To replace an existing steel making furnace by other more modern design.

Several economic and technical factors are to be considered to justify the implementation of the above necessities. The nature of the problems involved in designing a lay-out in each one of the above cases are quite different and need careful attention. Planning of a layout is a complex problem involving a large number of factors to be properly considered. A preliminary project report about its economic feasibility is first made and once it is accepted for implementation, a detailed project including the layout design is made.

7.1. Location

The problem of selection of a site arises only if an entirely new plant is to be set up. The cost of production per metric ton of steel should be most economical to justify the installation of a new plant at the proposed site. In general the availability of the following facilities is considered before deciding the site for the location of a steel plant.

(*i*) Good grade raw materials at close proximity, or transportable via economical routes (like water transport).

(*ii*) Adequate power, fuel, and water supply.

(*iii*) Cheap and efficient mode of transport.

(*iv*) Nearby market and/or export potential.

(*v*) Cheap real estate.

(*vi*) Ample labor of required quality.

(*vii*) Maximum taxation benefit.

(*viii*) Solid underground soil strata.

(*ix*) Good topography of the region.

(*x*) Healthy climatic conditions of the area.

In addition, the site should, as far as possible, be strategically safe and relatively free from labor troubles.

It will be a rare occasion when all the above facilities are available at one single place. A few of these may be favorably available at each place. Generally a steel plant is located where either raw materials or market is available. Whether it is a raw material-based or a market-based plant, the location of a steel plant depends upon the overall economy and site, resulting in a minimum of cost of production. In a democratic country, social aspirations of the people in the region may, at times, predominate over other economic considerations in the selection of a location for plants set up by government agencies. For an integrated steel plant all these considerations apply to iron and steel making as a whole.

7.2. Size

There is always what is known as an economically viable size of any plant. It is the minimum size; a plant of any bigger size must be set up to produce steel at a profit. The actual size—the annual output in metric tons of ingots—depends on a number of factors:

(*i*) Type of steel to be produced, its demand and the export potential.

(*ii*) Production technology, economical size(s) and number of steel making furnaces to be employed.

(*iii*) Administrative problems and expenditure.

(*iv*) Strategic and social considerations.

(*v*) Tax concessions related to annual production.

Since alloy steels are required in small amounts, steel plants producing these are of a much smaller size than those producing plain carbon steels that are required in large bulk. Steel making shops vary in size from one producing a few thousand metric tons to one producing several million metric tons annually. At one time a steel plant producing less than 10,000 t/annum was considered a small plant, and one producing one million metric tons per annum was a big plant. The situation has, however, changed. Even one million metric tons per annum plant is now considered small, and only 5-15 million metric tons per annum as big plants. Those producing less than 100,000 t/annum are often referred to as mini-steel plants.

In India there are several such mini-steel plants, each producing around 10-30,000 t/annum or even less. These have now become, by and large, uneconomical. Consequently, their size is being increased to make them economically viable.

7.3. Layout

A steel making shop may have some or all of the following facilities and equipment:

(*i*) Mixer—to store hot metal.

(*ii*) Melting furnaces—to melt cold charges in shops running on cold charges.

(*iii*) Furnace(s) proper—to refine metal.

(*iv*) Working floor—to carry out various operations of the furnace during refining.

(*v*) Control cabin—to house process control instruments.

(*vi*) Raw materials bins—to store solid raw materials.

(*vii*) Scrap preparation yard—to receive, sort out, and prepare the scrap charges.

(*viii*) Pit-side—to teem ingot molds.

(*ix*) Vacuum treatment plant—to degas steel.

(*x*) Stripper yard—to strip-off ingots from molds.

(*xi*) Mold reconditioning and assembly—to make mold suitable for reuse.

(*xii*) Calcining plant—to calcine dolomite, limestone, magnesite, etc.

(*xiii*) Tar mixing unit—to make tarred dolomite, or magnesite, as ramming mix.

(*xiv*) Chemical analysis laboratory—to chemically analyze the slag, gas, metal, etc.

(*xv*) Repair and maintenance section—to reline the furnace and its auxiliaries, and for general repairs.

(*xvi*) Overhead cranes, railways—to transport materials and movable system and belt equipment conveyers.

(*xvii*) Computer section—to carry out data collection, processing, and interpretation for process control.

(*xviii*) Fuel and power supply—to supply these to the various equipment.

(*xix*) Weigh bridges—to weigh materials at various stages.

(*xx*) Office—to collect information and issue instructions for running the shop.

Depending upon the size of the plant, the quality of materials available, steel making process adopted, and the quality of the product to be produced, the various facilities and items mentioned above may be included in designing the layout of the shop.

Broadly, a large steel making shop may have the following main zones:

1. Furnace bay— contains the furnaces, working floor, instrument cabin, raw material bins, fuel, or power supply arrangements, etc.
2. Stock —it stores all or some solid raw materials.
3. Pitside — It contains tapping and teeming platforms, and working space.
4. Mixer bay.
5. Stripper yard.
6. Mold reconditioning and assembly.
7. Calcining plant.

The other minor facilities and equipment are suitably located in these main sections. The vacuum degassing unit, continuous casting machine, etc., may be located on the pitside at appropriate places. The chemical analysis laboratory, computer section and the like, often form a common facility

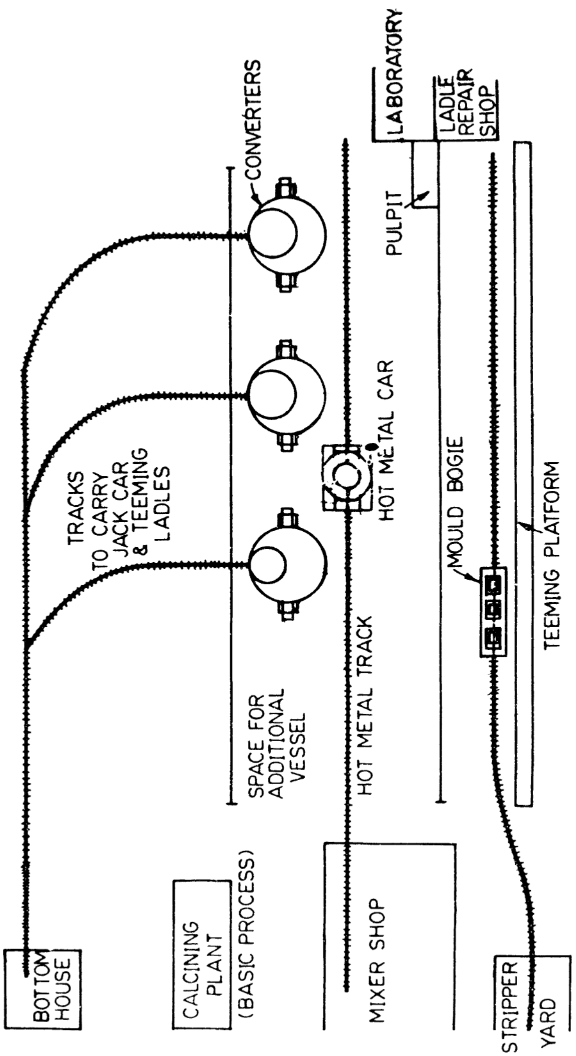


FIGURE 7.1. Layout of a basic Bessemer/Thomas shop (schematic).

for the entire plant and may be situated away from the steel making shop as an independent units(s). In that case pneumatic channels are used to send samples from the shop to the laboratory and the results are communicated back to the shop through the LAN of computers. This may not take more than a few minutes.

In small plants, particularly those that are not integrated steel plants, the above list of items is much smaller and these are all put together in one or two bays only. A typical example is a small electric furnace steel making shop.

The quality of a layout is judged in terms of the following factors:

1. Simplicity in design.
2. Efficient production planning by way of standardization of equipment, its arrangement and materials handling.
3. Incorporation of technological advances in the original design with scope to incorporate techniques that may show promise in future.
4. Low capital cost with ample scope for future expansion with a minimum of extravagance.
5. Adequate and efficient transport facilities between various units of the shop.
6. Short and direct material flow lines for raw materials, finished products, and by-products with a minimum of handling cost, delays, and interference.
7. Adequate stores, repairs and maintenance facilities.
8. Comfortable, well illuminated, properly ventilated, and safe working conditions.
9. Efficient communications system.

The importance of material handling is of paramount significance in designing a steel plant layout so much so that in one type of a layout, railway tracks are laid out first, in terms of material movements, and the rest of the equipment is arranged accordingly. This is called a railway type layout. Several overhead cranes are also required in the shop.

Apart from the routine movements of main materials, a steel making furnace needs to be repaired periodically. The repair schedule, which is included in the layout, should not interfere with the regular working of other furnaces. In spite of adequate provisions of equipment and maintenance, standby capacities may be included in times of emergency.

It must be clearly understood that an ideal layout is almost impossible to achieve, and that some degree of improvement is always possible at a later date. However, while designing, particularly a new plant, all possible care should be taken to approach an ideal layout. No two plants are alike, although the basic principles utilized in designing them are the same.

As different units develop inefficiency at different rates, an initially well planned layout may be thrown out of balance in time, and its overall

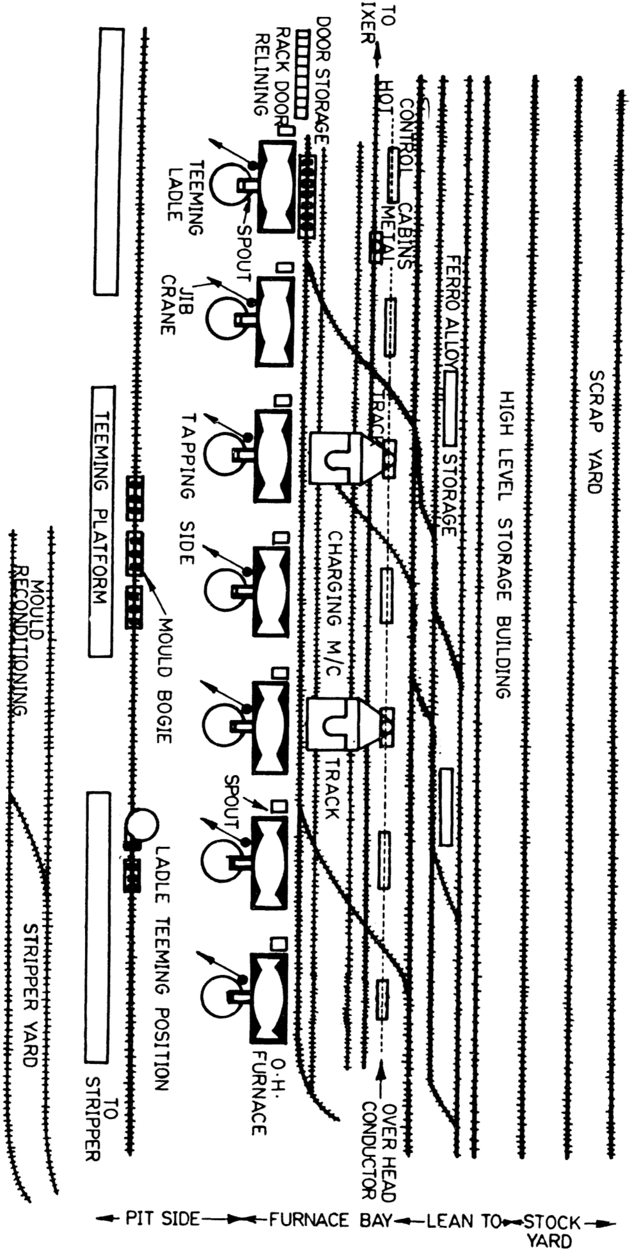


FIGURE 7.2. Layout of an open hearth or twin-hearth shop (schematic). Similar are used for their modified versions also.

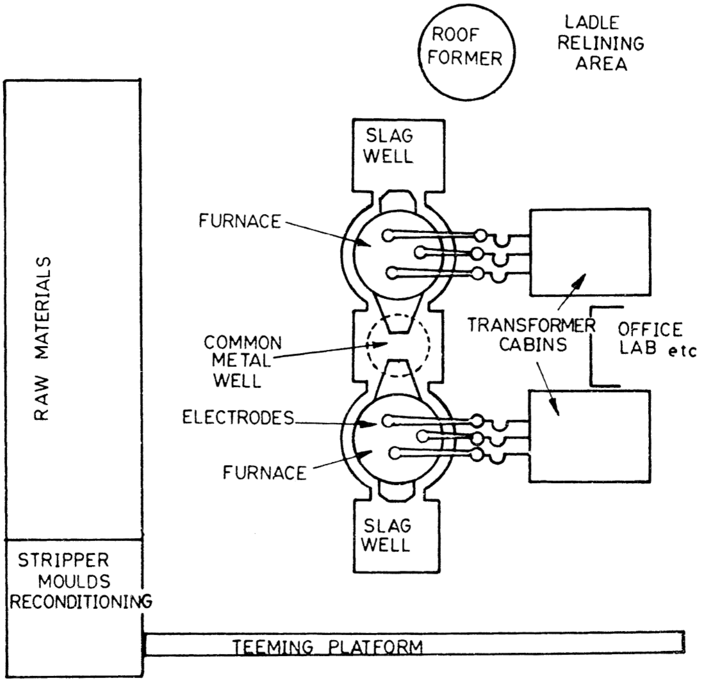


FIGURE 7.3. Layout of an old electric arc furnace shop (schematic).

efficiency may decrease. All technological processes and their future potentialities should, therefore, be duly considered in the initial planning itself.

Lastly, the social and psychological state of the workforce employed to run the shop must also be properly considered while designing a layout.

A few of the schemes of layouts of steel making shops are described in Figures 7.1 to 7.4. Generally the pitside and the scrap preparation yard are on either side of the furnace bay, but in some designs they may be located side by side, with the pitside in between the furnace bay and the scrap preparation yard.

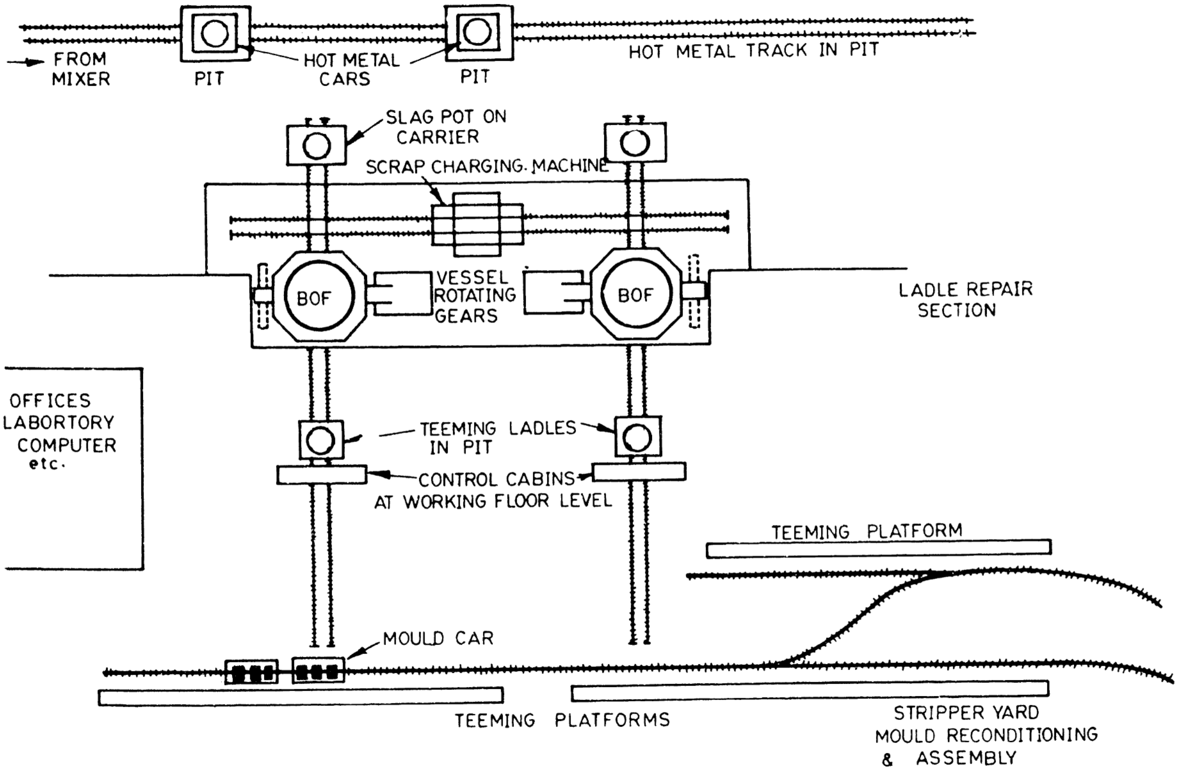


FIGURE 7.4. Layout of a BOF plant (schematic).

CHAPTER 8

RISE AND FALL OF STEEL MAKING TECHNOLOGIES-I PNEUMATIC PROCESSES

In This Chapter

- Ancient Steel Making Processes
- Modern Steel Making
- Nitrogen Problem
- Concluding Remarks on Old Pneumatic Processes

It should be understood that in ancient times the term steel, as distinct from iron, was unknown. The iron age means, in fact, the iron and steel age in the modern sense of the term. In this sense the history of steel making means history of not less than a few thousand years, of iron and steel making together. The details can be seen in the author's book on ironmaking by the same publishers. Some old processes of steel making are described below to appreciate the philosophy of the development of steel making.

8.1. Ancient Steel Making Processes

Pig iron as such is hardly a useful material for practical purposes because of its high impurity and gas contents. On solidification most of these gases are desorbed. By remelting and minor adjustments of composition, it can be rendered useful as cast iron for founding. Cast iron is not malleable. The malleable form of iron can only be produced by removing most of the impurities present in pig iron.

8.1.1. Wrought Iron Making

Pure iron melts at 1536°C. The presence of impurities decreases the melting point of iron so much so that pig iron is usually tapped from a blast furnace around 1250-1300°C in the form of a free-flowing carbon-rich liquid. Prior to the 1850s the maximum attainable temperature in furnaces fired with chemical fuels and cold air was of the order of 1450°C. In such furnaces pig iron could be melted. As the impurities of iron are eliminated by oxidation, the melting point of iron increases and the refined mass becomes semi-solid. This pasty mass of iron, nearly pure but contaminated mechanically by slag particles, came to be known as wrought iron. It was malleable but very soft. The process of making wrought iron was known as the puddling process. Later a process known as the Aston-Byer process was developed to make better quality wrought iron. Wrought iron was used extensively as a soft ferrous material for various applications like ship hooks, chains, etc.

Wrought iron lacked the necessary hardness required for making cutting tools. It was carburized in solid state by heating in contact with carbonaceous material for a sufficiently long time for carbon to diffuse in. The iron-carbon alloy, currently called steel, was thus produced and used for various applications. This is known as the cementation process of steel making. Its origin is lost in antiquity but similar processes dominated the steel industry prior to the 19th Century. The product was known cement or blister steel.

8.1.2. Crucible Steel Making

The cementation technique was well developed to produce as uniform a product as possible. But it still lacked homogeneity with respect to carbon distribution. It was Huntsman who in 1740 successfully melted this material in crucibles to produce molten steel for the first time. This was possible because as the carbon in steel increased, the melting point decreased, and then furnaces could melt the high carbon steel. This was known as the crucible process of steel making. This process dominated the steel industry for making cutting tools, swords, daggers, clock springs, etc., in the 18th and 19th Centuries, even for some time after the advent of modern steel making.

8.2. Modern Steel Making

Modern steel making begins with the Bessemer process. It revolutionized the steel making going forward. Henry Bessemer patented his process in England in 1856, and in the same year Robert Mushet established the beginning of the deoxidation practice necessary to produce good quality steel ingots. He established that a small amount of *spiegel* (low manganese Fe-Mn) must be added to refined steel to render it workable. The process came to be known as the Bessemer process.

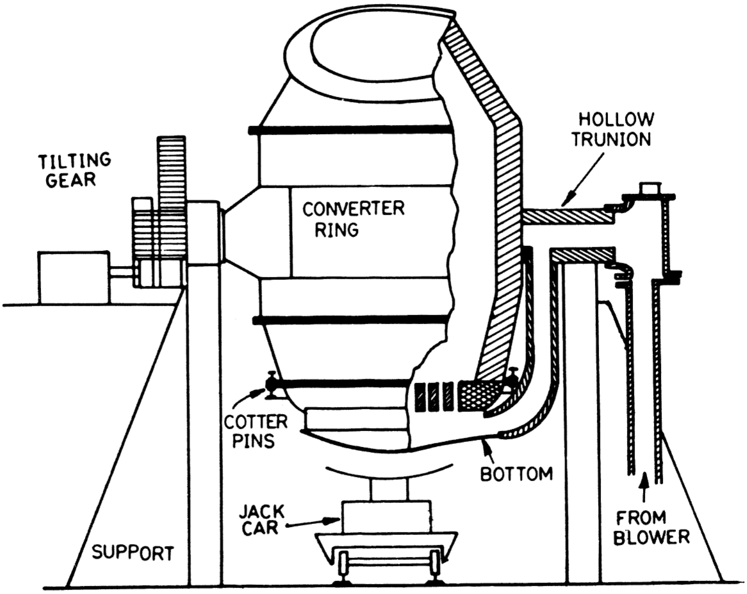


FIGURE 8.1. Bessemer converter mounted on trunions.

8.2.1. Bessemer Process

The first Bessemer plant was installed in the UK in 1861. It was an acid Bessemer process. In this process, cold air was blown through molten iron held in a bottom-perforated vessel called a converter, that which converts iron into steel. The vessel was lined with silica bricks. It could remove only Si, Mn, and carbon impurities from iron. It produced viscous, nearly silica-saturated dry slag, which readily separated from molten steel to produce reasonably clean steel. The physics of the process resembles gargling to cure a sore throat. Bessemer developed the necessary engineering required for charging the converter, blowing, tapping, and so on, as shown in Figure 8.1.

The molten iron was charged at around 1300°C and blown for about 15 minutes when the final temperature was of the order of 1600°C, because of the exothermic reactions of refining. The rate of refining was very fast because of turbulence created by blowing action. A decarburization rate of nearly 10-15 wt %C per hour was readily obtainable. The process was autogenous and the thermal balance had to be established by adjusting the

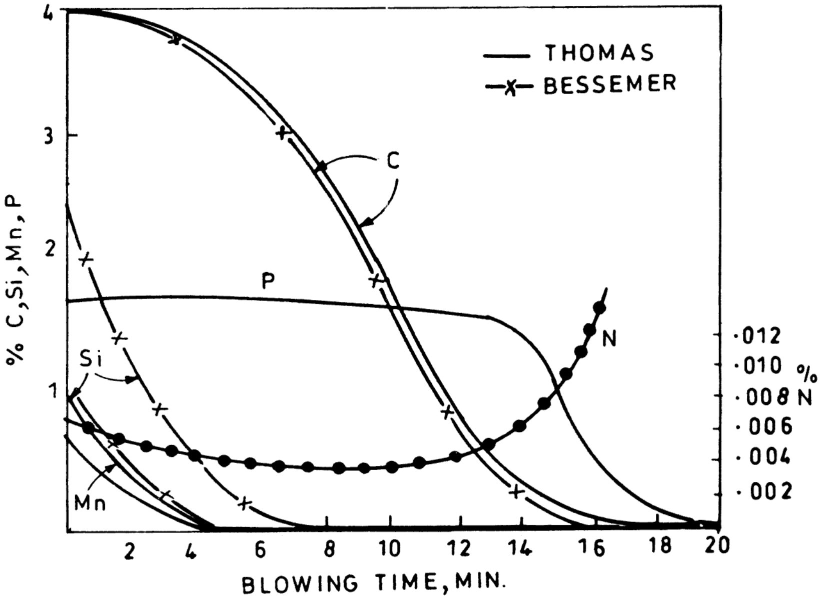


FIGURE 8.2. Schematic of sequence of evolution of impurities during blowing in Bessemer and Thomas processes of steel making. Solid lines are for Thomas and those with a cross in-between are for Bessemer. The nitrogen line is for Thomas only.

charge iron composition, particularly silicon at around 2.5% to chiefly produce the required heat for the process.

The required iron composition was 2.5% Si, about 1.0% Mn, around 4% carbon and P and S both below 0.05% each. The process thus required Swedish iron as charge. It could not remove P and S.

The slag produced analyzed as:

Silica	*Mn-oxide*	*Fe-oxide*
55-65%	12-18%	18-12%

balance 5-10% as other minor oxides.

Typical thermal balance is as follows:

Heat Input (%)		
Molten iron	32	
Warm converter	6	
Slag formation	1	nearly 62% from the refining
C - oxidation	25	
Si - oxidation	26	
Mn - oxidation	5	
Fe - oxidation	5	
	100	

Heat Output (%)		
Molten iron	45	
Molten slag	8	
Waste gases	25	nearly 41% is a waste
Decomposition of blast moisture	1	
Heat losses to surroundings	15	
Warm converter	6	
	100	

Metallic yield = 100 –; (%C + %Si + %Mn + %Fe) oxidized – iron ejected
= 100 – 3.5 – 2.5 – 1.0 – 4.0 – 2.0 = nearly 87%

The sequence of elimination of impurities is shown in Figure 8.2.

8.2.2. Basic Bessemer/Thomas Process

The early Bessemer process was modified by Thomas to enable to accept the continental irons containing around 2% phosphorus. The lining was changed to basic, made from dolomite, and limestone was incorporated in the charge to make good basic slag capable of interaction with metal to remove not only silicon and manganese but phosphorus as well. The silicon had to be kept below 1% to contain slag volume. The phosphorus is the chief heat producer to make this an autogeneous process. The iron composition needed is:

Si—less than 1·0%. Mn—0·7—1%, P—minimum 1·5%, S—max 0·05%

The slag produced analyzed:

CaO	P_2O_5	FeO	SiO_2	
45 – 50%	18 – 22%	15 – 18%	5 – 10%	balance other minor oxides.

The thermal balance is like:

Heat input (%)		*Heat output* (%)	
Molten iron	34	Heat in gases, decomposition of moisture, etc.	25
Oxidation of impurities	60	Slag formation	20
Slag formation	6	Finished steel	42
		Heat losses	13
	100		100

The slag produced is generally 20-25% of the charge weight and thus lime requirement will be 10-12% of the charge metal weight. The sequence of elimination of impurities is shown in Figure 8.2.

8.2.3. Side-Blown Converter

It was developed to produce steel on a small scale to meet local demands for steel castings in particular. The refining vessel was fairly similar in design, and working as the Bessemer converter with the exception that blowing was carried out from the bottom-side rather than from the bottom as in the Bessemer process. The scheme is shown in Figure 8.3. It was an acid process and worked on selected pigs of Swedish grade. Usually the pigs were melted in cupola and then blown in the side-blown converter to produce steel. These were of very small capacities. These were also at times meant to produce steel from steel scrap.

The side-blown converter was in use until World War II, more so during the war. These were used in the railway workshops of Ajmer and Parel (Bombay) during the war but have long been abandoned. A similar plant was installed at Khopoli near Pune. It used a lining-less cupola to melt steel scrap at the rate of 8 T/hr. Coke and Fe-Si were added in the charge to produce impurity levels of 2% Si and 2.5%C in the molten iron. This artificial iron was blown for 20 minutes in a 2 t converter to produce steel. All of these have been abandoned long ago.

Much of the world's pig iron contains phosphorus in the range of 0.2-0.4% and as such, neither of the Bessemer processes can be used for making finished steel. After the war, the side-blown converter practice was under active investigation to deal with such pig iron. The successful development of the LD process in the West around the same time, to treat the same type of irons, resulted in abandoning further investigations on basic side-blown practice there. It was however practiced in the 1950s in what are known as back-yard steel plants in China. The information about it is quite scanty.

The Indian pig iron contains 0.2-0.35% P and is no exception to the above. The possibility of using a basic side-blown converter in India to produce steel on a small scale at a large number of places was seriously investigated at the NML Jamshedpur (1970-74), then shown to be commercially viable. The industrial situation has since changed and the electric furnaces were preferred for conversion of scrap to steel, and the LD process for molten iron to steel. The side-blown converter is now more of historical significance only.

8.3. Nitrogen Problem

The classical Bessemer process, being acid in nature and since only at exceptional places that the Swedish grade iron was produced, was not adopted to any large extent even in its heyday. The basic Bessemer or the

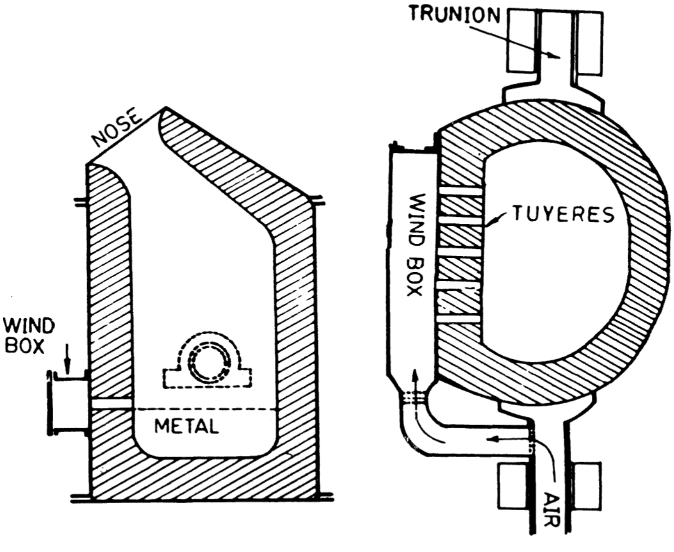


FIGURE 8.3. Side-blown converter.

Thomas process was widely used on the Continent as it suited their iron most. During the fore-blow, i.e., during the period when Si, Mn and C are eliminated, the gas phase in the converter contains a substantial amount of carbon monoxide, hence the partial pressure of nitrogen in the gas phase is low. During the after-blow, that is when only P is being oxidized, the gas phase is almost wholly nitrogen, and this led to pickup of nitrogen by way of dissolution in the liquid iron. This was almost 0.018% as against 0.003% in the basic open hearth and 0.002% in LD processed steel. In the absence of LD, open-hearth was preferred, but once LD was available the Thomas process was in trouble.

The amount of nitrogen dissolved in iron under equilibrium conditions is given by Sievert's law. Since equilibrium is not attained in practice, the amount of nitrogen dissolved in iron will be determined by the following factors:

(*i*) Partial pressure of nitrogen in the blast.

(*ii*) Time of contact as decided by the bath depth.

(*iii*) Length of after-blow.

(*iv*) Temperature of the bath.

Any alterations that decrease any one or more of the above factors will automatically reduce the nitrogen content of the bath. The modifications that have been used in practice and found useful in reducing nitrogen contents are as follows:

(*i*) The partial pressure of nitrogen, particularly in the after-blow, is reduced by increasing the oxygen content of the blast or addition of stream, CO2, etc., in the blast.

(*ii*) The vessels have been designed to have a shallower bath per metric ton capacity than earlier designs.

(*iii*) Use of oxygen-enriched blast and surface lancing of oxygen to reduce length of the blow.

(*iv*) Addition of iron oxide from top or along with the blast to cool the bath, since it consumes heat in its own dissociation.

To withstand competition from the equally fast and better-quality-ensuring BOF processes, the Thomas process had to be modernized to reduce the nitrogen contents of its products in the 1950s. It underwent modifications with respect to the following:

1. The blast was enriched up to 40% oxygen, and consequently the nitrogen content fell down to 0.006-0.008%. Oxygen enrichment beyond 40% proved disastrous because of extremely rapid wear of bottom tuyeres, hence it could not be adopted in spite of the fact that higher oxygen would bring down the nitrogen level still lower.
2. The air blast was replaced by a mixture of 40-60% oxygen and balance steam and was found to lead to nitrogen levels quite comparable to those of open hearths. This was considered as very low nitrogen.
3. The air blowing halfway through the process was replaced by oxygen and carbon dioxide mixture in 50 : 50 ratio. This also resulted in final nitrogen levels comparable to LD levels.
4. The blowing practice was drastically altered. First 40% oxygen enriched blast was blown from the bottom with simultaneous additions of iron oxide as coolant from the top. Then pure oxygen was blown from the top with the vessel in an inclined position as in the Kaldo process. The heat was finished with small oxygen-enriched blow from the bottom. This process came to be known as the Phoenix Lance process and was once quite popularly adopted on the Continent.

8.4. Concluding Remarks on Old Pneumatic Processes

The continental steel making was characterized by the wide adoption of the Thomas process capable of dealing with high phosphorus irons. This process also produced an additional byproduct in the form of phosphoric slag salable as fertilizer. The productivity of Thomas plants was of the order of 60-70 t/hr as compared to 20-30 t/hr of open hearths then.

In the form of modified Thomas processes, the products could be used for rolling into thin sections or deep drawing and non-aging variety useful for the car industry. It thus continued production in the 1960s and 1970s until the advent of oxygen bottom-blowing in the form of OBM, Q-BOP, or LWS processes. These bottom-blown oxygen processes rapidly replaced the old Thomas process shops in the 1970s and 1980s.

Due credit must be given to the ingenuity of Bessemer and Thomas in developing the necessary engineering required for the design, construction, installation, and operation of the converter to produce steel with a minimum of interruption in production. These engineering features are still being used by even the latest LD, OBM, or the hybrid process converters. Similarly these pneumatic processes established the hallmark of production rates even for the latest steel making processes. Their study from these angles is important to understand the technological developments in steel making.

CHAPTER 9

RISE AND FALL OF STEEL MAKING TECHNOLOGIES-II OPEN HEARTH FURNACE PROCESSES

In This Chapter

- Introduction
- Open Hearth Furnace - Design and Shop
- Heat Supply
- Thermal Efficiency
- Types of Open-Hearth Practices
- Oxidizing Medium
- Acid Open-Hearth Practice
- Basic Open-Hearth - Cold Metal Practice
- Basic Open Hearth-Hot Metal Practices
- Modified Open-Hearth Practices
- Products of Basic Open-Hearths
- Characteristics of Open-Hearth Practice
- Later Day Open-Hearth Practices

9.1 Introduction

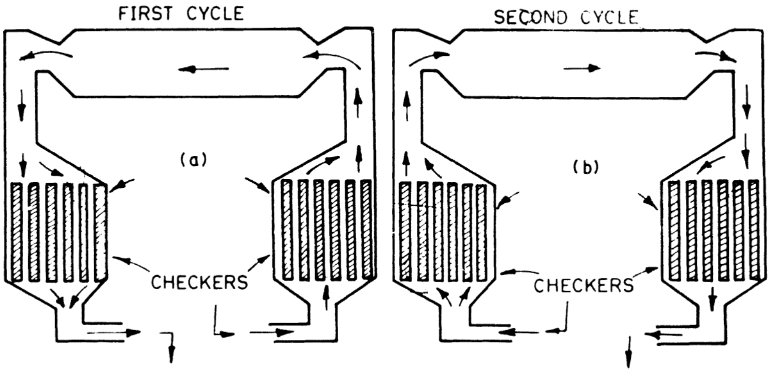
The open-hearth furnace processes are now almost extinct in their original and varied modified forms. In the modified form it does exist at some exceptional or isolated places. For example the open-hearth in its modified form as twin-hearth still continues to operate at the Bhilai steel plant in India. Elsewhere these have been replaced either by the BOFs or EAFs. It however dominated the steel making industry for over 100 years producing a variety of good quality steels to suit the existing requirements. The reasons it dominated the steel industry for such a long time, and finally why it was replaced by other modern processes of steel making, must be understood clearly to appreciate not just its chronological history but the underlying thermodynamics and kinetics, and heat transfer principles.

Even the history of development of the open-hearth process is very interesting and needs to be appreciated in the light of general scientific developments that were taking place around the same time.

The open-hearth process not only became possible but actually developed along with notions underlying thermodynamics in general, more precisely, the concept of quality and intensity of heat energy expressed as its Second Law. Prior to the 1850s the furnaces, fired by chemical fuels burned with cold air blast, were conditioned to maximum achievable temperatures. The maximum temperature that could be achieved by burning chemical fuels with cold air was in the range of 1400-1450°C. No other additional means were then known to raise the furnace temperatures beyond this limit. This level of temperature was nevertheless adequate to dissolve carbon in wrought iron to finally produce molten high carbon tool steels in closed crucibles. Homogeneous mild steel containing around 0.2-0.3% carbon was therefore nonexistent. The chief reason for this limitation was that the theory of heat was not understood until then.

Around 1815-1820 R. Stirling, a clergyman by profession, understood the principle of heat regeneration and developed the methodology for heat regeneration as shown in Figure 9.1.

In this, a part of the heat in the exhaust gases is somehow stored and returned to the furnace as sensible heat in a reversible cycle. The part of the heat was stored in refractory checker works. The works of Carnot, Clapeyron, Clausius, and several others led to the establishment of the science of thermodynamics on a sound footing by the 1850s. The Siemens brothers in England, being well versed in basic sciences, could utilize the concepts of heat energy and its intensity; and the heat regeneration principle enunciated by Stirling to modify the then glassmaking reverberatory furnace with additional fittings developed temperatures around 1600°C for the first time in 1861.



(*a*) (*b*)

(a) First cycle. The checkers on the left-hand side are being heated by the outgoing gases and those on the right are giving up their heat to incoming air of fuel gas.
(b) Second cycle. The checkers on the left-hand side having been heated, are now giving out their heat to incoming air or fuel gas and those on the right, having given up their heat in the first cycle, are now being heated by the out-going gases.
FIGURE 9.1. Illustration to depict the heat regeneration principle.

The Bessemer process of steel making, commercialized in 1858, had started producing mild steels but the scrap generated could not be recirculated and utilized in steel making. Heaps of such steel scraps were lying idle. Steel scrap could now be melted in such a modified glass furnace. Products of the refining of iron could be kept molten even after the refining was over for a desired duration. The contributions of the Martin brothers in France to the development of such a furnace was more regarding the practical improvements in its design rather than the very notion of such a furnace. However this new process of steel making using scrap as a charge material soon came to be known as the *Siemens-Martins* process. But because of its size and shape it was popularly called the open-hearth process; this name continued thereafter until today. Some of the original features of the furnace, in principle, remained even in the last of its designs. The design underwent several modifications in due time to improve upon its efficiency. These modernizing influences led to nearly mutated designs in which, except only the flat open-hearth shape of the original design, most other things were either changed or modified to suit the current requirements of steel making. These furnaces were given new names such as the Ajax furnace, the Twin Hearth furnace, the SIP furnace, and so on.

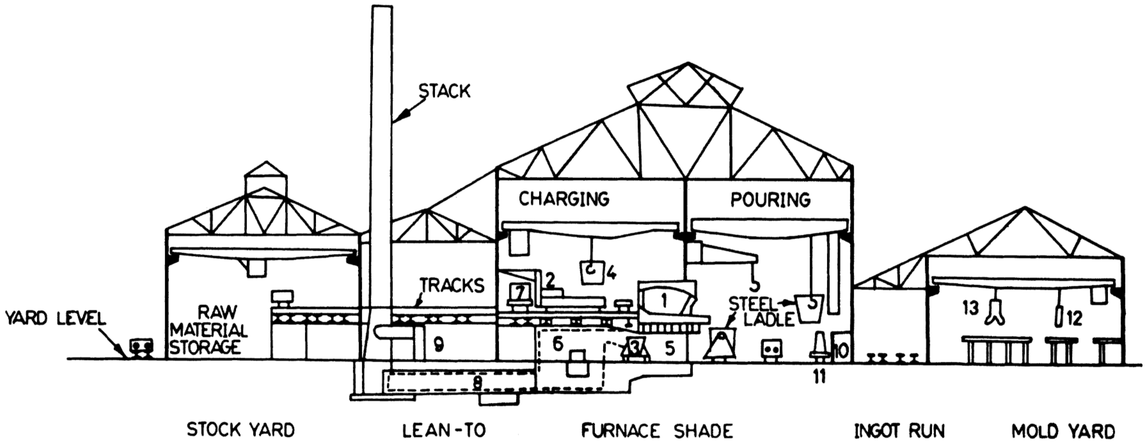
The open-hearth furnace was heated by a gaseous and/or liquid hydrocarbon fuel introduced inside the furnace from either ends through the ports. The gaseous fuel and the air blast for its burning were preheated in the regenerator checkers located at both the ends. The flame sweeps the hearth surface from one end to the other, and thereby heats the slag and the metal. The slag, being lighter than the metal, floats on the metal. It therefore acts as a barrier in heat transfer from flame to the metal.

This system of heating and melting of steel scrap and its refining was described jokingly by illustrating an example. It was like heating water with a candle by holding it above the surface, instead of from below. As if this were not enough, an insulating layer was laid on the water surface before heating. What could possibly be the efficiency of such a heating system? In principle, an open-hearth furnace looked exactly like this. Such a cumbersome and most inefficient furnace should not have lasted for over 100 years, but for the lack of understanding of the principles underlying its design and operation.

The early open-hearth furnaces were of 10-15 t capacity, but over the years the sizes increased to about 500-600 t by the 1960s. Even an 800 t capacity furnace was designed but could not be commercialized in the late 1950s or early 1960s. Although more modern and more efficient oxygen steel making processes had been developed and put to commercial practice, even in India as early as 1956, the open-hearth furnace could not be discarded so readily because of its established quality steel making record of over 100 years. Therefore, even in the late 1950s and early 1960s, open-hearth furnaces were still preferred in India in Bhilai, Rourkela, and Tata Steel (then known as TISCO) plants. Steel plants are very capital-intensive and hence established technologies are preferred in preference to the new ones having no track record.

9.2 Open Hearth Furnace - Design and Shop

Initial open-hearth furnaces were stationary in design and essentially consisted of a flat bath, with the length nearly twice the width and covered by a roof. The two ends were provided with ports acting as burners and exhaust alternatively. Beneath the furnace were laid the heat regenerators in the form of checker works. These features are shown in Figures 9.2 to 9.6, to get the glimpse of its construction. These figures



Thick letters: 1. Furnace. 2. Charging machine. 3. Flush slag thimble. 4. Hot metal crane. 5. Slag pocket. 6. Checkers. 7. Solid charge train. 8. Flues to chimney. 9. I.D. fan. 10. Teeming platform. 11. Mold train. 12. Tar gun. 13. Stripper.

FIGURE 9.2. A vertical cross-section of an open-hearth stop to illustrate the relative locations of various buildings, equipment and material transport.

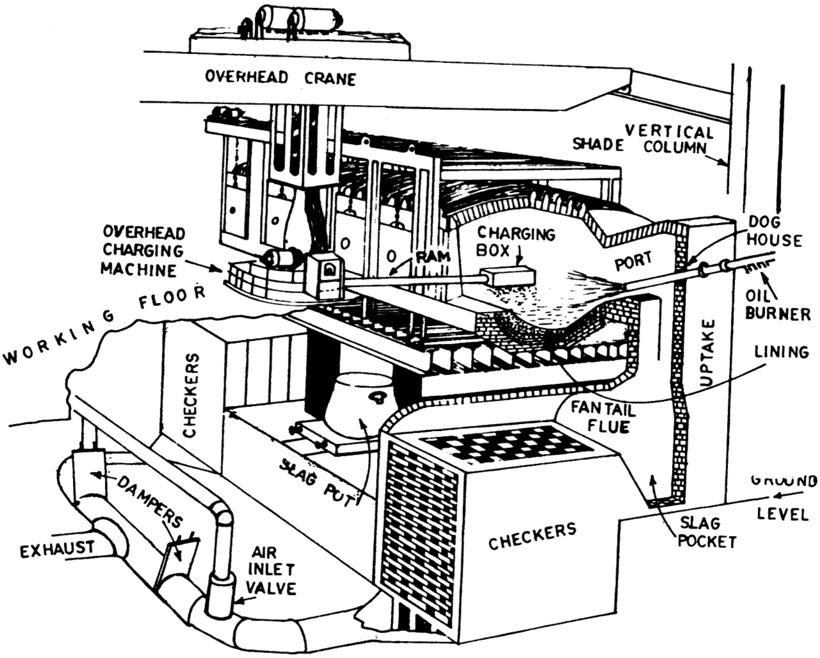


FIGURE 9.3. Sketch of an open-hearth furnace with cutaway sections to show the details of construction, method of charging, firing, etc.

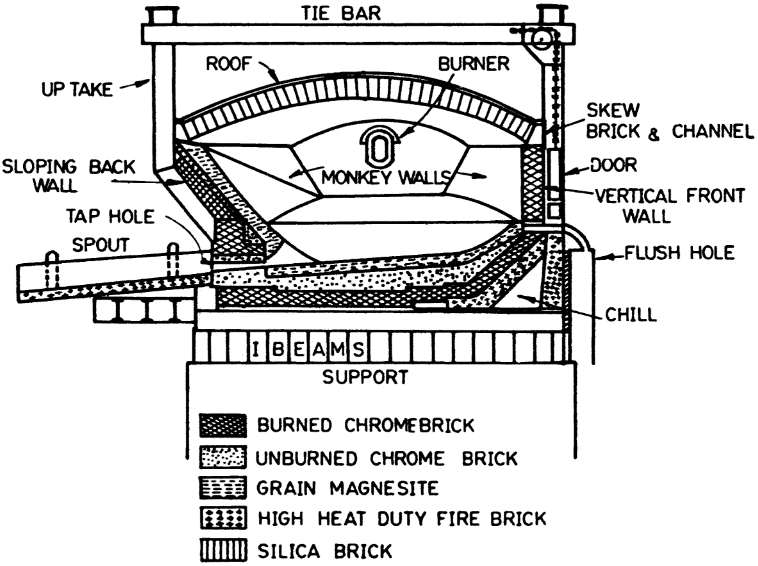


FIGURE 9.4. Transverse section of an open-hearth furnace.

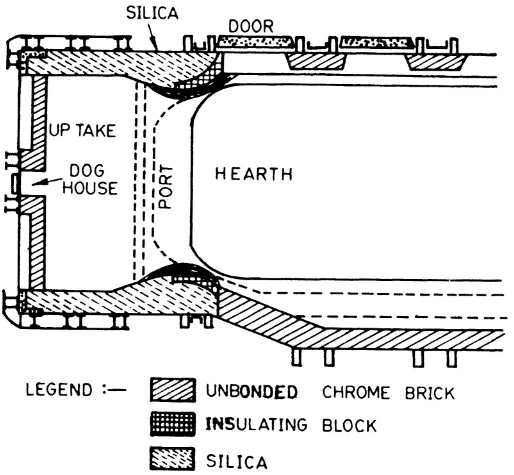


FIGURE 9.5. Plan-section of an open-hearth furnace.

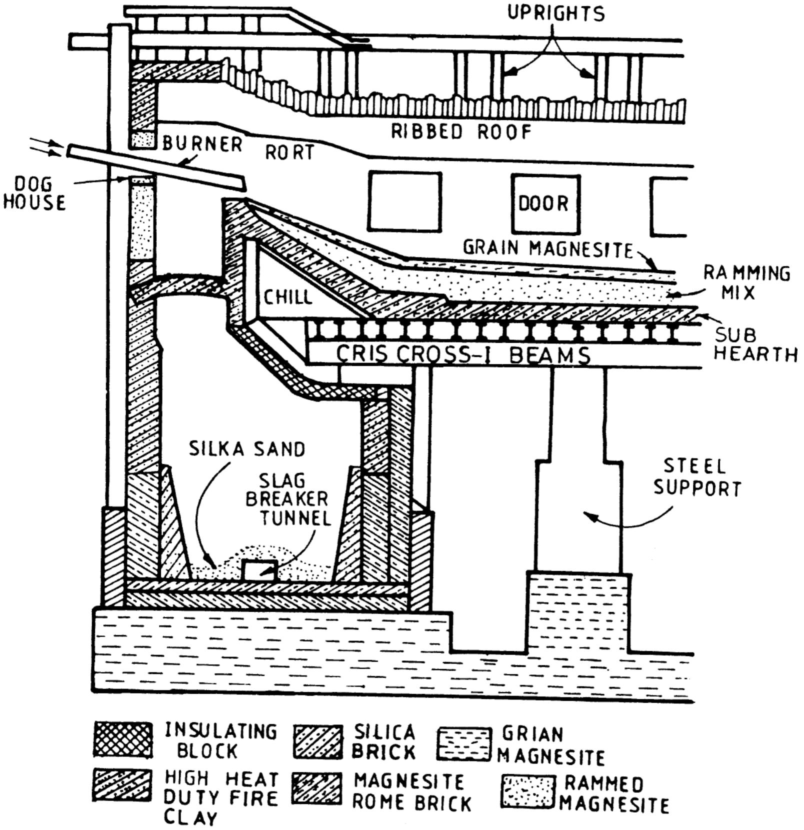


FIGURE 9.6. Longitudinal vertical section of an open-hearth furnace.

represent the stationary type of open-hearth furnace. Tilting open hearths were also built and operated for several decades. These are shown in Figures 9.7 and 9.8.

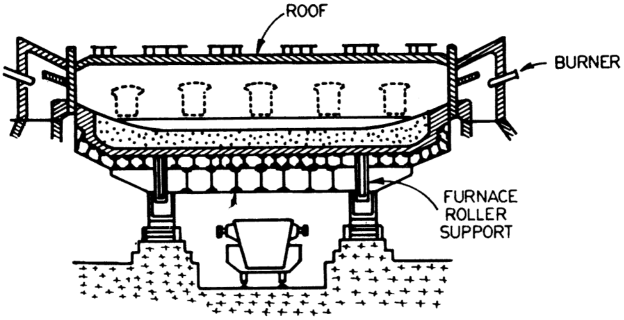


FIGURE 9.7. Tilting open-hearth furnace (longitudinal section).

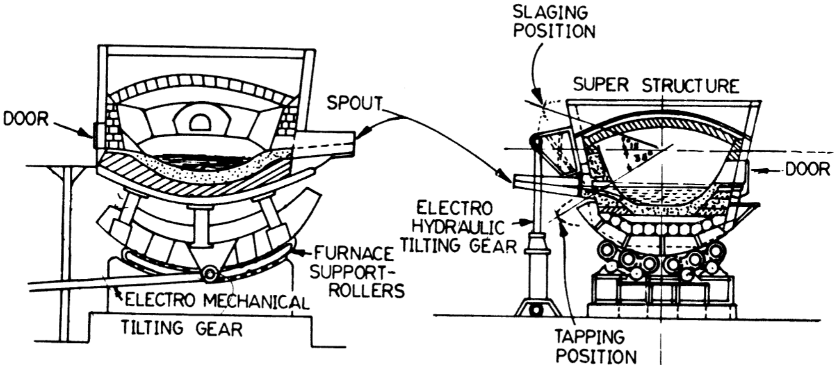


FIGURE 9.8. Tilting open-hearth furnace (transverse section).

The open-hearth steel melting shops were generally long with several furnaces of similar construction and designs laid in a row. The pitside and the charging areas were located on either side of this line. Adequate space was required in front of each furnace to install and manipulate charging machines for solid charges since all solid charges, scrap and flux, were charged horizontally in the furnace through front doors. Several overhead cranes were provided on the furnace line in the shop for handling and charging raw materials. The steel making shops therefore came to be known as steel melting shops (SMS). Incidentally it also means steel making shops, the term currently in vogue.

The tilting open hearths were introduced for steel making for the first time in 1902 in the UK. Their design differs from fixed furnaces with respect to the reaction chamber and the ports. The hearth is much deeper in tilting furnaces. The ports are free of the reaction chamber for effecting rotation of the hearth; the rest of the parts being stationary. The furnace could be tilted 10-15° on the charging side and 20-35° on the tapping side. Furnace gases do escape from the joint-areas of ports and the chamber, hence the thermal efficiency of tilting furnaces was relatively lower than fixed furnaces. For many years the tilting furnaces were built larger than the current fixed furnaces, e.g., when the 100 t tilter was built, the current practice was to build only 30 t capacity fixed furnaces and when 250 t tilter was used the fixed furnace size was around 60-70 t only. The largest tilter built had a capacity of 400 t.

The thermal efficiency of open-hearth very much depended on the thickness of the slag overlying the metal since it acted as an insulating layer in the path of transfer of heat from flame to metal bath. The thickness of the slag layer increased with increasing impurity levels for the given furnace. Generally the impurities increased in the charge with increasing proportion of pigs or hot metal. The open-hearth operation was therefore limited to the use of lower proportion of pigs or hot metal in the charge, or for decreased proportion of these if the impurity level was on the higher side. The tilting furnace could take any proportion of impurity level in the charge and refine it efficiently. The tilting arrangement facilitated removal of slag in part or full and paved the way for making a fresh slag to continue the refining. It also facilitated tapping of part or full metal bath at the end of refining. The leftover metal acted as metal *heel* and when fresh hot metal was charged for the next heat it effectively diluted the total impurity level for refining and thereby helped to maintain low slag volumes and efficient heat transfer to bath.

9.3 Heat Supply

Producer gas was used as a fuel in the well developed early open-hearth furnaces. In an integrated steel plant a mixture of blast furnace and coke oven gas was used as these gases were readily available. In that case producer gas was used as a stand-by or alternatively it was mixed with this mixture. Natural gas was also used at places where it was available. The calorific values of these gases are given below:

Name of the gas	*Approx. calorific value in Kcal/Nm^3*
Producer	1400
Coke oven	4300
Blast furnace	900
Natural	8400

Liquid fuels were invariably used in later years, as the calorific values were higher and in relatively small volumes large heat input was feasible. Many large gas-fired furnaces therefore switched over to liquid fuel firing even by altering the port designs.

The heat from the flame was chiefly transferred to the charge by convection and by radiation. The luminosity of the flame was required to be very high. Use of liquid fuels in preference to gaseous fuels had several advantages:

1. The calorific value of oil was very high

Heavy fuel oils	approx. 10,000 Kcal/liter
Pitch creosote	approx. 8500 Kcal/liter

2. This means volume-wise a very large amount of energy could be fed in the form of liquid rather than in gaseous fuel. The furnace could then be driven hard, i.e., high melting rates were feasible and hence high production rates.
3. Since liquid fuels were not preheated, the regenerator capacity was available for preheating blast only, hence more preheated blast was available for efficiently burning the fuel.
4. Furnace design was simple since only one port and one regenerator were needed.
5. A substantial amount of fuel was saved during reversal period since liquid could be switched off easily; this was not the case with gaseous fuel.
6. The flame was very luminous and long and was under much better control.
7. There was less danger of overheating of the roof and its failure.

Liquid fuels, however, suffered from extra sulfur content, which helped sulfur pickup by the bath.

An oil-fired furnace, of say 200 t capacity, generally needed 4000-5000 liters of oil per hour depending upon the quality of the oil and the mode of furnace operation. It had been found that best open hearths consumed nearly 7-8 Kacl/t of steel produced.

9.4. Thermal Efficiency

When the fuel and air pass through a just-heated regenerator, they are preheated to a maximum level and the flame temperature inside the furnace is maximum. This means that the existing gas temperature is also maximum at that time, and it falls as the regenerator temperature falls with the progress of the cycle. The preheating continues until the regenerator temperature falls to a certain level, and the cycle is then reversed; the same thing happens on the other side. The furnace, therefore, operates between two existing gas temperature limits. These two limits are 1680°C maximum and 1450°C minimum for furnaces with silica roofs. The average temperature is around 1550-1580°C, which is called the critical temperature of the furnace. Since the maximum existing gas temperature is dictated by the refractories that are used, the minimum level shall be decided by the time of the reversal so as to obtain the desired level of critical temperature. If the cycle is prolonged, the critical temperature decreases; if the reversal is quick, the critical temperature increases. An optimum cycle of reversion is fixed for a furnace depending upon its use.

The efficiency of the process can be assessed very simply in terms of the percentage energy, of the fuel supply to the furnace, available in the reaction chamber in usable form, that is, at a temperature level above the critical temperature of the furnace. In a badly designed furnace or wrongly conducted operation, a lot of the input energy may go to waste. Yet the useful part of the energy still remains as a useful basis for assessing the thermal efficiency of the process. The efficiency of an open-hearth largely depends on: (1) the potential efficiency of the regenerators, and (2) the relative amounts and locations of cold air leakages in the furnace system. It has been found to vary from 15-20% to 70-75%. The smaller the checker system and higher the percentage of cold air leakage in the system, the lower is the efficiency. The efficiency value can be increased by designing a big enough checker system and sealing the furnace tightly to keep the cold air leakage to below 5%.

During refining, carbon oxidizes to CO, which can further be oxidized with oxygen supplied in the form of an additional preheated air or cold oxygen. The heat so produced is nearly all useful energy (released above the critical temperature). Even if cold air leakage is directed to the slag surface, about 55% of the CO to CO_2 reaction heat is available inside the chamber. With adequate size of regenerators, therefore, and with a properly sealed furnace it is possible to obtain 30-70% of the heat requirement of the process above the critical temperature level from the oxidation of products of refining, mainly CO. The efficiency values as high as 75% have thus been obtained. For an electric furnace, taking into account the energy losses, in converting chemical fuels to electrical energy, transmission losses etc., the energy efficiency does not exceed 35-40%. The open-hearth can, therefore, be more efficient than an electric furnace, in terms of chemical fuel used as a base.

9.5 Types of Open-Hearth Practices

The open-hearth process could be used both as acid and basic, depending upon the chemistry of the charge materials. The open-hearth practice also varied with the type of metallic charges.

The charge types vary as follows:

1. All steel scrap
2. Pigs (or cast iron scrap) plus steel scrap
3. Hot metal or liquid iron plus steel scrap
4. Pre-refined metal plus steel scrap
5. Hot metal plus pre-refined metal (e.g., Bessemer blow metal)
6. All hot metal from the blast furnace.

Out of the above charges, types 1 and 2 alone were used for the acid open-hearth practice, while all these types were adoptable for the basic open-hearth practices. The early open-hearths were both acid and basic and were meant to deal with all cold charges only and no hot metal was incorporated in the charge. In that also the process started off with only steel scrap as the charge. However it was soon realized that it was economical to use steel scrap and cold pigs, in appropriate proportion, as the charge.

The vast amount of garbage of iron and steel lying unused then could thus be swept and converted into useful steel. In fact, conversion of scrap into finished steel was the sole aim of early open-hearth and it continued to accept any proportion of steel scrap in the charge until its recent demise. It was therefore customary to compare open-hearth with electric furnace in their ability to convert scrap into steel on economic grounds. Under the highly oxidizing conditions of the open-hearth furnace it was not easy to convert all steel scrap charge into steel, whereas under the neutral conditions of the electric furnace it is far easier to do so; hence the superiority of the electric furnace over open-hearth always existed without any doubt.

9.6. Oxidizing Medium

It is interesting to understand how the oxidizing medium was provided and the way it acted in an open-hearth furnace. Steel scrap charged in the beginning was allowed to oxidize to a certain predetermined extent. This not only generated iron oxide but also heat, and when the scrap reached almost an incipient fusion stage the pigs or the hot metal were charged so that the already present iron oxide violently reacted with the impurities in the charge, and much of the Si, Mn, and even some P was thus oxidized. The limestone (not lime) charged at the bottom helped to form the slag initially. After complete melting, lumpy iron oxide ore was used to provide the oxidizing medium; being heavier than slag but lighter than the metal bath it floated at the slag-metal interface and provided the necessary oxygen for refining of mainly carbon and phosphorus. The mechanism of transfer of oxygen from the charged iron ore in an open-hearth is given earlier in Sections 4.3.2 and 4.3.3. The physical nature of the steel scrap played a very important role in this respect and hence was taken into account in designing the refining strategy.

The furnace atmosphere was decided by the combustion of fuel with excess air, and hence it was always oxidizing; but the oxygen from furnace gases had to diffuse across the stagnant nitrogen gas layer on the slag first, then through the slag layer and then through the metal layer to reach precisely the impurity atoms in iron for their oxidation. This was slow and hence it was expedited by putting iron lumps that settled at the slag-metal interface, thereby eliminating the first two steps of oxygen transport. The iron oxide first must dissociate into iron and oxygen, which is a net endothermic reaction, and for which heat is required which again has to flow across the stagnant nitrogen layer and slag layers to reach the required position. All this made the process refining very slow; as a result, the tap-to-tap time in early open-hearths was of the order of 15-16 hours. This continued to decrease with many improvements but it was still around 12 hours when iron oxide was used as refining media even in the later designs.

In those days on-line analysis of metal was not possible as it is today using spectrophotometric analyzers. The carbon analysis was carried out on the basis of fracture of solidified metal sample. Slag quality was estimated on the basis of its flow pattern. The oxide ore addition, known as *oreing* in those days, was all based on the operators' judgments. In the basic process limestone was used as flux instead of limestone as the base had to be calcined before it could flux the oxides and form slag. Hence a carbon level below 0.1% could not be achieved in open-hearths readily. Similarly the bath could not be deoxidized to a great extent, particularly in basic open-hearth practice because of danger of reversion of phosphorus from slag.

9.7. Acid Open-Hearth Practice

The first open-hearth furnace designed and constructed by the Siemens brothers was meant to melt steel scrap that was much more cheaply available than pigs. Very little refining was required in that process. Silica, the usual lining material, was used for lining the furnace. The roof was also of silica. It was an acid-lined furnace meant to refine steel under acid slag. It could not refine P and S and the scrap had to contain each of these below 0.05%. It required Swedish grade pigs as charge and that was not readily available.

The process consisted of charging pigs first followed by the scrap. It was melted at around 1530-1570°C. Refining was carried out by addition of oxide iron ore. The slag had to be silica-saturated to avoid attack on silica lining. The slag was of the type SiO_2-FeO-MnO and with nearly 60% silica, 15% each of FeO and MnO as in the Bessemer process. The silica content of the slag was measured during the refining process using Herty's viscometer. The iron oxide content was estimated by its color and fracture of the sample. The refining was thus very subjectively controlled.

Refining of the heat was temporarily blocked using Fe-Si and Fe-Mn as deoxidizers to temporarily stop the refining. A sample was taken and analyzed and refining started if required, or if the heat was ready it was deoxidized further and then tapped. The carbon level was corrected by anthracite addition in the ladle. A tap hole was opened by oxyacetylene torch to tap the heat.

The acid open-hearth steels had a reputation of being very clean and of superior quality as per standards prevailing in those days. Hence for many special applications the specifications of steels included the process of making of steel as well. The acid open-hearth was also associated with alloy steel making nearly from its inception until the electric furnace gradually replaced it. It was still in use until after World War II. Thereafter it was gradually and completely replaced by the electric furnace process for its superior quality products and production economy.

9.8. Basic Open-Hearth - Cold Metal Practice

Acid open-hearth, even in its heyday, had only a limited applicability for steel making because of its stringent quality scrap requirements and non-availability of hematite grade pig iron. The iron and steel garbage accumulated then often had both P and S above 0.05% and that necessitated its removal during refining. In 1884 basic lined open-hearth furnace was therefore developed and put into commercial practice. The roof was still of silica because of problems in constructing a basic roof, owing to its heavy weight. It however did not interfere with the basic working of the furnace. This soon became very popular because it could accept anything ferrous as charge material without any reservations. These early basic furnaces were of 10-15 t capacities only and could take up only cold charges of steel scrap and pigs. This basic process operated much in the same way as the acid process. Normally the heat in such a process had to be conserved as far as possible, hence the use of limestone was discouraged and use of lime was recommended. Whereas when hot metal was part of the charge, limestone use was recommended. Slag was kept fully basic by addition of lime and adequate basicity was maintained.

The products of this process included all types of plain carbon and low alloy steels with low nitrogen and hydrogen contents. The early success of this process was mainly due to this ability as compared to Bessemer steel quality.

Once it was established as an acceptable process of steel making, it was further developed to accept hot metal in the charge and was made applicable in conjunction with blast furnace to pave the way for installation of integrated steel plants. In due course of time the role of converting scrap into steel was taken over by the more efficient electric furnace.

9.9. Basic Open Hearth-Hot Metal Practices

The necessary engineering required for charging hot metal in the open-hearth was well developed in the early Bertrand-Thiel (1894) and Hoesch (1904) processes. In these, the solid pigs were first melted and partially refined and the resultant charge formed the charge for the subsequent refining process. It was Talbot who, in 1899, developed the practice of using hot metal as the charge. A tilting type open-hearth was used and after every heat some amount of refined steel was left in the furnace as hot heel in which fresh hot metal was poured to continue the refining. It revealed the problems and difficulties impurities posed when the total contents were beyond a certain level in the form of extra hot metal resulting in large slag volumes. The larger the slag volume, the more inefficient became the furnace operation since it hindered the heat flow from furnace gases to the metal, across the insulating slag layer. For the first time it was understood that the slag volume could not be allowed to go beyond certain safe limit since it adversely affected the refining rates of an already slow process . Hence the effect of impurities had to be reduced in some way so that the flux addition remained within limits and thereby the slag volume. The hot metal practice in open-hearth tackled this problem in a variety of ways since the hot metal compositions varied in different ways at different places. This gave rise to a variety of solutions for accepting hot metal in various proportions in the charge, and yet not facing any serious problem of excess slag volumes.

Both stationary and tilting open-hearths were adopted for such hot metal practices. When the hot metal proportion was large, intermediate slag removal became necessary in the early days, which was possible only in tilting open hearths. But later stationary furnace practice was also developed using intermediate slag flush over the door sills to decrease the slag volume accumulating at any time inside the furnace. The inefficient tilting designs therefore were wholly replaced by stationary designs in the later years of open-hearth practice.

This can be understood from the steel making practices adopted at the Tata Steel during the period since its inception. In the early 20th century when it started, the plant adopted both the acid and the basic open-hearth processes for steel making. Since the Indian pig iron contained a high proportion of silicon (1.0 to 1.2%) it adopted the duplex practice with acid Bessemer in conjunction with basic tilting open-hearth practice. After World War II it adopted the stationary open-hearth process with flush slag practice, as it was considered more efficient then. Even the Bhilai Steel plant set up in the 1950s adopted the same process with stationary open-hearth furnaces adopting flush slag practice. The presence of excess silicon along with phosphorus or the presence of high phosphorus contents in hot metal required extra flux addition resulting in extra slag volume that could not be accepted in a basic open-hearth. This problem was aggravated when the proportion of hot metal in the charge increased beyond a certain limit. This problem was solved for open-hearth practice in the following alternative ways depending upon the hot metal composition and its proportion in the charge.

9.9.1. Duplexing

The name itself is indicative of the use of two different steel making processes for carrying out the required amount of refining. This was meant to solve the problem of excess silicon and prolonged refining period. Here the acid Bessemer converter was used to pre-refine the hot meal by eliminating most of the silicon and manganese and part of the carbon; such partially blown metal was charged in the tilting or stationary open-hearth to complete the refining and produce finished steel. The earlier duplex process employed tilting furnaces for its greater flexibility of making and removing the slag. It could thus take almost 100 percent hot metal as charge and produce steel at a very high production rate, which was nearly two and one-half times that obtainable in stationary open hearths using all solid charges. The duplex process was adopted where scrap was in short supply and hot metal was in abundance. It however involved high losses of iron in slag and yield was relatively poor. This was readily accepted even for the continental irons containing high percentages of phosphorus (1.5%). This produced one intermediate fertilizer grade slag as a byproduct. This was possible readily in tilting open-hearth only.

9.9.2. Ladle Desiliconization

When the silicon content of hot metal is very high it can be removed externally in transfer ladle by blowing oxygen and addition of lime as flux to fix up the silica in slag. This is, in principle, a duplex process only, except that the desiliconization is carried out in the ladle itself. The oxidation of silicon and manganese, as is possible in this case, and the resulting slag formation are all exothermic reactions, and hence can be carried out safely in a ladle without danger of temperature drop. The fumes generated herein however need some hood to carry them further for their treatment.

It is the cheapest method of pre-refining since the capital and maintenance costs are very low. Hardly any other equipment is required besides a lancing platform and a hood. This was adopted at the Durgapur Steel Plant in the late 1950's. It had British collaboration and this process proved very successful in Britain at the time. Durgapur hot metal had nearly 1.0% Si in their iron in those days. The metal transfer ladle was of 120 t capacity and oxygen was blown at a pressure of 10 kg/cm^2 through two consumable lances of 24 mm diameter each.

The basic principle underlying this process is to produce relatively less basic slag taking all the silicon as silica in the slag and flushing it out. This rest of the refining was carried out in the open-hearth furnace.

The basic limitation of open-hearth or its inability to deal with the extra load of impurities led to the development of a series of new steel making strategies after World War II. One was the flush slag practice. The silicon was reduced from 1.0 to 0.4% producing a slag which analyzed as 42% SiO_2, 20 % FeO,, 19 % MnO, 1 % MgO, and negligible phosphorus oxide. The temperature of the metal increased up to 200°C depending upon the percentage of silicon oxidized during the refining.

In this process the lance could not be inserted more than 30-40 cm below the liquid surface. As a result the top layers were desiliconized, leaving the bottom metal relatively much less desiliconized. The temperature at the top layers was 200°C more than the bottom layers.

9.9.3. Active-Mixer Practice

In contrast to the inactive mixer, that is normally used between the blast furnace and the steel making to ensure a smooth supply of hot metal to steel making, the active mixer is one in which some metal refining is carried out using some oxidizing refining slag. Since metal stays in the mixer for some time, it might as well be pre-refined during that period to some extent particularly with respect to its silicon content. The mixer lining has to be basic in nature to accommodate slightly basic slag that is maintained inside the mixer. By this means the silicon was controlled to around 0.5%, which was acceptable for the open-hearths.

Pre-refining in an active mixer is very quiet as compared to violent reactions in the desiliconization process in the ladle. Hence it is under much better control.

9.9.4. Flash-Slag Practice

The thermally very inefficient operation of tilting open-hearths led to the development of a sort of duplex practice in stationary open-hearths.

The idea was to prepare slag of relatively low basicity in the beginning to oxidize silicon and manganese, and to allow this slag to flush off over the door sill by a gentle carbon boil. The bulk of the slag was thus prevented from accumulating in the furnace, and thereby affecting the heat transfer adversely. In this way much of the silicon was eliminated during the flushing, and the phosphorus was eliminated thereafter by preparing another highly basic slag by fresh additions of lime. Iron ore was used as a refining medium. The door sill, particularly of the central door, was adjustable with respect to its height in this practice.

The flush slag technology, in principle, developed in open-hearth practice to avoid buildup of slag inside the furnace during the refining operation of high impurity level hot metal, is very much in use elsewhere, particularly in electric are furnaces, even after the nearly complete demise of open-hearth. The essentials of this practice must therefore be understood by the steelmaker of today even after the adoption of most modern steel making technology.

9.10. Modified Open-Hearth Practices

The faster oxygen blowing steel making processes that were developed and put into commercial practice after World War II, in the 1950s, influenced the open-hearth practices to the extent that they had to develop and modify practices using oxygen for refining. The influence was so tremendous that the existing furnaces made provisions for oxygen lancing and altered the designs, to the extent possible, to improve the practices.

In steel making, oxygen can be supplied in the form of air, iron ore, enriched air and oxygen for refining. In open-hearth, oxygen is also required for burning the fuel to supply the required thermal energy for the process. The oxygen requirement of the bath in a basic open hearth is determined by the quantity of hot metal or pigs in the charge. For a typical 60/40 charge hot metal practice, nearly 60 kg of oxygen is required to refine a ton of charge and additional 300-400 kg of oxygen is required for combustion of fuel. The manner in which these two requirements are met determines the rate of production, and it is in this respect that the open-hearth practice had to be revolutionized.

In the conventional open-hearth practices the oxygen requirement is met from hot blast and solid iron oxide added as charge and as feed ore. In the days when oxygen was costly, it was used for decarburization of the bath of low carbon levels to produce soft steels as this could not readily be achieved by addition of ores. As the price of oxygen decreased further it was used for decarburization of below 0.3-0.4 % C, if otherwise economically permissible. As it became quite cheaper and abundantly available, for oxygen steel making, it was used for combustion and refining in open-hearth practice. In the open-hearth practice, therefore, soon after the addition of hot metal, oxygen lancing was started without waiting for melting, to take full advantage of the exothermic reactions of refining for concurrent melting.

In the early open-hearth practices using oxygen for refining, it was introduced through consumable lances introduced in the furnace through doors. However, soon they were replaced by the standard water-cooled lances introduced in the furnace from the roof vertically, either one in the center or three equally spaced over in the roof in the longitudinal direction.

9.10.1. Consumable Lance

This is the simplest arrangement in which a common steel pipe, 5-6 m in length and 2-3 cm in diameter, is inserted in the furnace through the wicket hole. The other end is connected to the oxygen supply hose. The pipe is either held manually or in a holding rack as shown in Figure 9.9 and

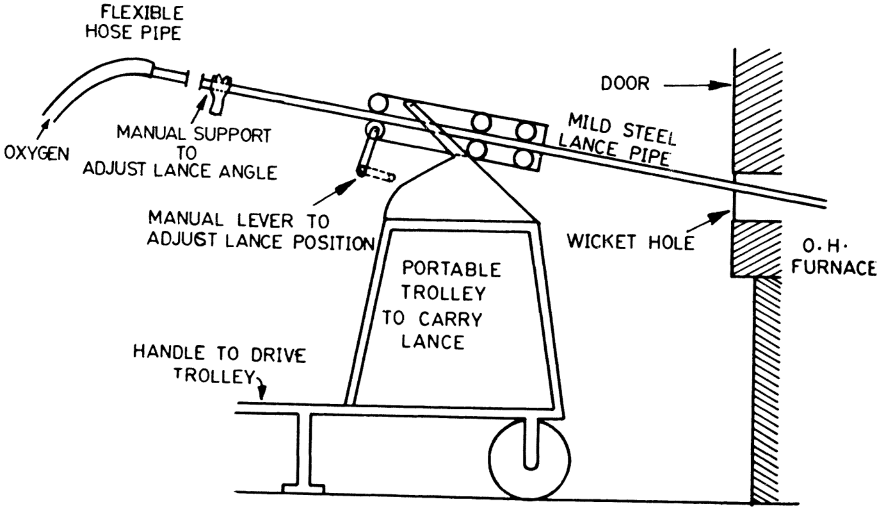


FIGURE 9.9. Manually operated consumable lance for oxygen blowing. It is inserted in an open-hearth furnace through the wicket hole.

is maneuvered manually. An oxygen flow rate of 15-20 m^3/minute cools the pipe and gives it a reasonable life. The pipe may be coated with a refractory or is aluminized to prolong its life, particularly if lower flow rates of the order of 3-7 m^3/minute are employed. A consumable lance is used to blow oxygen after the addition of hot metal in the furnace to hasten decarburization of the bath.

It is the cheapest method of oxygen lancing of the bath and using multiple pipes can blow up to 100-120 m^3 of oxygen per minute without causing excessive fuming and splashing of the slag. Oxygen at low pressure can also be blown by consumable lance with equal ease. The tip of the pipe, however, gets consumed and needs to be adjusted in the blowing position almost continuously throughout the blowing period. It needs frequent replacement as well. It also interferes with movements of the charging machine and cranes on the shop floor.

9.10.2. Water-Cooled Lance

The disadvantages of a consumable lance can be eliminated by using a water-cooled lance like that used in LD, Kaldo, and like processes. The tip of the lance is made of copper and may have one or more nozzles. In general multi-hole lances are preferred to single-hole lances because of their higher ability to distribute oxygen over a much larger area, thereby causing less fuming and splashing. It is shown in Figure 9.10. In whatever position the lance is inserted it has some disadvantages. The back wall installations were totally abandoned because of their interference with crane movements on the pitside. The lance is more commonly introduced vertically through the roof as shown in Figure 9.11. The arrangement is found quite satisfactory and can be installed on an existing furnace without much alteration in the roof structure. Usually two or

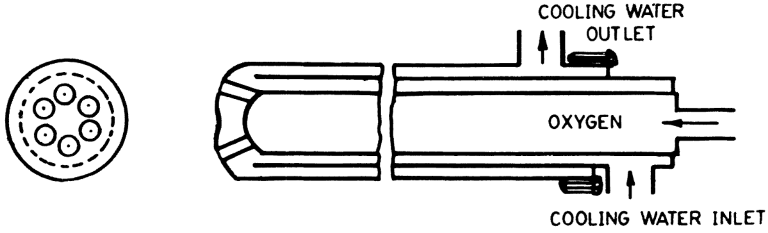


FIGURE 9.10. Water-cooled oxygen lance for hearth processes.

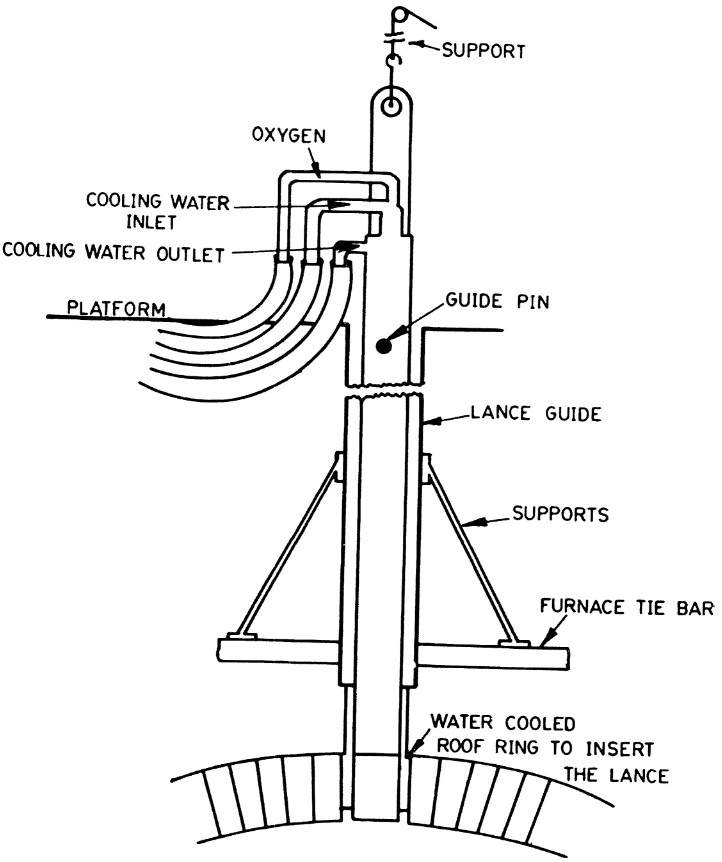


FIGURE 9.11. Details of insertion of a water-cooled lance through the roof of an open-hearth furnace.

three lances are used and are evenly located on the long axis of the furnace roof. The arrangement does not interfere with any other usual movements in the shop. The position, condition, and operation of the oxygen jet can be observed through an open door from a safe distance on the floor. The lance is raised and lowered by an electrically operated winch, and the movement and oxygen flow rate are controlled from the furnace control cabin. The safety arrangements usually provided for lances in LD, Kaldo, and similar processes are provided for the open-hearth lances as well. A water-cooled box or a refractory ring is fitted in the roof to allow insertion of the lance in the furnace.

As the lance is progressively lowered, the amount of splashing increases to a point, and then decreases to a point of least splashing when the tip is just below the slag surface. This is the correct position of the lance. It usually supplies oxygen at the rate of 12-15 m^3/minute to keep fuming and splashing of slag to a minimum. Excessive fuming tends to clog the checkers and excessive splashing of oxidizing slag causes the roof and the walls to wear rapidly. Some of the furnaces do blow oxygen at the rate of as high as 40 m^3/min. It has not been possible to design a lance to blow more than 50 m^3 per minute without the splash becoming a serious problem.

Initial applications of roof lances were on furnaces with a silica roof. The time saved in accelerating the refining, however, was generally lost in additional repairs required due to severe corroding action of fumes and splash generated by lancing. As a result, basic roof and end walls were adopted as a matter of routine practice for adopting roof lances. The basic roof also allows accelerated melting of the solid charge. The basic roof height was increased to reduce slag and metal splash reaching the roof. A gas cleaning plant became necessary to clean the fumes generated by bath lancing.

The lances are lowered immediately after the pouring of hot metal and blowing begins. As the CO generation attains good intensity the fuel may be reduced or at times completely shut off. The heat is generated due to the exothermic oxidation reactions below the slag layer, i.e., it need not flow across the insulating slag layer as in a conventional process. The slag, therefore, need not be so hot and so high in iron oxide content as in a conventional process. This reduces the attack on the refractory lining of the hearth in terms of severity and duration since refining time is now reduced.

If oxygen is not available at the required pressure of 5-6 atmospheres, spraying of low-pressure oxygen on the slag surface is also quite beneficial during refining. It increases the supply of oxygen to the slag by eliminating the nitrogen blanket on the slag, which hinders the oxygen flow in a conventional practice. The spraying of oxygen can burn CO evolved during the boil to CO_2 and keep the process nearly autogenous during refining.

In small furnaces oxygen lancing is carried out at low pressure only. There are not much fumes and hence no gas cleaning is necessary. Oxygen fed through the roof lance is also being used to accelerate combustion in the flame. Usually two (or two pairs) are used, one at each end; one is used at a time from the end in which fuel and air are being admitted, the other lance being withdrawn at this time. The fuel, air, and oxygen are all reversed together. The position of the jet and the rate of oxygen supply are adjusted to release the heat very close to the cold charge and thereby expedite melting. The scrap can be piled up at the hot zone on either side of the furnace hearth. The exact location of the lance varies considerably in different designs. Some operators prefer this method of introducing oxygen to that of introducing it at the fuel input point. In general the saving in fuel and accelerated melting should more than compensate for the extra cost of oxygen and the loss in yield, if any. Some believe that it is economical to allow over-oxidation of scrap by lancing as a source of heat to assist melting. There is however no basis for this being so.

Roof lances have also been used as an additional oxy-fuel burner to assist melting. The lance is invariably a multi-nozzle lance to increase the effective area of heating. It is, however, yet to be conclusively proved that this method is better than supplying the fuel and oxygen through the ports. In any case it may only be useful if the scrap proportion is very high in the charge.

In either of the above cases, if the lance is used to accelerate melting, the same lance is later used to blow oxygen for decarburization of the bath. The number and locations of the lances should be decided in the modified design from the point of view of their dual use.

9.11. Products of Basic Open-Hearths

The introduction of the hot-metal open-hearth practice enhanced the production rate by nearly 30-50% over that obtainable from basic open-hearth cold metal practice. It is unrivalled in its ability to take any proportion of hot metal, pigs and steel scrap as a charge and yet produce a very wide range of steel products with better cleanliness and particularly low gas contents. The process was therefore very extensively adopted in integrated steel plants in the 1950s. By this time oxygen-assisted, oil-fired and all basic furnaces accounted for nearly 90% of the total steel production. Productivity of the order of 20 t/hour was then achieved. Around the same time, productivity of the Thomas process shop (on the Continent) was of the order of 100 t/hour. This was tough economic competition for open-hearths then. Around the same time the newly born oxygen steel making processes posed additional competitors for open-hearths. The BOH practice therefore had to be modified by using as much as possible, oxygen for refining, while maintaining its quality standards and improving upon its productivity.

9.12. Characteristics of Open-Hearth Practice

One must understand the importance of the open-hearth practice while it was the dominant steel making process against the available alternative processes. It was therefore always compared with the Bessemer/Thomas processes and against them it was far more superior in terms of quality of the products. Open-hearth steels contained very low hydrogen and nitrogen contents as compared to the rival processes. Being very slow, it was also under much better control, as compared to the rival processes. The open-hearth steels were therefore more preferred as compared to the Bessemer/ Thomas process products. These were especially preferred for producing flat products because of their especially lower hydrogen and nitrogen contents. Open-hearth steels were also preferred on account of better cleanliness since at least part of deoxidation was carried out inside the furnace, improving the elimination of the deoxidation products, making it cleaner.

9.13. Later Day Open-Hearth Practices

After the successful adoption of fast oxygen steel making processes in practice the existing open-hearth shops had to withstand competition from them. This pressure led to reduce the tap-to-tap time from earlier around 16 hours to nearly 8-10 hours. Some more efficient furnaces attained even less than this time. The major changes that were responsible for the spectacular increase in production rate are given below:

(*i*) Rapid material handling and repairs facilities and their proper scheduling in the entire multi-furnace shop.

(*ii*) Fast melting of cold charges using oxygen for burning the fuel.

(*iii*) Improved furnace design and refractories to allow faster working at higher temperatures.

(*iv*) Rapid refining, using oxygen lances for refining and concurrent melting.

(*v*) Efficient control and charge balancing (both thermal and material).

In the modified practice, oxygen blowing commenced even before the scrap was molten because the exothermic heat of refining of hot metal was utilized for melting. The lance was progressively lowered until a point of least splashing when the tip was just below the slag surface.

Usually 12-15 cu. m/min of blowing was just correct, giving a minimum of splashing. Excessive fuming resulted in clogging of the regenerator checkers and excessive splashing of oxidizing slag eroded the roof quickly. Some furnaces, however, did blow up to 40 m^3 per min without any serious problem.

The stirring of the slag and metal baths by the usual carbon boil taking place at the bottom was replaced by the top-oxygen blowing action. This increased the rate of supply of oxidizing medium manyfold and, thereby increased the rate of the refining.

The importance of scrap oxidation in the open-hearth practice should be understood carefully. It produced iron oxide that reacted with the hot metal charge very vigorously, resulting in early good quality slag formation. This oxidation also produced heat to hasten the preheating of scrap. The excess iron oxide so produced was required to keep slag with adequate iron oxide content for subsequent refining of impurities in particular phosphorous of the hot metal.

All these modifications in open hearth design and practice led to obtain a production rate as much as 115 t/hr from a set of three 500 t open hearths, i.e., 19-20,000 t per working week. A hot metal charged 450 t furnace produced nearly 7500 t per working week in 1965, and the same furnace with intense oxygen lancing produced 16,000 t/working week. All these improvements resulted in a tap-to-tap time of nearly 4-5 hrs in a most efficient open-hearth shop. These figures indicate that the total effect of all other innovations and changes made in open-hearth design and practice during its entire history was far less significant than what would be achieved by simple intense oxygen lancing. With all this the LD/OBM was not considered with less suspicion and hence, as an alternative, different hearth furnaces, known differently by their designs, were put into commercial practice, to withstand competition from oxygen steel making.

With all the modified versions, it could not stand competition from the upcoming LD or the OBM processes in terms of productivity, and this finally led to its demise.

CHAPTER 10

RISE AND FALL OF STEEL MAKING TECHNOLOGIES-III RECENT HEARTH PROCESSES

In This Chapter

- Ajax Process
- Tandem Furnace Process
- Continuous Steel Making Processes
- SIP Process
- EOF Process
- Twin-Hearth Process (Dual-Hearth Furnace Process)

During the two decades after the first oxygen steel making process was successfully put into commercial practice, the existing open-hearths were hard put to run in competition with them. Many design alterations were suggested and put into practice in the existing shops to improve upon their productivities and efficiencies. It is essential to know these mutated open-hearth designs to understand how steel making technology was continuously improved in the existing plants to stay in competition with other newer technologies. Such a situation is always faced by the steel plants because once any steel making process is adopted it can't be discarded, just because some other better process has become available. That is the reason why the choice of a steel making technology is always made, looking into the possibilities of any other process under development that is likely to come up soon on the commercial level.

10.1. Ajax Process

The difficulty in introducing oxygen lances through the roof of a tilting open-hearth furnace, as was possible in stationary furnaces, led to introducing them through the port ends. The exploratory work was carried out on existing furnaces. Since it proved to be practically feasible and economically viable, several other modifications in design were also incorporated in the new installations. The process was developed at the Appleby Frodingham Steel Co., in the UK in the late 1950s, and several such units were installed thereafter at different places. This process was popularly known as the Ajax process and the furnace employed as the Ajax furnace. It was essentially a tilting open-hearth furnace and is shown in Figure 10.1. The end walls, ports, down takes, slag pockets and checkers were suitably altered keeping the general design of open-hearth furnace in view. Therefore it still looked outwardly as open-hearth only.

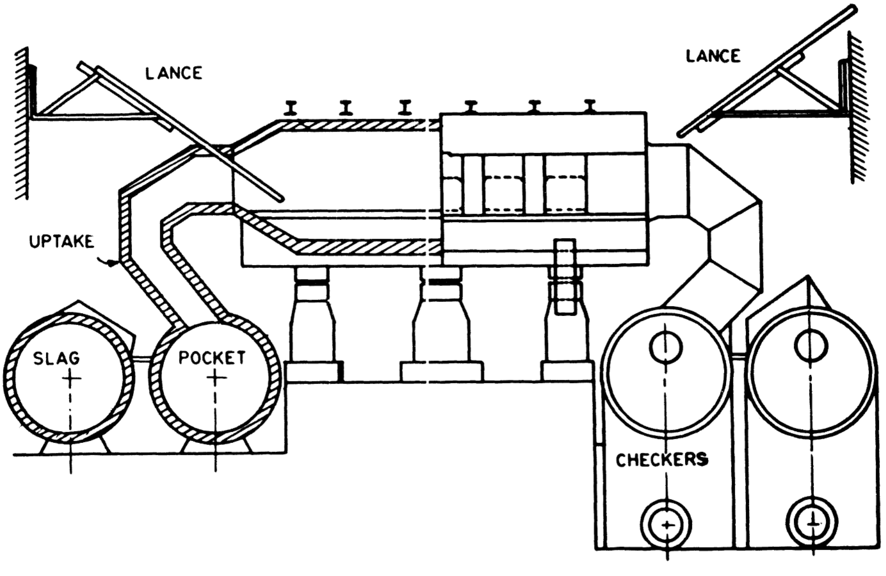


FIGURE 10.1. Ajax furnace. A separate sets of slag pockets and checkers are also shown along with.

The down-takes, slag-pockets and checkers were cylindrical in shape rather than rectangular shape adopted in open-hearth. Each one of these was encased in cylindrical steel shells with appropriate openings and flanges for fastening them with each other in place. The slag pocket had its axis in horizontal and the checker as vertical direction. A spare set of slag pocket and checkers were provided on each side so that at any time one is in service while the other is being repaired. The furnace campaign therefore depended only on roof and hearth life in spite of the fact that lot of fumes generated during oxygen refining led to clogging of the checkers. Gas cleaning facilities were provided with the furnace.

The furnace was fired with coke oven gas through a pair of burners inserted on either side. One water-cooled oxygen lance was provided on each end. It was used alternatively in conjunction with the direction of fuel firing and was completely withdrawn when not blowing. The lance was held at an angle of 27-34° to the surface during blowing and usually had three nozzles as was popular for oxygen lance then. Oxygen gas was blown at 5-6 atmos pressure and at a flow rate of about 30-35 m^3/min.

Scrap was charged in the furnace and was preheated by fuel firing as in the open-hearth practice. Hot metal was poured in and fuel supply was put off. Oxygen lacing was immediately started from one end by lowering the lance inside the furnace. At the end of usual fuel cycle time it was withdrawn, and the one on the other side was inserted and oxygen lancing continued. It works alternately.

The process was meant to refine Thomas grade iron. The blowing therefore continued until carbon drops to 1% or so and phosphorus to 0.1-0.15%.

By this time, the fertilizer grade slag was ready and was removed. Fresh lime and ore were charged to continue the process and to form the second slag. Blowing was continued again until carbon was down to 0.4-0.5% and bath temperature around 1570°C. At this stage, oxygen lancing was stopped and the heat was finished as in conventional open-hearth practice.

It took over three hours of lancing for a 200 t furnace and usually 2.0-2.5 hours were required for charging, de-slagging, de-oxidation, tapping, etc. It means tap-to-tap time was of the order of 5-6 hours.

The process was fast enough and at the same time retained all the desirable features of the open-hearth practice so that it could make a very wide range of steels. It was mainly developed and used for Thomas grade iron but it could take all scrap charge. The tilting design helped to remove the slag in between when required or a part of the liquid steel as per requirements readily. It resulted in increased production rate up to 70-100% over similar capacity standard tilting open-hearth practice. The cost was reduced to 60-70% of the standard open-hearth practice. The chief drawback was that the iron content of the slag was between 15-20% as against the 9-10% in standard open-hearth practice.

10.2. Tandem Furnace Process

When it was realized that nearly 50% of the available exothermic heat of the steel making was in fact wasted, and that it added to worsening of the conditions of working around, every attempt was made to recover this heat for useful steel making operations. The rotor process was originally designed to utilize the heat of burning of CO to CO_2 reaction for melting scrap in the process. This practice had to be discontinued because of heavy refractory erosion arising out of this and consequent extra downtime for repairs. This did not deter the investigators to find alternative ways of utilizing this heat. This then led to the development of the tandem furnace process where this principle was more successfully and efficiently utilized. It was put into commercial practice at Vanderbijlapark South Africa in 1965. The schematic of the furnace design is shown in Figure 10.2.

Two identical tilting open-hearth furnace hearths were put side by side and were inter-connected via a throat as shown in the Figure 10.2. The

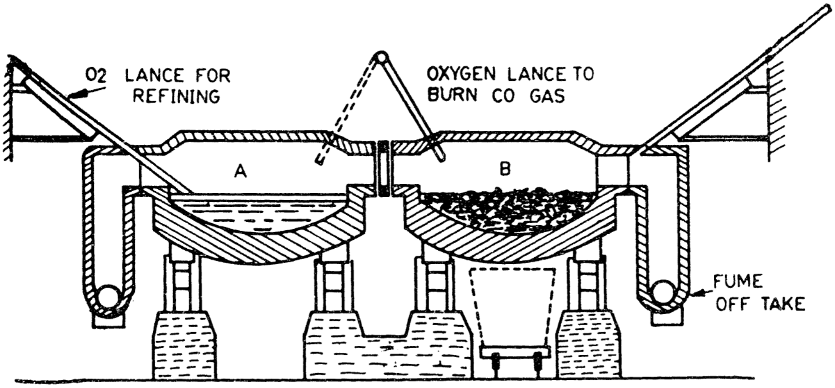


FIGURE 10.2. Tandem furnace. In vessel A refining is going on and the evolved CO is burned in B to melt scrap.

outlets on the other side were directly connected to the fume offtakes. Since neither gaseous fuel nor hot blast was needed in this process, the checkers were altogether eliminated thereby saving in cost of its construction. One oxygen lance was provided on each furnace at the outer ends for blowing oxygen during refining as in the Ajax process. In addition one lance each was provided at joint-throat of the two furnaces for each furnace to burn the CO gas evolved during refining. When refining was going on in one, the burning of CO was carried out in the other furnace with the help of the throat-lance, while the other throat-lance was put off. The lances were inserted and withdrawn independently.

These two furnaces were operated out of phase with respect to each other. When refining was going on in one, the evolved CO in that chamber was burned at the throat of the other chamber by supplying oxygen through the combustion lance. The heat thus released was used in preheating the solid charge that had just been charged in that chamber. When refining was over in the first chamber, the oxygen lance was withdrawn and the heat was finished and tapped as in the open-hearth practice. During this period the charge in the other chamber was preheated and molten metal was charged to make it ready for bowing oxygen for refining. The cycle was repeated.

For a two 110-ton furnace pair, tap-to-tap time of two hours on each furnace was readily achieved.

The process proved to be very prospective since it was completely autogenous and accepted 35-40% scrap in the charge. It was fast enough and retained all the advantages of an open-hearth practice.

10.3. Continuous Steel Making Processes

In the 1960s and 1970s there was a craze to develop metallurgical processes as continuous operations like the general chemical engineering operations. Because of their mostly room temperature reactions, processes are designed to run as continuous operations. This is the main reason why they refer to the metallurgist as bucket-lifters. Because of their major operations being at very high temperatures, the metallurgical operations are carried out in batches involving lifting of ladles. This was accepted as a challenge to convert even the metallurgical high temperature operations like steel making into continuous operations. The BISRA, IRSID, and others did develop pilot plants of continuous steel making.

The IRSID process was developed in France to produce 100 t/hour steel continuously and were confidant enough to install it on a commercial level. Similarly the WORCRA process was perfected to produce 5 t of steel per hour. They could not however, instill confidence of the steelmakers to accept these processes because of some inherent difficulties like producing different grades by changing the chemistry on request.

In the meantime, very large LD vessels like 400 t capacity were installed and put into production successfully to meet the rising steel demands. Confidence in the LD process had been increasing exponentially and that finally put to rest all the attempts in developing a continuous operation; but because of its extremely high decarburization rates, the only process that was installed on a commercial level was the spray steel making [15] process. It was put into commercial operation in 1966 primarily not because of its continuous process that was pretty well designed, but from the point of view of mass and heat transfer involved in refining. At one time it worked well in three different commercial plants in the UK. But all of these were soon scrapped against the competing hybrid processes wherein the refining rates were pretty high and even near-equilibrium conditions could be attained.

10.4. SIP Process

It was easier to install water-cooled oxygen lances piercing through the open-hearth roof vertically or through the ports in an inclined position. These could be readily inserted and withdrawn as per requirements during the refining operations.

The alternative position of introducing oxygen in open-hearth was through the back sloping wall in stationary or the tilting open-hearths. In this way, oxygen was introduced at the usual slag-metal interfacial area or more directly into the metal bath below the slag level. This process became known as the submerged injection process, or SIP. This was developed by the Sydney Steel Corporation, Canada, where a 200 t tilting open-hearth was converted into a SIP design in the late 1980s. The tuyeres were located nearly 300 mm below the usual slag-metal interface.

KORF Technologies Inc., USA, also designed a similar system and marketed it under the name KORF System (KORF Oxy-Refining Fuel, System). These were adopted in Brazil, USA, Hungary, and in India at Tata Steel, on 15-200 t existing open-hearth furnaces and claimed to have successfully operated with distinct advantages over normal open-hearth practice.

In this process, oxygen was blown through a horizontal submerged tuyere fitted in the back wall. It somewhat resembled the OBM process or World War II-fame side blow converter. It definitely improved the metallurgical reactions by stirring the metal phase.

The exothermic heat of refining generated at the tip of the tuyere proved to be useful for melting scrap. The thermal efficiency therefore improved as compared to standard open-hearth practice. The advantages claimed were:

(i) saving of up to 80% energy
(ii) increase in productivity by about 100%
(iii) reduction in refractory consumption by about 20%
(iv) increase in yield up to 2%
(v) better temperature control
(vi) better coordination between furnace and casting operation
(vii) improved steel quality
(viii) better flexibility in charge proportion of hot metal and scrap.

In a given open-hearth shop the adoption of SIP or the KORF technology meant almost doubling the production rate. The life of hearth, walls, ports, chambers, etc. remained practically the same. The tuyere wear was around 3-6 mm per heat and could be repaired. The maintenance of submerged tuyeres was, however, a big headache. The maintenance problems in tilting furnaces were relatively fewer when compared with the stationary furnace. This was purely because of flexibility of getting the tyueres above or below the melt surface with ease by merely tilting even during the processing of a heat.

With all these advantages the experiences at Tata Steel in using this technology in their stationary open-hearths of SMS 3 did not bear the desired improvements and economy in steel production. This is perhaps one of the reasons why they finally changed their SMS 3 into LD Shop No 2 in 1993. Two of these open-hearths were retained to take care of the surplus scrap left over after running the LD shops. These were also replaced later by LD Shop No 3.

10.5. EOF Process

This process made a later appearance on the steel making horizon. The full name is the energy optimizing furnace process. It was also developed by the KORF group at their Siderurgica Pains plant at Divinopolis state of Minas Gerais in Brazil. It was originally developed to replace the open-hearth furnace without losing its flexibility in accepting any proportion of hot metal and scrap in the charge. Ever since the inception of oxygen steel making, attempts have been made to utilize the chemical heat of CO evolved during refining for the process itself. The Bessemer process could only watch this CO burning at the mouth of the converter in the form of a long luminous flame in open atmosphere, thereby not only losing all its thermal value but also making conditions around the steel making furnace more unbearable to people around it. The Kaldo process to some extent and the Rotor process of steel making attempted to recover this heat for scrap melting by burning CO inside the furnace in part or full. It however had to face severe refractory wear problems to the extent these processes could not compete with LD in terms of overall economics of steel making. These processes therefore appeared on the steel making horizon like a comet and disappeared.

In the LD process modification all the CO was collected and cooled and then used elsewhere in the plant as high calorific value fuel. This was called the OG modification of the LD process.

This innovative drive led to the development of the EOF process wherein the CO evolved during refining was used successfully for heating the scrap. The system first developed by Bengal is shown in Figure 10.3. It has however undergone several changes and improvements since then. It can now take a cent percent scrap charge or a mixture of hot metal and scrap in any proportion.

The EOF furnace consists of a reaction/refining chamber with provisions to inject oxygen below the molten metal bath surface for refining and above the bath surface to burn CO inside the furnace. It allows a part of this heat to be used for refining as such, and the other for heating scrap placed on water-cooled withdrawable bars. This arrangement helps to very efficiently transfer the heat of the furnace gases to scrap and leave the furnace as off-gas at very low temperatures.

It has been observed that the furnace gases pass off the refining zone at around 1300-1350°C and then preheat the scrap to 800-900°C before

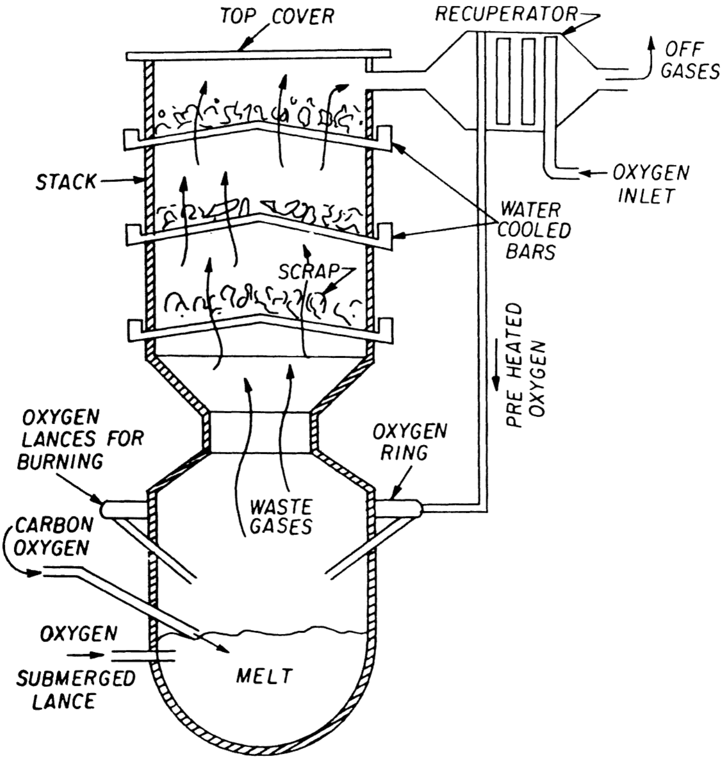


FIGURE 10.3. Schematic of EOF process.

leaving the furnace at around 350°C. This remaining heat is used in preheating the incoming oxygen in recuperators, to achieve high thermal efficiency. The preheated oxygen is used for blowing as well as burning the off-gases.

The process has a provision to blow carbon and oxygen to melt the preheated scrap when almost 100% scrap charge is used. This indicates the possibilities of use of chemical fuels for steel making, which principally was later exploited in modifying the EAF practices.

The conventional EOF practice was meant to process 60% hot metal in the charge. The preheated scrap is dropped in the furnace in stages and hot metal is poured immediately thereon. The carbon content is thus adjusted to take care of the proper heat balance of the process.

When a high proportion of scrap is to be used, a hot heel from the previous heat is maintained to effectively adjust the scrap proportion in the charge. The higher the scrap, the higher is the proportion of the hot heel. The operation in this case is slightly altered. It starts with injection of carbon in the hot heel through special tuyeres until a carbon level of nearly 3% is achieved in the bath. This is followed by preheated scrap charging and the blowing follows immediately thereafter. The refining as well as burning oxygen are blown simultaneously.

In both of these types of operations the slag is foamy and is allowed to flow over the sill down into the pit. The process is controlled through control of the rate of blow of oxygen through both the tuyeres. It is a quick responsive control strategy and hence amenable readily to process control systems.

The refractory consumption is kept within limits by using water-cooled panels and scrap supporting bars. The maximum and minimum permissible size of the scrap pieces and their shapes are predetermined to increase the furnace life as well as its efficiency by way of its preheating.

The EOF owes its high thermal efficiency to optimized use of:

(i) thermal energy of CO evolved in refining for use in refining as well for scrap preheating.

(ii) heat of other exothermic reactions of refining in raising bath temperature.

(iii) designing the furnace main body in a circular shape to minimize heat losses to surroundings.

(iv) fixed non-tilting design enabling the use of water-cooled panels, water-cooled doors, etc.

(v) provision of auxiliary fuel burner to compensate for additional heat, if required.

(vi) use of KORF submerged injection tuyeres with adequate life.

(vii) preheated oxygen for blowing and burning.

(viii) burning of furnace gases at variable angles to effect heat transfer.

(ix) computer controlled oxygen flows.

(x) removable bottom of the furnace to decrease downtime.

(xi) better flow design to minimize infiltration of unnecessary air as leakage into the system.

The Tata Steel did set up an EOF of 80 t capacity in their SMS 3 for a rated production of 1600 t of molten steel per day or an equivalent of 0.5 Mt per annum. They aimed at 24 heats a day as against 40 achieved in Brazil.

The EOF was considered to revolutionize the steel industry once again in a big way. It was also considered to surely replace the modified open-hearths in a more efficient ways. This was best suited for scrap-based mini-steel plants to replace electric furnaces. It means it was a battle between coal and oxygen on the one hand, and the electricity and electrode on the other. However the electric arc furnace seems to have won the race at least for some time. Yet the debate is far from over. The increasing use of chemical fuels in electric arc furnaces, again for steel making, is bound to reverse the trend at least partially.

Quite contrary to expectations, the EOF could not make much of a dent into the steel making in spite of being an efficient process. The reasons for such reversal need to be probed in detail.

10.6. Twin-Hearth Process (Dual-Hearth Furnace Process)

The necessity to improve the economy and productivity of existing open-hearth furnaces became more and more imperative with increasing adoption of BOF processes. A simple solution of modernization of open-hearth shop was to replace it with BOF processes of one type or another. In that case the replacement cost of open-hearth by BOF was substantial, even the existing low-height shop had to be totally changed to much taller shops. The other alternative was to improvise open-hearth design and practice to improve upon its performance. The oxygen lancing of open-hearth baths for intensive refining, introduced in the 1960s was a well-established practice. This was carried out in two ways:

1. Oxygen lancing with moderate intensity adopting usual duration of charging and heating periods (1500-2000 m^3/hr).
2. Oxygen lancing with higher intensity while reducing charging and heating periods (3500-4000 m^3/hr).

Out of these two the former resulted in increasing the output of open-hearths by about 15-20%. The latter improved it further but it often got restricted due to operational and thermal restraints. The roof life in this case was drastically reduced.

A third alternative evolved in the form of drastically altering the design of the existing open-hearth into a twin-hearth design more on the lines of a tandem furnace but with a much simpler design.

The existing stationary open-hearth was subdivided into two equal hearths by putting a wall on the line of the original tap hole as shown in Figure 10.4. The wall was short and did not extend right up to the roof. The arrangement thus resembled the tandem furnace arrangement with each half of the hearth acting as an independent chamber with independent oxygen lances. The operation was also similar to the tandem furnace operation, i.e., the carbon monoxide evolved in one chamber during oxygen lancing and refining was burned in the other chamber to preheat the solid charge. No traditional regenerators were necessary as there was no external fuel to be burned for the steel making operation. The exothermic heat of the process was utilized to meet the thermal requirements of the refining process during heating of solid charge.

The design was slightly altered by raising the roof to accommodate the intensive oxygen lancing without endangering the roof life.

The intensive oxygen lancing therefore did not result in overheating the furnace and the chemical heat of carbon monoxide could be fully recovered for scrap heating in the second chamber. The success of the process lies in synchronization of the technological and thermal conditions in the two hearths. The operational sequence is shown in Figure 10.5.

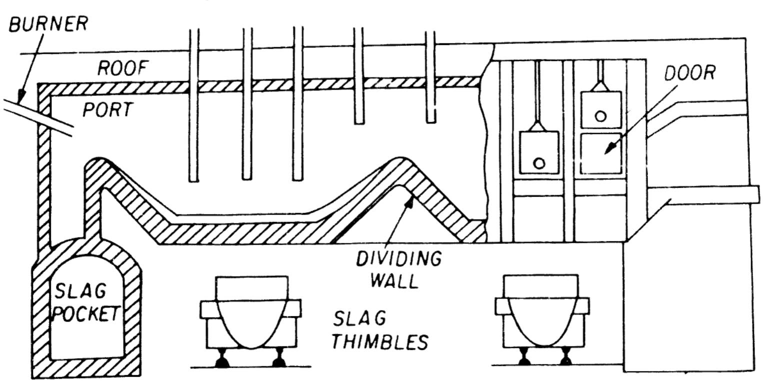


FIGURE 10.4. The twin-hearth furnace.

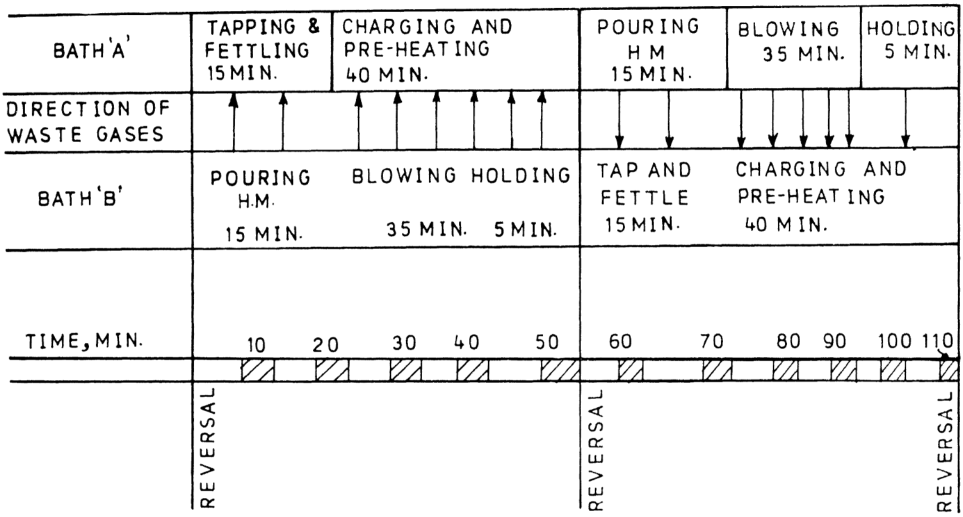


FIGURE 10.5. Graph of sequence and combinations of operations of the high productivity T.H. furnace of the former U.S.S.R.

The twin-hearth process was introduced for the first time in the former USSR and since its inception in 1965 several such units were made and put into operation. The construction was simple. Three roof lances were provided on each hearth. Oxy-fuel burners were provided on each hearth for additional heat supply, when needed. If it worked on around 80% hot metal charge it did not require any external additional thermal heat because sufficient heat was evolved by way of oxidation of impurities contained in the 80 % hot metal charge. If this share fell to 74-76 % of the charge then 12-25 kg of oil per ton was required which was ¼ to 1/6 of what was needed for a conventional open-hearth practice. Since the roof height was raised it was found to be 3-6 times better in twin-hearth than that for oxygen lancing in standard open-hearths. The specific consumption of refractories was also almost 1/3 that for open-hearth with oxygen lancing.

All varieties of steels produced by standard open-hearth could be produced by twin-hearth. The other advantages claimed were:

1. Changing over to twin-hearth would nearly treble the production rate of the same.
2. The design below the working floor was simple since regenerators were eliminated.

3. Heats were tapped at short and regular intervals and hence pitside problems were proportionately reduced.
4. No major change in civil structure was necessary, the same shop could be used.
5. Since one by one the existing open-hearths could be converted to twin-hearths the changeover was not likely to affect normal production adversely.
6. It could be readily incorporated into existing big open-hearth furnaces.

One by one all the 500 t open-hearths of the Bhilai Steel Plant were converted into twin-hearths. Each such conversion surpassed the expected production rate. The production rate almost matches that of the modified LD process as adopted in the other shop in the same plant. The special feature of this process was the use of nearly 90% hot metal in the charge.

This satisfactory switchover of traditional open-hearth to twin-hearths paved the way for others to adopt similar modifications. In some cases even this was later abandoned in favor of BOFs. It however continues to be used in the Bhilai Steel Plant to date (2008).

The problem with twin-hearth at Bhilai is quite different. The twin-hearth has a spout, as in the old open-hearth furnace, to tap the heat. This arrangement suffers from carrying a considerable amount of refining oxidizing slag along with liquid steel during tapping. The iron oxide contained in this slag gets reduced during deoxidation of the bath, hence its consumption unnecessarily increases. It interferes with the subsequent teeming process. The Bhilai plant has not yet found any practical solution for this problem.

CHAPTER 11

ELECTRIC ARC FURNACE PROCESS

In This Chapter

- Electric Heating
- Electric Arc Furnace
- Power Ratings and Consumption
- Charging
- Charge Materials
- Plant Layout
- Arc Furnace Operation
- General Outline of an Arc Furnace Heat

Steel making in an electric furnace became a commercial practice at the beginning of the last century, and the relative economics of this process vis-à-vis the open-hearth process has been the subject of controversy since then. Until recently the electric power, relative to chemical fuels, was costly and the electric furnace sizes were not large enough to successfully compete with the open-hearth for the production of bulk steel, i.e., mild and medium carbon steels. The electric furnace was therefore mainly used for alloy steel production until recently. The high metallic and ingot yields coupled with high quality of the product obtainable in this process were fully exploited in the production of costly alloy steels, which are required in large varieties but on smaller scales than carbon steels. The early electric furnaces were therefore of smaller capacities like 1 to 10 tons and in which 5-10 t were more popular for alloy steel making. The conventional use of the electric arc furnace for making alloy steels (without using oxygen for refining) is described in this chapter. Recent advances including the use of oxygen for refining and bulk steel production using hot metal in the charge will been described in later chapters.

11.1. Electric Heating

Electric furnaces are of three types:

(i) the resistance furnace,

(ii) the induction furnace, and

(iii) the arc furnace.

Electric resistance heating is not useful for steel making for a variety of reasons. The induction furnace will be described in the next chapter. The arc furnace and its operation are the subject matter of the present chapter.

The temperature of an electric arc using carbon electrodes exceeds 4000°C, hence steel making temperatures can be readily maintained in an arc furnace. Arc furnaces are of two types, namely, the indirect and the direct arc furnaces. In an indirect arc furnace, the arc is struck between two carbon electrodes and heat is transferred to the charge by radiations. Indirect arc furnaces are of fairly small capacities and do not develop steel making temperatures readily. These are, therefore, generally used in non-ferrous foundries although a few isolated examples may be found wherein these are used for scrap melting in ferrous foundries.

In a direct arc furnace, current flows from the electrode to the charge and heat is transferred from the arc to the charge primarily by radiation, but a part of the heat is also generated in the charge itself. The three-electrode arc furnace originally developed by Dr. Paul Heroult is most popular for steel making. The modern arc furnace still has some of the features developed by Heroult. The modern furnaces are supplied by various manufactures like Heroult, Birlec, Lectromelt, Efco, Demag, GEC, Brown-Broweri, Kulkarni Foundreis, etc. Although these are similar in their general design, they differ considerably in detail. In principle the three-electrode arc furnace is fed from a three-phase supply input, one phase connected to each electrode and the charge is the neutral point.

Acid-lined electric furnaces are used in steel-foundries and suffer from the same disadvantages as the acid open-hearth process suffers (purity of raw materials) and, therefore, are not commonly used. The basic electric arc furnace is very popular and is, therefore, described below.

11.2. Electric Arc Furnace

The furnace proper looks more like a saucepan covered from the top with an inverted saucer as shown in Figure 2.7. The electrodes are inserted through the cover from the top. Arc furnaces are of two different designs:

(i) The roof along with the electrodes swings clearly off the body to facilitate charging from top.

(ii) The roof is lifted a little and the furnace body moves to one side clearly off the roof to facilitate charging.

For smaller furnaces both of these alternatives are equally well suited, but for bigger sizes the body becomes too heavy to move and hence the swing-aside roof design is favored. It is quite popular even with small furnaces. The electric arc furnace is shown in Figure 11.1 with cutaway sections to show the details of construction.

The furnace unit consists of the following parts:

(i) Furnace body—i.e., the shell, the hearth, the walls, the spout, the doors, etc.

(ii) Gears for furnace body movements.

(iii) Roof and roof-lift arrangements.

(iv) Electrodes, their holders and supports.

(v) Electrical equipment—i.e., the transformer, the cables, the electrode control mechanism, etc.

11.2.1. Furnace Body

(i) **Furnace shell.** The furnace shell is a welded or a riveted steel plate construction and has a cylindrical saucepan-like shape. At least one elliptical furnace is in operation in the USA. Some of the old furnaces have a flat bottom but a spherical bottom is now almost universally adapted. The spherical bottom is stronger, it does not need structural bracings, and it allows free

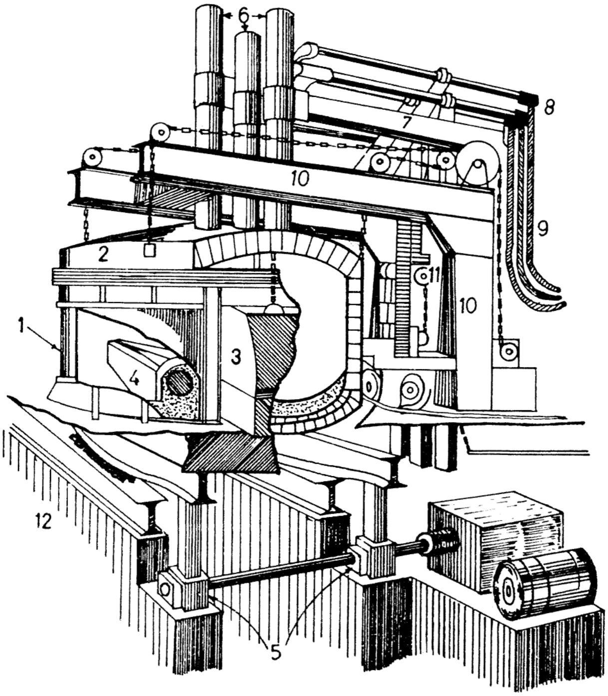


FIGURE 11.1. Electric arc furnace with cutaway sections to show the details of construction. *Legend:* 1. Furnace shell. 2. Roof. 3. Door. 4. Spout. 5. Tilting gear. 6. Electrodes. 7. Arm to support electrode. 8. Bus bars. 9. Flexible cables. 10. Roof lifting arm. 11. Electrode control. 12. Concrete foundation.

expansion and contraction. There is no room left for contraction and expansion of the shell and in some furnaces the shell plates did crack during use. Some bigger furnaces have, therefore, been designed to actually allow the plates to move within a steel structure. This however is not common.

The spout for tapping the metal is welded to the bottom and the main door is situated directly opposite the spout.

The height of the wall is chiefly governed by the nature of the raw materials and the practice of refining. For bulky scrap the height of the wall should be increased. Similarly if oxygen lancing is adopted taller walls are preferred.

The top of the circular wall takes the main roof-seal-ring to support the roof. Several sealing designs are in use. In one design a channel ring is used. It is filled with sand and the roof ring rests on this. In another one a flat plate is used to support the main roof ring. The top sealing ring is water-cooled.

The following table indicates some approximate dimensions of the furnaces:

Normal capacity	t	5	10	20	80	150
Shell dia	m	2.7	3.3	4.5	6.0	7.2
Depth from sill	m	0.45	0.52	0.72	1.00	1.07
Total thickness of hearth	m	0.45	0.52	0.72	0.72	0.73
Roof thickness	m	0.22	0.30	0.30	0.35	0.35

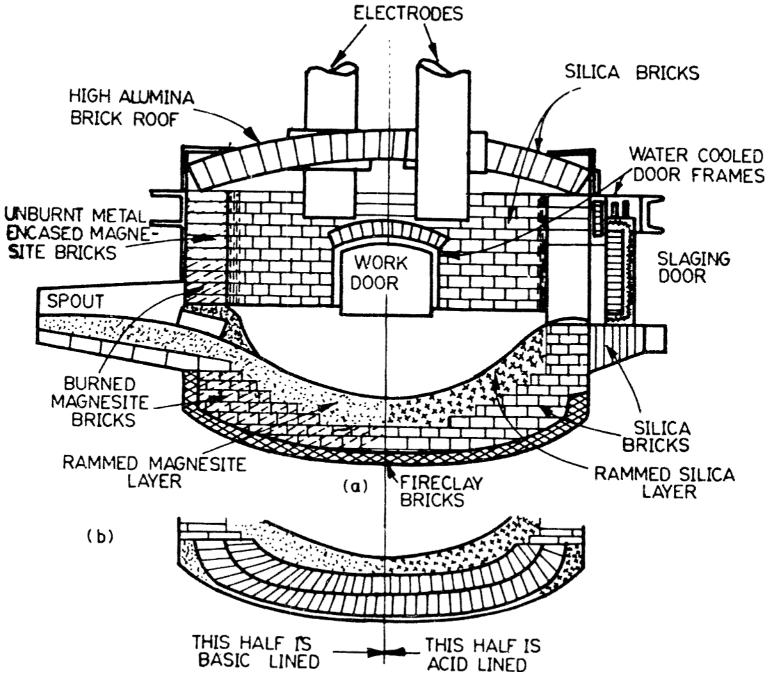


FIGURE 11.2. Cross-section of an electric arc furnace showing the details of refractory lining. The stadium type sub-hearth construction is shown in (*a*) and the inverted sub-hearth construction in (*b*).

(ii) **Refractory lining.** The furnace shell is lined from inside with suitable refractory to suit the refining operation. The mode of lining is illustrated in Figure 11.2. Parts of the furnace lining are referred to as the bottom or hearth, the sidewall, the tap hole, the roof, and the runner or spout.

The bottom or hearth is the lining of the shell at the bottom up to a little above the door-sill level. The sidewall lining extends above the door-sill level up to the top edge of the shell. The tap hole is located slightly above the slag line and opens up into the spout, which is lined, assuming it to be a part of the hearth.

Hearth. Next to the shell is a layer of firebricks at the bottom and is followed by magnesite brickwork to form the brick sub-hearth. Magnesite is preferred to dolomite for making the sub-hearth. Necessary gaps are left in the brickwork for its expansion on heating. A good sub-hearth lasts for several years. The working hearth is made by ramming tarred dolomite or magnesite as in the case of an open hearth. The contour of the hearth should be such that when it is tilled for tapping all the metal and the slag should freely flow out. Many times steel formers are used to help make a rammed bottom. The formers are left in and are allowed to melt subsequently. Rammed bottoms have proved to be more efficient than burned in-bottoms, contrary to the open-hearth practice.

It is illogical to try to make high quality steels in a furnace having an improper bottom. Many of the ills of electric steel making can be traced to a poor furnace bottom.

The newly rammed hearth is slowly dried and brought to the working temperature by placing coke on the bottom and striking an arc against it. A freshly rammed hearth is given a slag wash at the working temperature as is done in an open-hearth furnace.

Sidewall. The sidewall—the cylindrical part of the shell—extends vertically from the slag line to the top of the shell. Except for small furnaces, the use of a silica sidewall is now rare. Nowadays magnesite, dolomite, or chrome-magnesite (preferably metal encased) is used to line the sidewall. Magnesite is better if the furnace is worked hard. Sometimes magnesite is preferred at the lower part and silica in the top portion with an intermediate layer of chromite. This is all right if a silica roof is used. The sidewall thickness is usually in the range of 35-50 cm. The general tendency is to use large blocks rather than usual sizes of refractory bricks to construct the sidewall. The use of blocks reduces the relining timings and, since wrecking bars are inserted, wrecking of worn-out lining is also made easy. The blocks however, being not fired, give trouble in handling. Use of metal encased magnesite chrome bricks is quite popular for this purpose except around the tap-hole.

Tap Hole. Generally a gap is left at the proper place while laying the bricks for making the tap hole. A round former is inserted and the space left around is rammed to make the tap hole. This is quite cheap to make and maintain than the use of shaped bricks.

Spout. The teeming-ladle-quality fire bricks are used to line the spout next to the steel plate. The rest is rammed along with the hearth as a part of the hearth.

(*ii*) **Door.** Smaller furnaces have only one door directly opposite the tap hole, and any additions during refining are made through this door. This is the main door and is also used for slagging. Bigger furnaces have another additional door at right angles to this door. The door is lined with basic bricks or is a rammed basic monolithic and is water-cooled. It resembles the open-hearth door. The door should be at a proper level to help remove the slag by tilting the furnace on this side. The sill should be well protected to stand the slagging-off operation.

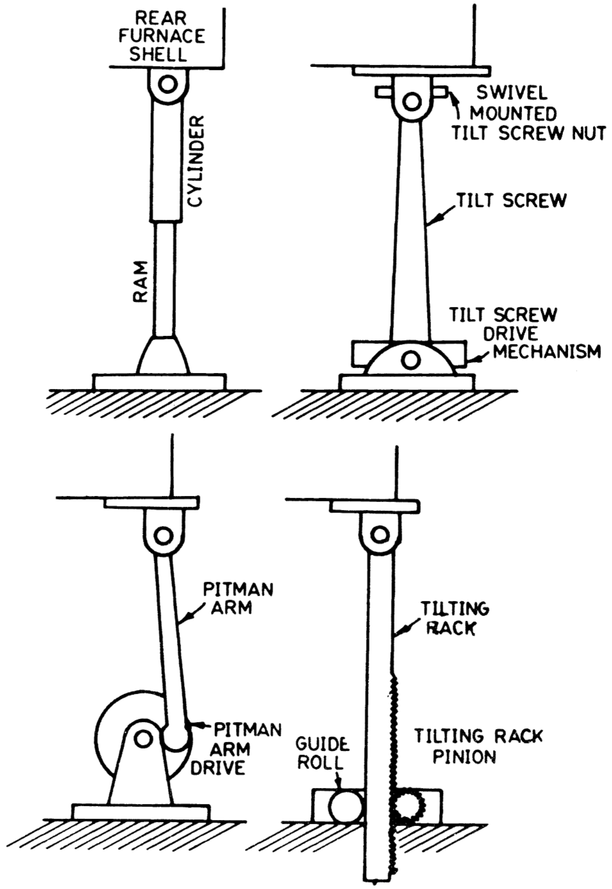


FIGURE 11.3. Different types of electro-mechanical and electro-hydraulic gears for tilting arc furnace.

11.2.2. Gears for Furnace Body Movements

The furnace body needs to be tilted nearly through 45° on the tapping side and 15° on the slagging side. The tilting gear is either hydraulic or electrical. In electrical, a motor actuates the rear of the furnace as shown in Figure 11.3. Guide rails are used to ensure true movements. Some locking arrangement is necessary for the horizontal positioning of the furnace. Hydraulic tilting arrangement is smooth and is specially suited for large furnaces. The dead weight of a furnace may be nearly ten times its nominal capacity.

The furnace can be tapped by two alternative methods. In one the spout stays in position and the furnace is lifted up. The ladle in this case remains stationary. In the other, the furnace is tilted around its center of gravity so that the spout also changes its positions. The ladle needs to be either suitably located even for later positions of the spout or it should be adjusted during tapping.

Large furnaces are rotated in a horizontal plane as well. Usually the furnace is rotated through 30° in the forward and 30° in the backward direction. This gives nine different positions of arcing to distribute the hearth, and thereby melt the charge more evenly. Small furnaces need not have such an arrangement. In more recent furnaces a separate tap hole is provided for tapping the heat. It is situated at the bottom but eccentrically by extending the furnace bottom in one direction to accommodate the tap hole. This will be described later while dealing with recent modifications in arc furnace design and practice.

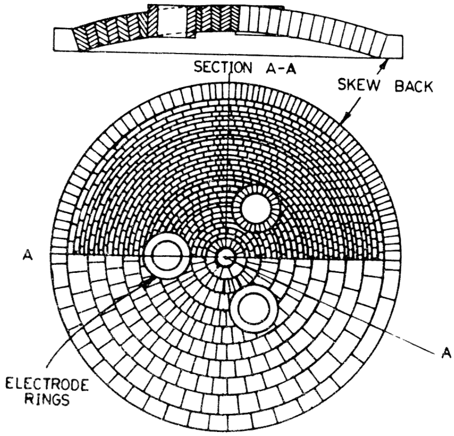


FIGURE 11.4. The electric arc furnace roof.

In some designs the tilting gear is attached to a platform or a skirt attached to the furnace at the working floor level. This skirt seals off the gap between the furnace and the floor and safeguards the equipment underneath. It helps to make electrical and water piping connections safely.

11.2.3. Roof

The roof is a domed construction and rises about 1 percent in span towards the center. The roof has three holes located symmetrically to allow insertion of the electrodes. The holes are made from ring-type bricks. The entire roof takes the form of rings as shown in Figure 11.4. The roof is built from skewback in the form of a ring, of which several designs are in use. A good design seals off the furnace properly. The diameter of the roof ring is nowadays kept little more than that of the shell to protect the skewback from direct heating. The bricks are laid on a concrete dome of the required contour, as a former, to support the bricks in the initial stages.

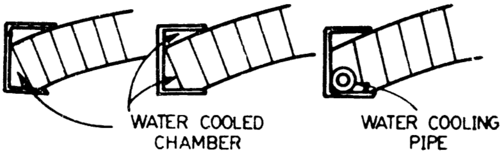


FIGURE 11.5. Water-cooled skewback channel rings for an arc furnace.

Expansion allowance is provided while making the roof. Roof thickness varies from 25 to 45 cm. The skewback is supported in a water-cooled skewback channel ring as shown in Figure 11.5.

A silica roof is popular as in a basic open-hearth furnace. The roof in an electric arc furnace (Figure 11.6 a-c) is subject to greater thermal fluctuations and hence basic roof cannot be used. The current trend is to use high alumina, 70-80% Al_2O_3, bricks to make the roof.

These bricks are costlier than silica but are superior in performance to silica. The roof fails chiefly because of splashings of iron oxide-rich slags that corrode it at the steel making temperature.

Nowadays more than one roof is provided for each furnace unit so that even if one fails, a new one can be immediately replaced and the campaign continued, unlike in open-hearth practice. The change of a roof does not take more than an hour.

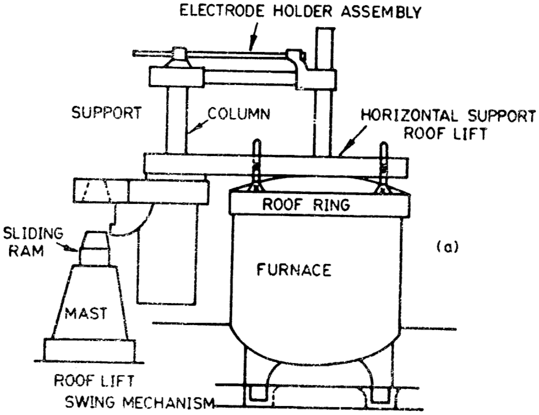


FIGURE 11.6.(a). Electric Holder Assembly

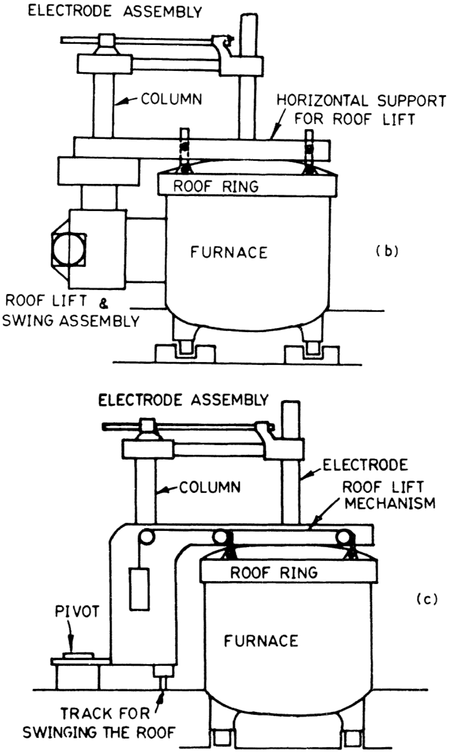


FIGURE 11.6 (*b* & *c*). Various types of roof lift and swing arrangements: A mast independent of furnace shell is used in (*a*) and in (*b*) it is attached rigidly to the shell; a track and pivot is used for lifting and swinging the roof in (*c*).

It is a general practice now to charge the furnace from the top. This requires the roof to be clearly off the furnace top. In the swing-aside roof design several arrangements are used to lift the roof and swing it away from the furnace. Three such arrangements are shown in Figure 11.6. The roof of a modern large capacity arc furnace is provided with additional holes for charging continuously, as and when required, at the desired rate, solid additions like lime, dolomite, DRI, ferro-alloys, etc., with roof in position and during working of the heat actively.

11.2.4. Electrode and Its Support

Electrode. The electrodes are either of carbon or graphite and are capable of carrying current at high density. Graphite, having better electrical conductivity than carbon, is generally preferred. Their sizes vary from almost a few centimeters to nearly 100-110 cm depending upon the furnace capacity. These are circular in cross-section and each piece is about 1-3 m long.

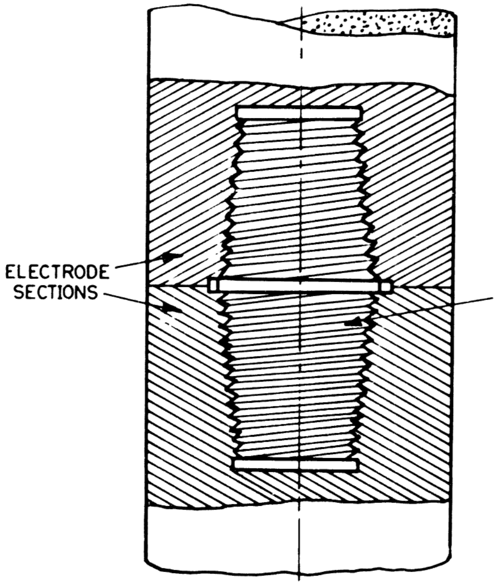
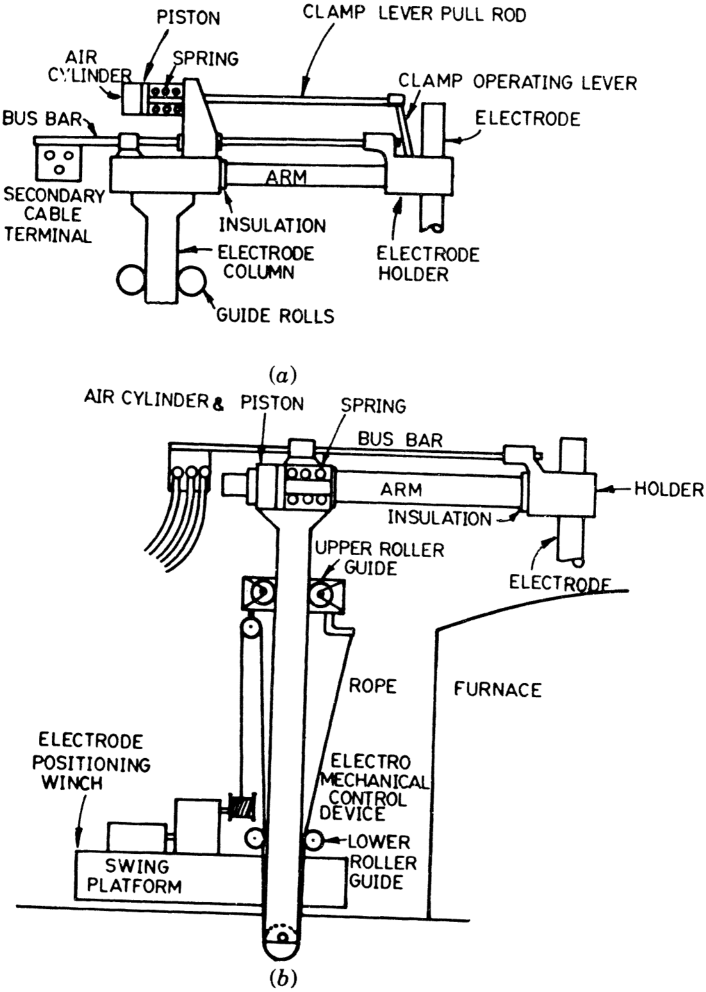


FIGURE 11.7. Electrode nipple joining two pieces of electrodes.

The electrode pieces are joined, one to the other, with the help of a nipple as shown in Figure 11.7, to virtually act as an endless electrode. An electrode stand is used for coupling the pieces. The electrode is a costly material and hence its consumption during the operation should be minimum. The consumption varies with practice and is generally 3-6 kg/t of steel made.

Continuous self-baking Soderberg electrode using carbonaceous paste has been developed, and is in use in Scandinavian countries, but has not gained much popularity elsewhere.



(*a*)

(*b*)

FIGURE 11.8. Power-operated electrode clamp and the electrode manipulating gear. In (*a*) the electrode is clamped through a lever operated by a compressed air cylinder and piston, and in (*b*) directly.

Electrode control. For efficient melting the arc must be stabilized to supply uniformly a maximum amount of energy. This necessitates keeping the distance between the electrode and the charge constant. As melting occurs, this distance increases and if any solid piece collapses in, it may decrease. The electrode is automatically adjusted to maintain a stable arc. The electrode control systems either try to maintain a constant current or constant voltage across the gap. A third type of control tries to maintain constant impedance in the individual electrode circuit. Each electrode is provided with an independent electromechanical or electro-hydraulic system of control. A number of these designs are in use and are still being improved to obtain a relatively maintenance free electrode control which will maintain a balanced and constant flow of energy into the arc. The system is expected to sense any off-balance in the circuit and correct it within less than even a hundredth of a second.

Electrode support. The furnace is provided with a vertical column or mast to which horizontal arms are attached and the electrode is held by the arm in position. The masts are of two types. In one design the mast is fixed and the electrode arm travels up and down the fixed mast. In the other, the mast itself rises and falls; the electrode arm is rigidly attached to the mast. The later design is more popular because it allows easy tilting of the furnace. Electro-mechanical or electro-hydraulic drives are used for activation of the masts in effecting electrode control. The electrode arm supporting the electrode clamp is generally tubular and is water-cooled. It is insulated from the mast as well as the clamp. Current is carried from the transformer by flexible cables to the copper bus bar or tubing. Use of copper tube to connect cables to the clamp is preferred since it can be water-cooled.

The electrode is held in a vertical position in a water-cooled copper clamp situated at the end of the electrode arm. It is vital to have the contact of the clamp with the electrode as clean as is possible, so as to keep the contact resistance to an absolute minimum. Power operated clamps of various designs are increasingly being adopted in preference to wedge type earlier designs. In these designs the movable clamp shoe is pressed against the electrode directly or through a lever using springs as shown in Figure 11.8. The spring is released by air pressure to allow slipping of the electrode downwards.

In general the possibility of parts in the furnace super structure being heated by eddy currents, induced due to the heavy secondary currents, is minimized by using non-magnetic type of metallic material, i.e., austenitic stainless steels.

11.2.5. Transformer

Large transformers are required to run electric arc furnaces. The primary voltage may be 33 KV or more and the secondary with a series of tappings from 60 to 300 volts. A simplified circuit arrangement is shown in Figure 11.9. During melting, more power is required than during refining. The transformer capacity is designed to suit melting requirements. The capacity is, therefore, under-utilized during refining. It is economical to have at least two furnaces operating side by side to improve the overall electrical power factor. Since high current densities are involved, special circuit breakers are used.

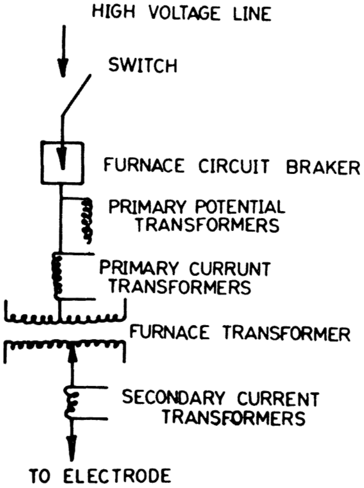


FIGURE 11.9. Simplified electrical circuit of an arc furnace.

The capacity of the transformer is usually 470-650 KVA per metric ton of furnace capacity. For small furnaces this value is on the higher side of the range. In terms of hearth area the transformer capacity is in the range of 750-900 KVA per square meter.

11.3. Power Ratings and Consumption

The arc voltage as well as the power factor decreases with increasing current in the circuit. The power consumption in the circuit and in the arc increases with increase in the current in the circuit up to a certain limit, and then decreases. All such relationships should be plotted for an individual furnace. These are only qualitatively shown in Figure 11.10. Certain optimum conditions of working are established from such relationships. The furnace is best operated at the amperage where maximum possible power is available in the arc at maximum possible power factor, as shown qualitatively by the shaded area in the above figure. Electrode ratings should also be taken into account in deciding the working conditions. The electrode is fully utilized by drawing maximum allowable current through it at maximum possible voltage. Excess current unnecessarily overheats the electrode and lower current results in poor utilization of the electrode. The current-voltage ratings of an electrode are provided by the supplier in relation to its size and chemical composition.

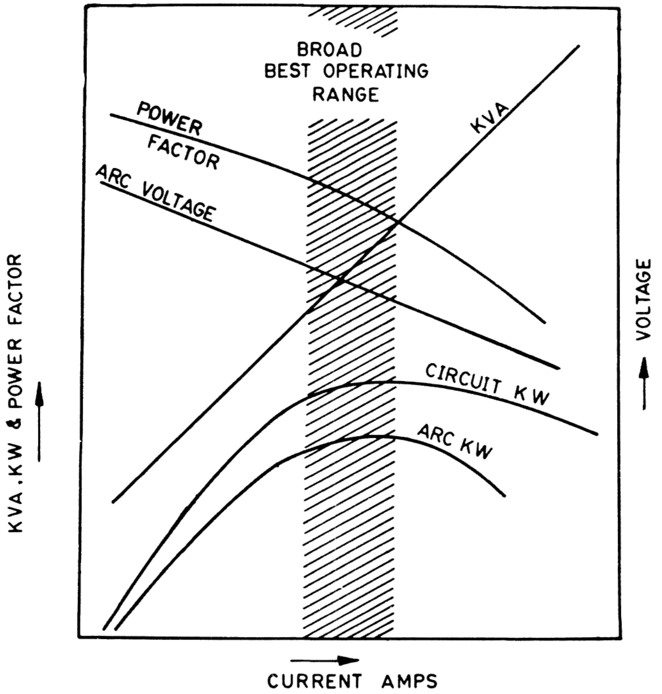


FIGURE 11.10. Electrical characteristics of an arc furnace operation (schematic).

The power consumption for melting the charge in small furnaces is about 600 KWH/t, and it falls to 450 KWH/t in big furnaces. Additional power is required during refining, which varies considerably with practice between 150-400 KWH/t. The total power input is spent approximately in various ways as follows:

To the charge	…	50-65%
Electrical losses	…	8-12%
Water cooling losses	…	3-5%
Radiation losses	…	20-30%

11.4. Charging

Earlier smaller furnaces were charged through the door manually. Door charging continued even on bigger furnaces using a chute or using a mobile open-hearth type box charging machine. Charging is now invariably carried out from the top. A drop bottom basket is used for charging as shown in Figure 11.11. The earlier orange-peel type of its design has now been replaced by a clamp-shell type basket for bigger furnaces. The bottom of the basket is split in two halves and can either be closed or opened by the auxiliary hoist on the crane holding the basket.

A mobile charging machine is used for big furnaces to charge small amounts of additions during refining through the additional door provided on these furnaces.

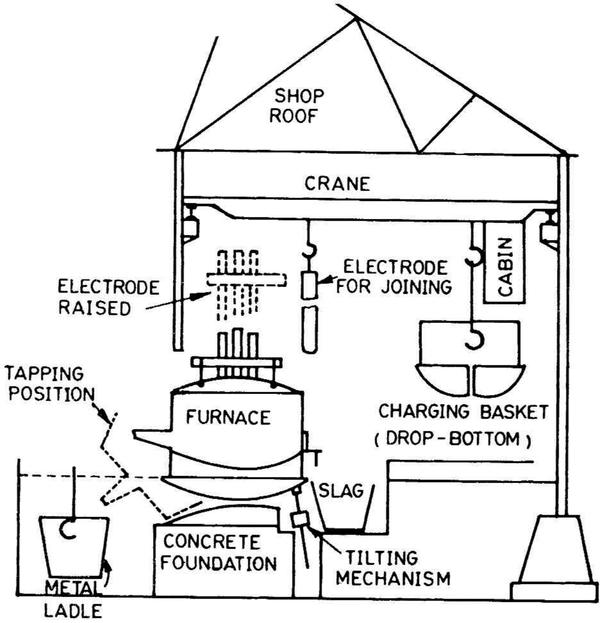


FIGURE 11.11. Vertical section of an electric arc furnace shop.

11.5. Charge Materials

The raw materials have already been discussed in Chapter 5, and are not being repeated except for scrap, and that for the sake of further details relevant to an electric furnace process alone.

Steel scrap is the single most important raw material used in steel making in an electric furnace. It may constitute 60-80% of the raw material cost in this process. The problem of scrap in an electric arc furnace process is very complex compared to that in an open-hearth because of a wide variety of grades of steels to be produced by an electric furnace process. Besides this, the electric furnace takes nearly 100% scrap charge as compared to the usual 75% maximum in an open-hearth. The plant may produce some part of its output as circulating home scrap but the rest perhaps has to be bought from outside from a number of different sources. The scrap can be classified in four broader groups on the basis of its chemistry from the point of view of steel making.

1. Scrap containing volatile elements like Zn, Cd, Pb, etc.
2. Scrap containing non-oxidisable elements (during refining) like Cu, Ni, Sn, Mo, As, W, etc.
3. Scrap containing partially oxidisable elements like P, Cr, Mn, etc.
4. Scrap containing completely oxidisable elements like Al, Si, Ti, V, Zr, etc.

The second category gives rise to what is known as the problem of residuals. In open-hearth the scrap is diluted to some extent by virgin pig or hot metal in the charge. This is not readily possible in an electric furnace since nearly 100% scrap charge is used. This situation is largely changed since hot metal forms a substantial part of the charge for large-scale steel production using large capacity furnaces.

Plain carbon steel scrap, that is ordinary scrap, is cheapest of all scraps and is used in large bulk. It is supplied either as low phosphorus scrap or ordinary mixed scrap in various grades as per its physical size and shape. Care exercised in selecting the right proportioning of heavy and light scrap is rewarded subsequently during melting and refining of the charge. Any lack of attention towards the quality of scrap may affect the quality of the product and process economy.

Alloying elements obtained from scrap are always very much cheaper than those from ferro-alloys or virgin metals. The recovery of alloying elements in an electric furnace is much better than that in an open-hearth, particularly of oxidisable elements. Moreover since most of the alloy steels contain only one or two main alloying elements, the process economy can improve if a properly classified alloy steel scrap is used. The scrap must, therefore, be properly segregated in order to recover its alloying contents.

The low alloy steel scrap is generally segregated as Ni, Cr, Ni-Cr, Cr-Mo, Ni-Mo, Ni-Cr-Mo, etc., type of scraps.

The stainless steel scrap is classified as Cr, Cr-Mo, Ni-Cr, Ni-Cr-V, Ni-Cr-Mo, Ni-Cr-Cb, Ni-Cr-Mn, Cr-Mn, etc., type of scraps. The manganese steel scrap is classified as Mn (12-15% with low Ni, Cr and Mo), Mn-Cr (1-3% Cr and otherwise like the previous), Mn-Mo (with less than 0.4% Mo), Mn-Cr (1-3% Cr and less than 0.4% Mo) and Mn-Cr-Ni-Mo (1-3% Cr and less than 0.4% each of Ni and Mo) type scraps.

Solid iron scrap refers to a material high in impurity contents such as pig iron or cast iron.

Even the mill scale of an alloy steel is classified for use as not only an oxidizing agent but as a source of the corresponding alloying elements, hence needs to be segregated appropriately.

The above description is included to indicate the practical modes of classification of scraps of some of the important alloy steels.

In principle the charge is made as a mixture of various grades of scraps, such that when it melts the bath contains lower percentages of various elements than are required in the final product. This will allow for proper final adjustments with a minimum of necessary alloy additions. If the oxidizable impurities are more in the melt, some of the oxidizable alloying elements will also be oxidized during refining and may be lost in the slag. Due care should be taken about such eventuality while preparing the charge. As far as possible the charge for an alloy steel heat should be developed to take advantage of the availability of the same type of alloy steel scrap in the market, since it proves quite economical.

11.6. Plant Layout

The layout of an electric arc furnace steel making shop varies considerably from plant to plant. The difference is essentially due to the difference in the quality of the product and the scale of production. Several plants have just one arc furnace. Some may have two, but shops with more than two furnaces are definitely uncommon. The variation is also due to whether the shop is provided with oxygen lancing facilities, gas cleaning equipment, a vacuum treatment unit, continuous casting unit, and so on. The shop designed to produce finished castings is different from the one meant to produce ingots.

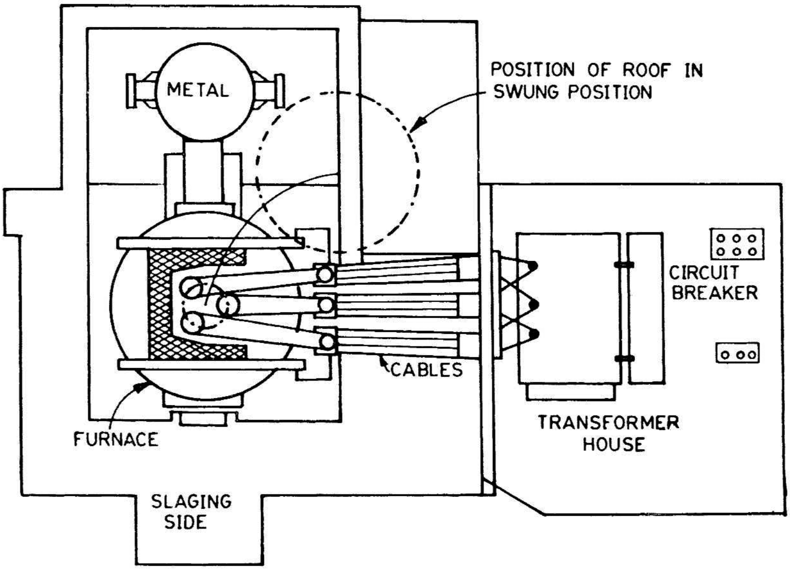


FIGURE 11.12. Plan view of an electric furnace bay.

One of the modern layouts was already shown in Figure 7.3. The vertical elevation of the main bay housing the furnace, transformer cabin, control cabin, etc., are shown in Figure 11.11, and Figure 11.12 shows its plan details. On the spout side of the furnace is the teeming bay (or mold bay) and the stockyard for all the raw materials is located on the other side of the furnace bay. The roof formers, the roof storage, electrode stand, etc., may be suitably located. If any other facilities are provided, these will have to be properly situated with respect to minimum overall distances involved in material handling.

In general, overhead cranes are used for material handling. The furnace may be charged by one or more baskets depending upon its size. The ferroalloy bunkers are located close to the furnace. Mobile chargers may be used for adding slag making additions. In large furnaces hoppers are used.

11.7. Arc Furnace Operation

The operation of an arc furnace varies with the furnace design, raw materials used to make up the charge, charging facilities available in the shop, process of refining adopted, quality and type of the product required, and so on. For a given shop, the melter has a variety of refining practices to choose from, depending upon the quality of the charge and the product required therefrom. The variation is mainly related to the practice of slag making and it is, therefore, aptly said that the melter makes the slag and the slag makes the steel.

11.7.1. Process Types Known by Their Slags

1. Acid Process. If the raw materials are very low in P and S, an acid lined furnace can be used for refining, using an acid slag as in an acid open-hearth practice. It is generally restricted to foundries.

2. Basic Process. It is capable of refining any type of charge by maintaining basic slag in a basic lined furnace. Unlike any other steel making process, the electric furnace has practically no oxidizing atmosphere of its own. Oxidizing as well as reducing conditions for refining can be maintained by making slags of suitable compositions. Oxidizing refining is carried out under a slag containing a good amount of iron oxide. Reducing conditions can be maintained by having the slag highly basic but practically free of iron oxide. The following describes the ways in which these slags are used for refining in an arc furnace:

(i) Oxidizing single slag practice. It is used for making carbon or low alloy steels of a quality attainable in an open-hearth process. The charge is melted and refined under a basic oxidizing slag as in an open-hearth. The alloy additions may be made in the furnace or in the ladle.

(ii) Oxidizing double slag practice. It is a modification over the single slag practice. The early slag is removed and a similar new slag is made again to obtain effective desulfurization and dephosphorization during refining.

(iii) Reducing single slag practice. It is used for high alloy steel making to effect maximum recovery of alloying elements from the scrap. Hardly any refining takes place. Carbon and phosphorus contents in the scrap must be well below the specification levels since these will not be removed during refining. Sulfur, however, could be readily removed in this practice since the conditions are reducing.

(iv) Oxidizing slag converted to reducing. It is meant to remove carbon but recover most of the alloying contents like Cr, Mn, etc., present in the scrap during high alloy steel making. Phosphorus content of the charge needs to be below the specification level, since it will otherwise revert to the metal during the reducing period.

(v) Double slag practice. It means refining under oxidizing as well as reducing slags made separately. The first slag is oxidizing and it eliminates all impurities like P, Si, C, Mn, etc. This slag is removed and a reducing slag is made by fresh additions of lime, coke, and spar to desulfurize the metal and to carry out alloying very effectively. The practice is a must if effective desulfurization and the large alloying additions are to be made. It is costly but the yield of the alloying additions is very high and the quality of the product is much better.

Among the above practices the (*i*), (*iv*) and (*v*) types of practices are more widely adopted in practice. The (*iii*) and (*iv*) types of practices are used in induction furnace processes.

11.8. General Outline of an Arc Furnace Heat

Although the details of a heat very much depends on local conditions, quality of the charge and the product to be produced a general outline of a heat is only described below:

The essential steps are as follows:

1. Preparation of charge.

2. Charging.

3. Melt-down.

4. Refining.

5. Finishing and tapping of single slag heat.

6. Slag-off and making reducing slag.

7. Reducing period.

8. Finishing and tapping of the heat.

In principle all the above operations should be so organized that the active furnace life is utilized, with minimum of delays, without impairing the quality of the product, to obtain maximum overall economy.

The arc furnace, being a hearth process, resembles the open-hearth practice with respect to refining.

11.8.1. Furnace Preparation

After tapping the previous heat the slag is completely drained out and lining is inspected. The eroded portion, usually the slag and the slag-metal boundary lines, the doorsill, the tap hole, the spout and the damaged area of the hearth are all repaired in hot condition using granular dolomite or magnesite. Fettling may be done manually or by machines. The tap hole is repaired and plugged before patching the slag line. The doorsill and the spout are cleaned and then patched. Door-charged furnaces need more extensive repairs than top-charged furnaces.

11.8.2. Charging

A little of light scrap on the furnace bottom provides a cushioning effect and protects the hearth from mechanical damage due to the falling of heavy pieces of scrap. In top charging, therefore, the basket is loaded with a little of light scrap at the bottom. The heavy scrap is kept above this and is mainly in the center. The remaining light scrap is loaded around heavy and at the top. Burned lime and spar are added in the basket to help form early slag of the required qualities. If refining is to be carried out during melting, lumpy iron ore and/or mill scale is also included in the charge. Broken electrode pieces, coke, etc., are added if required, in the basket itself to obtain required carbon in the bath at the melt-out.

While the hot repairs of the furnace are being carried out, the charge is kept ready in one or more baskets. The furnace body is cleared off the roof and the basket is lowered in or held just above the furnace by an overhead crane. The bottom of the basket is opened and the charge drops out in the furnace. As far as possible all the charge should be put in the furnace in one lot using one or more baskets. If the proportion of light scrap is more, recharging becomes necessary. In this case the heavier scrap is charged in the first lot, as far as possible, and the remainder of light scrap after the melting of the first charge in a minimum number of recharges. The thermal and electrical conductivities, the physical condition, and the distribution of the charge in the furnace are quite important factors governing the operational economy. Care exercised in selecting the right type of scrap and correct proportioning of light and heavy scrap is rewarded subsequently during melting and refining of the charge. A certain amount of light scrap at the bottom permits easy melting of the charge with minimum damage to the bottom and overheating of the roof. As a rule of thumb 40% heavy, 40% medium, and 20% light scrap are incorporated to make up the charge that minimizes back charging. Heavy scrap pieces should be of such sizes and be charged in such a way that they do not cause bridging and short-circuiting of the electrodes.

11.8.3 Melt-down

After the first charging is over, the roof is replaced in position, the electrodes are lowered manually, the arc is struck and the electrodes are put on automatic control. As the metal just below the arc melts and drips down, the electrode travels automatically further down to maintain a stable arc. In a furnace that can be rotated in a horizontal plane the melting can be made more even by striking the act at different points. Where such provision is not available the electrodes are lowered and allowed to bore the charge to some extent and then raised. The voltage is raised and the electrode is again allowed to bore until there is no danger of short-circuiting. This is repeated, if required, until the electrodes travel to the bottom and a pool of molten metal is formed at the bottom. Hereafter maximum power is fed to the furnace for melting the charge. The hanging pieces of scrap may be pushed in during the off periods and the charge is melted to level. Recharging should be done, if necessary, at this point and the melting is continued until the charge is fully molten and is superheated to the desired level.

The presence of lime and spar in the charge helps to form slag during melting. If ore is incorporated in the charge most of the phosphorus and silicon will be oxidized by the time of melt out. The presence of lime in the slag protects the lining from being attacked. There is a small draught in the furnace because cold air gets in and hot gases leak out around the electrodes through the roof. This is not enough to provide adequate furnace oxidation (as in an open hearth) of the charge and hence charge ore is essential in the charge if oxidizing refining is to take place. This is applicable even when oxygen gas is used for refining. The charge ore concept must be understood from the point of view of necessity of iron oxide-bearing slag, from the very beginning when the solid charge melts down; any impurity in the scrap needs to be oxidized on melting and for which this charge-ore provides the necessary oxidizing media, until oxygen is introduced later for refining.

During melting, the choice of secondary voltage at each stage is made judiciously to obtain high power factor. Heat developed in the arc, as far as possible, should be utilized by the charge. Excess heat is reflected to the furnace lining and roof, and is not only wasteful but also damaging. A set of rules is therefore put up in each shop for power input manipulation as per the local conditions.

11.8.4. Refining

Refining continues even during melting as in an open-hearth process. The actual amount of impurities to be oxidized from the charge is generally very small in arc furnaces, and is carried out fundamentally much in the same way as in an open-hearth practice. The oxidizing slag at the melt-out may have composition like:

CaO 38-45%, SiO_2 10-15%, FeO 13-20%, and MnO 10-15% with a V-ratio equal to 1.7 to 3.0.

The melt-out stage must be quiescent for obtaining a correct bath sample for chemical analysis. The carbon content would indicate the extent of refining required. Refining can be commenced by the addition of ore. Oxidation of carbon gives rise to a boil that is equally essential in an arc furnace as in an open-hearth practice. The opening carbon should, therefore, be above the specification level by an amount equivalent to the carbon boil requirements. A 15-30 points i.e., 0.15-0.30% excess is considered adequate for most of the purposes as against 0.3-0.5% excess carbon required in an open-hearth practice.

At the end of melting, the first thing is to control phosphorus if it is above the specification level. Phosphorus can be eliminated by keeping the slag oxidizing and basicity of the order of 2.2-2.5 V-ratio. Since phosphorus is prone to reversion at high temperatures it must be fully eliminated before the bath picks up temperature. If the bath temperature rises before phosphorus is lowered, the bath has to be cooled down, e.g., by some scrap addition, and refining is continued with further additions of lime, spar, and ore. By the time all the phosphorus is eliminated, carbon boil should have attained full vigor to clean up the bath.

During refining, bath samples are periodically taken out to assess the progress of refining. Lime and spar may be added during refining to keep the slag in shape. Feed ore should be added earlier in the refining but its addition towards the end should be avoided as far as possible to prevent over-oxidation of the bath. If the carbon content of the bath is less than 0.3% the bath has a tendency to get over-oxidized and this has to be carefully avoided.

Once the chemistry of the bath and the temperature are at the desired level the heat is ready to be deoxidized and finished for tapping. Bath temperature is measured by an immersion, optical, or radiation pyrometer.

In the oxidizing double slag practice the slag is allowed to form and flow out towards the end of the melting period by tilting the furnace down on the door side. A second oxidizing slag is then made to ensure complete elimination of phosphorus. The heat is finished in the usual manner. This resembles the flush slag practice in an open-hearth furnace.

Preliminary deoxidation is done by Fe-Mn, Fe-Si, etc. Many operators block the heat by dipping the electrodes in the bath. This practice improves cleanliness of the bath.

11.8.5. Finishing and Tapping of a Single Slag Heat

Low carbon heats are tapped open without blocking which otherwise is a more common practice. Bath temperature at the time of blocking should be on the high side. Alloying additions for manganese, chromium, etc., can be made to a properly blocked heat since the lowered oxygen content protects them against oxidation and improves their recovery. Final deoxidation by Fe-Si and Al, and other alloying additions can be made in the ladle.

The heat is tapped by opening the tap hole. The furnace is tilted by a jerk to prevent slag from flowing along with the metal. There should be a minimum of slag in the ladle. The ladle should be placed in the pit beneath the spout well ahead of tapping time.

11.8.6. Slag-off and Making Reducing Slag

In a double slag practice at the end of the oxidizing period slag is removed from the furnace by tilting it on the back side. Small additions of spar thin down the slag that flows over readily. Rabble may be used to rake it off as completely as possible. Reducing slag is formed by adding fresh charge of lime and spar in which a little of sand may be mixed to help form the slag. Strongly reducing carbidic slag can be formed by putting coke on the slag after it has melted. It forms carbide as:

$$(CaO) + <3\,C> = (CaC_2)\,\{CO\} \qquad \ldots(13.1)$$

which is quite effective in desulfurizing the bath. The lime to coke ratio may be 6 : 1 to 12 : 1. Slag without coke or with little coke is white in color and is called lime slag; that with high percentage of coke is grey in color and is known as carbide slag. Both of these types are readily discernible from the dark brown oxidizing slags. Carbide slag tends to recarburize the bath and hence is not suited for making low carbon heats. For heats containing less than 0.15%C reducing slag is made by the addition of lime, Fe-Si and aluminum. No coke is added for it would recarburize the bath. The reducing slag compositions are approximately as:

Constituent	*Carbide slag*	*Lime slag*
CaO	65-70%	55-60
SiO_2	20-25%	25-30%
FeO	0.5% max	1.0% max
Mgo	5-10%	5-10%

Desulfurization of the bath is very effective under such a slag (see Section 4.2.5).

After the reducing refining is over, the bath is fully deoxidized by additions of Fe-Si and aluminum. The alloying additions may now be added based on the analysis of the bath. The metal can be held to pick up temperature without losing any alloying additions. The process is, therefore, better than even the acid open-hearth process to make any type of alloy steels.

It is not beneficial to hold the metal under reducing slag for too long since the bath is quiescent, and stratification of metal with respect to the composition may take place. This problem is overcome in modern designs by using an induction stirrer at the bottom of the furnace from outside. An induction stirrer is, however, quite costly and weighty to be adopted on all furnaces.

11.8.7. Finishing and Tapping of a Double Slag Heat

It is possible to adjust the bath composition, under reducing slag, closer to the specification levels. The finishing job is not distinctively different from refining in this respect. Since the alloying additions are made under reducing conditions, and that enough time is available for slag-metal separation, the product is cleaner. As the analyses are confirmed, the heat is tapped as described before. Care should be taken to avoid non-metallic pick-up from runners, ladles, molds, etc., during teeming.

CHAPTER 12

ALLOY STEEL MAKING—CONVENTIONAL ELECTRIC FURNACE PRACTICES

In This Chapter

- Induction Furnace Practice
- Arc Furnace Practice for Carbon and Low Alloy Ingot Steels
- Arc Furnace Practice for Carbon and Low-Alloy Steels for Castings
- Arc Furnace Practice for Tool Steels and Special Alloy Steels
- Scope of the Process

The principles of the various methods of alloy steel making have been described in Section 4.7. Alloy steel making practice in an electric furnace depends on the type of furnace employed, the quality of the charge used, the product to be made, and so on. The practice also varies for wrought and cast alloy steel production. For the sake of simplicity the description has been subdivided under various broad categories that are described below.

12.1. Induction Furnace Practice

The induction furnace was first patented by Ferranti in Italy in 1877. It used a low frequency AC power source as did all the earlier designs including those by Colby and Kjellin. A furnace of this type consists of an annular channel surrounding a core of laminated iron sheets. Molten metal is always present in the channel to act as a secondary winding of just one turn. The core is surrounded by the primary winding. Because of this type of design these furnaces, using low frequency power sources, are known as core-type furnaces. Such core-type furnaces were used until recently in India in foundries for producing steel castings. The capital cost is relatively low compared to induction furnaces. The slow rate of melting that is inevitable in core-type furnaces is advantageously used to produce steel castings economically. The rate of heat generation, besides other factors, directly depends upon the frequency of the power source. This fact was realized and put into practice by Dr. Northrup of Princeton University by designing commercial furnaces using power sources of frequency 500-2500 c/sec. (by 1925). The modern furnaces use currents of high frequency in excess of 10 Kc/sec. up to the level of radio frequency. High frequency furnaces do not require the core-type design. For steel melting, medium and high frequency induction furnaces are universally adopted, hence are described below. These units use multi-conductors, motor generators sets, spark gap converters or oscillators. The output may be at up to several thousand volts and 100 Mc/sec. with a total power output of up to 100 kW. In general, high frequency furnaces are costly but are more efficient in melting than low frequency furnaces.

The furnace is shown in, Figure 12.1. It consists of a crucible, usually of a magnesite monolithic construction, with a spout for pouring. The charge is placed in this crucible and it acts as the secondary winding.

The crucible is surrounded by several turns of water-cooled copper tubing that carries the high frequency primary current. The size of the furnace, that is of the crucible, varies from a few kilos to several metric tons. Once 0.5 to 2.0 t capacity furnaces were more popular. But now up to 5-20 ton size furnaces are more popularly used for steel making. The furnace capacity is limited depending upon whether it is to be used as a melting unit or as merely a holding furnace for molten steel. A maximum of 200 t holding furnace is in use. The holding furnace design is very commonly used when secondary steel making is adopted, wherein already molten steel is to be heated during secondary refining.

The furnace is generally charged manually but simple mechanical charging devices are not uncommon. The charge consists of graded steel scrap. It is essentially a process of melting, and hardly any refining takes

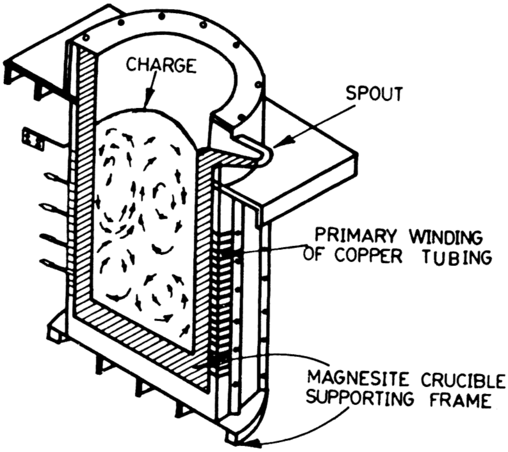


FIGURE 12.1. High frequency induction furnace.

place. It is a process wherein *what goes in must come out.* A thin layer of slag is maintained on the surface to prevent oxidation of the bath by atmospheric oxygen. Slag is an insulator and is not heated by induction. It is, therefore, usually very dry and it does not take part in refining.

The operation is quite simple. Light scrap is charged at the bottom and heavy at the top to prevent atmospheric oxidation of the scrap, as far as possible. The charge must be of accurately known composition since there is generally no time provision for analysis of the sample during the heat. The bath analysis is controlled by the charge composition. After melting, necessary alloy additions are made to meet the specifications. As the temperature reaches the required value it is tapped in a teeming ladle or directly in molds to produce castings.

At times a gentle boil is allowed to take place, and for this iron ore has to be incorporated in the charge. Alternatively oxygen may be sprayed over the bath to carry out the refining slowly, if needed.

Excellent stirring action and very low oxidation loss of metals and alloys make this process useful in reutilizing the well graded scrap to produce the same or nearly the same type of steel without much external virgin alloy additions. Since usually no oxidation is carried out, the steel bath is not deoxidized to any appreciable extent. It gives better cleanliness, which is further improved by adopting direct pouring in molds to produce ingots or castings. The temperature is also under very effective control.

The process is equally suited to produce any type of alloy steels on a small scale. In fact it is practically the only commercial process for making steels on a small scale. It is commonly used in alloy steel foundries. The crucible can be enclosed in a vacuum chamber, thereby better quality steels of wide specifications can be produced on a small scale. This is known as vacuum induction melting.

It has been estimated that nearly 60% of the energy input is usefully used in an induction furnace as against 65-70% in an arc furnace. The power consumption varies from 900 kwh/t for a 0.5 t furnace to 650 kwh/t for a 5 t furnace. Although it is costlier than an arc furnace, its flexibility to produce different steels in small lots is unsurpassed even by an arc furnace; that is why it still withstands competition from an arc furnace.

12.2. Arc Furnace Practice for Carbon and Low Alloy Ingot Steels

In general a two slag practice is adopted. Although good quality scrap charge is beneficial, dirty and rusty scrap in certain proportions may be used. As far as possible the scrap should be segregated to have a composition closer to the specification level. Whatever excess carbon is required at the melt-out for subsequent boil, it is added in the charge itself. Although the charge proportion varies considerably from plant to plant, ideally it should contain 35% heavy scrap, 40% medium, and 25% light scrap. The proportion should be such that charging is over in the beginning. If this is not possible, a minimum of back charging is adopted. The power and the electrode consumption are maximum during melting and hence this period should be bare minimum. Many operators prefer to charge lime and ore only after two-thirds of the charge is molten. Others add it partly with the charge and the remainder later. The lime consumption varies from 3-5% of the charge weight. The lime addition should not be delayed, otherwise the acidic impurities may attack the lining. During oxidizing refining, impurities such as P, Si, C, and Mn are oxidized. Any oxidizable alloying elements in the scrap are also lost to the slag. Initially the temperature of the bath is low and dephosphorization is possible even by a low basicity slag. As the refining continues and the bath temperature rises, iron oxide content of the slag decreases (supply of oxygen is limited) and sufficient lime is added to retain phosphorus in the slag.

If silicon is high, slag-off is carried out in the oxidizing period. As the carbon falls to just below the specification level the oxidizing period should be over. The boil helps the heat as it does in an open-hearth furnace.

Deoxidation prior to slag-off by Fe-Mn addition brings the bath-oxygen-level in equilibrium with the carbon content of the bath. The slag is then removed and the bath is further deoxidized by Fe-Si and Al.

Reducing slag is made by the addition of lime, spar, and coke and, it may take nearly 30 minutes to shape this slag. The duration of reducing period depends on the sulfur content of the charge, slag basicity during earlier oxidizing period, and the sulfur specification level required. The reducing period may be up to two hours. Extra care is needed to effect desulfurization if carbon level is below 0.3%. Extra addition of Fe-Si helps to desulfurize the bath. The (FeO) content of slag is maintained below 0.5% throughout the reducing period. Final test analysis is obtained and the requisite alloying additions are made. Bath temperature is checked and final deoxidation with Fe-Si and Al is carried out 15 minutes before tapping. The heat is then tapped.

The ladle deoxidation practice mainly depends upon the bath carbon content, the grain size required in the solidified ingot and other additions, if any, such as Zr, B, Ti, V, etc., to be made in the ladle. For coarse-grained steel no Al is added. For fine-grained, medium carbon steel nearly 0.3 kg of Al per metric ton is added and for fine-grained low-carbon steel 0.5 kg of Al per metric ton is added. For alloy heats as much as 2-2.5 kg of Ca-Si alloy per metric ton is added in the ladle uniformly throughout the tapping to obtain most effective deoxidation of the bath.

Typical log-sheets are shown in Tables 12.1 and 12.2.

Table 12.1
Log-Sheet of Low-Alloy Steel Heat Furnace size 5 t.

	En. 18 D	*Specifications* *C 0.38/0.43, Mn 0.65/0.80, Si 0.20/0.35* *P and S each max. 0.05, Cr. 0.85/1.15*	
Time Hrs.	*Operation*	*Remarks*	
	Charge:	Heavy scrap	3508 kg
		Turnings	1170 kg
		Total limestone	400 kg

1110 Power on

1300 All melt

Analysis:

C	Mn	Si	S	P	Cr
0.26	0.25	0.25	0.04	0.03	-

Slag:- 40 kg iron ore + limestone + spar

Analysis:

C	Mn	Si	S	P	Cr
012	015	-	0.022	0.01	-

1345 Slag off-complete

Temp. 1620°C

Bare bath additions:

Coke fines	15 kg
Fe-Mn	20 kg
Fe-Si	20 kg

Reducing slag additions —Fe-Si, coke fines, spar and limestone.

Alloy additions:

Fe-Cr (HC)	62 kg
Fe-Si	5 kg + 9 kg
Fe-Mn (HC)	14 kg

Analysis:

C	Mn	Si	S	P	Cr
0.35	0.75	0.27	0.013	0.017	0.96

Final adjustments : Fe-Cr (HC) 4 kg

Coke fines 1 kg

Aluminum—4 kg

1435 Silicon cup test O.K., Temp. 1595°C

1455 Tapping Temp. 1595°C

Analysis:

C	Mn	Si	S	P	Cr	Al
0.42	0.80	0.34	0.012	0.012	1.07	0.015

Tapping slag Color—Brownish/Greenish

Nature—Falling

Ladle additions 1 kg Aluminium

Pit analysis:

C	Mn	Si	S	P	Cr	Al
0.38	0.79	0.32	0.011	0.014	1.07	0.034

Table 12.2
Log-Sheet of En 24 Steel Heat
Furnace size 5 t

En 24-Specification (%):

C 0.35/0.45, Si	*Mn 0.45/0 79 P 0.05*
0.20/0.35, S 0.05	*max Cr 0.90/1.40*
max Ni 1.30/1.80,	*Mo 0.20/0.35*

Time Hrs.	*Operation*		*Remarks*					
0820	Power on		Charge:					
1015	All melt		Turnings			570 kg		
			Plant reverts			4630 kg		
						5200 kg		
1035	Analysis :							
	C	Mn	Si	S	P	Ni	Cr	Mn
	0.27	0.12	-	0.025	0.02	0.44	0.21	0.09
	Refining addition	Iron ore Lime stone spar		40 kg as required				
1055 Hrs.	Alloy addition		Fe-Mo	10 kg				
			Ni	50 kg				
1115	Analysis							
	C	Mn	P	S	Ni	Cr	Mo	
	0.14	0.25	0.07	0.024	1.53	0.52	0.24	

1120 Complete slag off — Slag off temp. 1640°C

1130 Bare bath additions

Coke power	6 kg
Fe-Mn	15 kg
Fe-Si	20 kg

1145 Analysis

C	Mn	Si	S	P	Ni	Cr	Mo
0.19	0.50	0.27	0.021	0.022	1.51	0.56	0.24

1155 Additions before alloy test:

Fe-Cr (HC)	40 kg
Fe-Mn	2 kg
Fe-Si	5 kg

1205 Analysis

C	Mn	Si	S	P	Ni	Cr	Mn
0.25	0.55	0.35	0.017	0.023	1.43	1.22	0.24

Temp. at alloy test 1585°C

Adjustments

Fe-Cr (HC)	5 kg
Fe-Mo	2 kg

Raking

Reducing slag is partially removed.

Fresh reducing slag is made.

Analysis:

C	Mn	Si	S	P	Ni	Cr	Mo
0.37	0.56	0.36	0.014	0.024	1.43	1.21	0.24

Aluminum 3 kg

1305 Tapping — Tapping temp. 1610°C

Tapping slag color greenish white

Nature—Falling

Ladle additions :

Aluminum	0.05 kg
Coke powder	1.0 kg

Pit analysis

C	Mn	Si	S	P	Cr	Ni	Mo
0.40	0.56	0.34	0.013	0.024	1.22	1.43	0.24

Total limestone consumed 504 kg

Tap-to-tap time 5 Hrs, 10 min

12.3. Arc Furnace Practice for Carbon and Low-Alloy Steels for Castings

Use of acid lined arc furnaces is popular for steel foundries. The acid lining is better for intermittent operation of the furnace than basic lining. Acid furnace is cheaper to construct and operate than basic furnace. The only limitation being that good quality scrap, low in phosphorus and sulfur contents, is required as a charge in an acid furnace.

Acid furnaces are generally small in capacities. A 0.5 to 3.0 t capacities are more common. These furnaces are seldom operated at rated capacity figures. The charge may be 1.3-2.0 times the rated capacity.

The furnace is lined by a permanent acid lining and the working hearth is a monolith of moist mixture of fireclay and ganister, rammed in cold. The side walls as well as roof are of silica bricks. Roof life is nearly 300 heats and hearth life is 200-400 heats depending upon the plant design and practice.

The charge consists of 30-40% foundry returns and the remainder as acid steel scrap. A little of low-phos pigs may be added to obtain carbon necessary for carbon boil. The charge should be free of excessive rust and oil. The charge is put in the furnace via a chute or drop bottom basket. The general operational details described in the previous chapter hold true in this practice. For the production of non-graded castings, partial oxidation of the melt is achieved during melting. The universal practice followed in foundries is, however, to obtain fully oxidized melt-out stage. Ore is used for oxidizing Si, Mn, C, and is added only after two-thirds of the charge is molten. Slag forming and shaping largely resembles the acid open-hearth practice. Refining in an acid process is faster than in a basic process. Final slag may contain 20% FeO, 55% SiO_2, and 15% MnO. The refining period is usually equivalent to 0.2% carbon removal to arrive at the specification level.

The metal bath is fully deoxidized in the furnace using stronger deoxidizer than silicon to ensure minimum of porosity in the finished castings. In many foundries Ca-Si or Ca-Si-Mn alloy is added in the ladle, while tapping, to ensure complete deoxidation. The tapping temperature is usually around 1650°C to permit metal movements in small pouring ladles and the running of thin sections in castings. The gas content of an acid electric steel is quite low.

Even basic lined furnaces are used in steel foundries if the scrap is high in phosphorus. Both single and double slag practices are employed. The acid and basic operations in foundries are quite similar in terms of charging, melting, deoxidation, etc. In the basic operation the usual care for dephosphorization, by deslagging, is normally adopted. In both the cases the product is a dead killed steel. The details of the practice are the same as described in the previous chapter.

A foundry may produce up to a dozen different alloy steels in one day. These may also be produced in small lots as per order. The sequencing of the different types of heats should be carefully planned. The fining does absorb some alloying elements and may desorb the same in the subsequent heat unless proper care is taken in planning the sequence of the various different types of heats.

12.4. Arc Furnace Practice for Tool Steels and Special Alloy Steels

Although varieties of tool steels are specified, these can broadly be classified as shown below:

Water hardening:	Straight carbon up to 1.4%
Shock resisting:	Medium C, with Si and Mo.
Cold work:	High C, high Cr, high V, and Mo.
High speed:	High C, high Cr, W and V, and may contain Mo and Co.
Special purpose:	Very low C, low Cr, Ni and Mo.

In essence, substantial quantities of carbide-forming elements and solid solution-forming elements are added as alloying additions in the production of tool steels. The special alloy steels are equally varied in specifications and from the point of view of melting practices may be classified as:

Constructional alloy steels —low Cr, Mn, Mo, etc.
Carburizing steels —low C, nickel bearing and may contain Cr and Mo.
High carbon bearing steels —High C and low Cr.
High strength heat resisting steels —Low C, Cr and Mo.
Special alloy types —Ni-steels, Al-steels, the nitralloys, etc.

All these steels are characterized by their high quality levels. Their specifications insist on the chemical limits, porosity standards, cleanliness, surface quality, carbide form and its distribution, hardenability, hot workability, etc. The number, shape, size, type, and distribution of the non-metallic inclusions are all taken into account in deciding the cleanliness. This is the dominant factor that decides the quality of any steel because the mechanical properties are very much affected by each one of these parameters. A dirty steel is more susceptible to fatigue failure than a clean steel. An inclusion distribution acceptable under low stress distribution may become unacceptable under high stress. The raw materials, melting practice, tapping temperature, mold design, teeming practice, etc., all influence the quality of the product.

If the charge is of poor quality, double oxidizing slag practice is employed. A single oxidizing converted to reducing slag practice is good enough for charge containing valuable alloying elements. The scrap must be low in residual elements and a proper balance of light and heavy scrap is made to expedite melting. Special ores of V, W, Mo, Ni, etc., may be used in place of their standard grade metals and ferro-alloys, to accomplish partial alloying.

In a single slag practice the early oxidizing slag is converted to a reducing slag towards the end of melting. In this case alloying additions of W, Cr, Mo, Ni, Co, etc., are made with the charge itself. Vanadium is added when the slag is carbidic; Si and Mn are added after the preliminary analysis is known.

In a double oxidizing slag practice the oxidizing slag of 60% CaO and 15% FeO composition is obtained in 15-30 minutes after clear melting. This gives a good boil and oxidizes all the phosphorus, silicon and about 10-20 points of carbon. If chromium runs high it should be oxidized at this stage. Low temperature favors oxidation of P and Cr to that of carbon and vice versa. If residual P and/or Cr are high, oxidizing slag flushings may be resorted to, in order to bring it down. By the time all the P and Cr are eliminated the boil should attain necessary vigor to clean up the bath. Over-oxidation needs to be carefully prevented. Many operators allow the bath to be decarburized to 10-15 points below the specification with the expectation that alloy additions would make it up later. This slag is raked off and a reducing slag is made. Metal oxides are now added to the slag. These are reduced at different rates but the reduction should be complete in about a maximum of two hours.

Vigorous stirring is resorted to obtain correct bath samples for analysis. The heat is temporarily blocked to obtain a sample for preliminary analysis. Corrective alloy additions are calculated from the preliminaries and are added. Bath is fully deoxidized by Fe-Si or Al. Deoxidation by aluminum is almost a must. It may be carried out partly inside the furnace and partly while tapping or occasionally while teeming. Ladle additions are kept to a minimum to improve cleanliness of steels.

Each type of these special alloy steels has its own characteristic viscosity and best pouring temperature, which is related to the teeming nozzle size, ingot size, and heat size.

The detailed specification of tool and special steels cannot be met only by proper melting and refining practice but also by proper mold design, hot topping, mold washing, teeming, stripping practices, and so on.

The problem of carryover of alloys from one to the next heat should always be minimized by proper sequencing of different alloy steel heats.

Three typical log-sheets of tool and special steels are shown in Tables 12.3, 12.4, and 12.5 as illustrations.

Table 12.3
Log-sheet of Ball Bearing Steel-Heat
Furnace size 5 t

	Specifications:		
SAE 521000	*C 0.92/1.10,* *S 0.25 max.*	*Mn 0.25/0,45,* *P 0.025 max.*	*Si 0.20/0.35* *Cr 1.30/1.60*
Time Hr.	*Operation*	*Remarks*	
1215	Power on	Charge	
		Heavy scrap	1788 kg
		Turning	1305 kg

Revert scrap (SAE 52100 + Plain C)	2000 kg
Hard coke (bottom)	140 kg
Dolomite	50 kg
Limestone (total)	450 kg

1430 All melt analysis Iron ore-24 kg + Limestone + Spar

	C	Si	Mn	S	P	Cr
Sample 1	0.58	0.07	0.32	0.036	0.027	0.23

Iron ore-50 kg + Limestone + Spar

	C	Si	Mn	S	P	Cr	Mo
Sample 2	0.89	0.04	0.25	0.037	0.022	0.32	0.03

1530 Complete slag off

Bare bath addition	Fe-Si	20 kg
	Fe-Mn	10 kg
Reducing slag	Fe-Si	1.5 kg
	Coke-fines	1 kg
	Spar	1.5 kg
	Aluminum (Stars)	2 kg
	Fe-Cr (HC)	65 kg

Analysis

C	Mn	Si	S	P	Cr
0.97	0.26	0.36	0.02	0.02	1.32

Temp. 1525°C

	Adjustments	Fe-Cr (HC)	10 kg
	Refining	Fe-Mn	3 kg
1620		Added Aluminum	3.5 kg
	Silicon cup O.K.	Temp. 1565°C	
1645	Tapping	Temp. 1565°C	

Pit analysis

C	Mn	Si	P	S	Cr
0.98	0.34	0.30	0.02	0.012	1.44

Tap-to-tap time 5 hr.

Table 12.4
Log-sheet of a Valve-steel Heat Furnace capacity 5 t

	En 52 Specifications:	*C 0.40/0.50, P 0.04 max Cr 7.50/9.50*	*Mn 0.30/0.60 S 0.04 max Si 3.00/3.75*
Time Hr.	*Operation*	*Remarks*	
	Charge	Heavy scrap	1950 kg
		Bundles	450 "
		Turnings	540 "
		Reverts (1% Cr, 1.5% Cr)	1180 "
			Total 4120 kg
	Lime stone		
0620	Power on		
	All melt		
	Analysis		

C	Si	Mn	S	P	Ni	Cr
0.48	-	0.09	0.05	0.015	0.11	0.10

	Refining (Boiling)	Iron ore	75 kg
		Lime stone	-
		Spar	-
	Analysis after boil		

C	S	P	Ni	Cr
0.16	0.045	0.006	0.10	Nil

		Coke fines	
		Calcium carbide 10 kg	
	Analysis before slag off		

C	Si	Mn	S	P	Ni	Cr
0.16	0.03	Nil	0.03	0.006	0.11	Nil

0955	Slag off complete		
	Bare bath additions	Fe-Mn	5 kg
		Fe-Si	30 kg

	Additions before alloy test		Fe-Si			300 kg (60%)
			Fe-Cr (LC)			450 kg
	Analysis					
	C	Mn	Si	S	P	Cr
	0.42	0.35	3.40	0.036	0.036	7.60
	Final adjustments		Aluminum			1.5 kg
	Analysis before tap					
	C	Si	Mn	S	P	Cr
	0.82	3.15	0.37	0.41	0.41	7.70
	Tapping slag		Color -			O.K.
			Nature -			falling
1140	Tapping		Temp. 1555°C			
	Ladle additions		Fe-Si 20 kg (75%)			
			Fe-Cr (HC) 65 kg			
	Pit analysis					
	C	Si	Mn	S	P	Cr
	0.46	3.20	0.45	0.03	0.03	8.33
	Total lime added		600 kg			
	Tap-to-tap time		5 Hours 40 min.			

Table 12.5
Log-sheet of High-Speed Steel Heat

	HSS 6-5-2	*Furnace size: 5 t* *Specifications* C 0.80/0.90 Mn 0.20/0.40, Si 0.20/0.40 P and S each 0.025 max W 5.50/6.70, Mo 4.50/5.50, V 1.80/2.20, Cr 3.80/4.50
Time Hours	*Operation*	*Remarks*
	Charge HSS reverts (6-5-2)	2800 kg

Plant reverts 1240 kg
(SAE 52100)
0625 Power
All melt
Analysis

C	Mn	Si	P	S	Ni	Cr	Mo	V	W
0.80	0.26	0.26	0.034	0.025	0.23	2.71	5.71	0.81	7.27

Refining Iron ore-30 kg + Limestone
0845 Temp. 1480°C
Temp. 1560°C
Slag off complete
2nd oxidizing refining slag Lime + Iron ore 15 kg
0925 Analysis

C	Mn	Si	P	S	Ni	Cr	Mo	V	W
0.76	0.20	0.08	0.03	0.02	0.23	2.36	4.96	0.48	7.29

Partial slag off—second slag sample sent
0945 Analysis

C	Mn	Si	P	S	Ni	Cr	Mo	V	W
0.79	0.17	0.05	0.03	0.02	0.23	2.24	5.49	0.34	6.50

Temp. 158°C
Iron ore 18 kg
0955 Temp. 1610°C Iron ore 12 kg
1015 Slag off
1020 Slag addition Iron ore 30 kg
Lime 46 kg
Spar 14 kg

Bath raked
1025 Tamp. 1595°C Iron ore 15 kg
Lime 21 kg
Spar 7 kg

Analysis

C	Mn	Si	P	S	Ni	Cr	Mo	V	W
0.76	0.12	0.02	0.26	0.02	0.23	1.91	5.41	0.18	5.66

1027	Reducers added	C-Si + A1…2.5 kg	
1030	Complete slag off	Slag sample sent	
1038	Reducing slag	Fe-Si	10 kg
		Limestone	30 kg
		Spar	20 kg
		Spar + Coke	5 kg
1042		Add Fe-Cr (HC)	80 kg + 15 kg

1045 Temp. 1530°C

Analysis

C	Mn	Si	P	S	Ni	Cr	Mo	V	W
0.79	0.14	0.27	0.026	0.016	0.20	2.69	5.37	0.18	5.99

1046	Add reducers	Fe-Si (75%)	5 kg
		CaC_2	2 kg
1048		CaC_2	1 kg
		Spar + Coke	1 kg
1055	Alloy additions	Fe-W	30 kg
		Fe-V	165 kg
		(HC) Fe-Cr	55 kg

1109 Temp. 1490°

Analysis

1110	C	Mn	Si	P	S	Ni	Cr	Mo	V	W
	0.85	0.18	0.37	0.03	0.015	0.20	3.91	4.74	2.05	5.45
	0.86	0.19	0.35	0.032	0.015	0.20	4.10	4.53	2.04	5.56

1128	Adjustments	Fe-W	5 kg
		Fe-Mo	20 "
		Fe-Mn	5 "
		Fe-W	5 "
		Fe-Cr	7 " (HC)

1130 Temp. 1550°C

1135 Tapping Analysis

C	Mn	Si	P	S	Ni	Cr	Mo	V	W
0.90	0.27	0.35	0.031	0.013	0.19	4.15	4.75	2.00	5.70

Total additions in kg

Lime stone	500	Fe-Mn	5
Iron ore	100	Fe-Si	15
Spar	80	(HC) Fe-Cr	157
Carbide	15	Fe-V	165
Dolomite	50	Fe-Mo	200
Coke powder	5	Fe-W	215
Slag			
	Al	3	
	Ca-Si	3	
With scrap			
Fe-W	175		
Fe-Mo	180		

NOTE: Since the opening carbon is rather on the high side the oxidizing refining period is somewhat longer than usual.

12.4.1. Conventional Arc Furnace Practice of Stainless Steel Making

Steels containing 12-30% chromium with or without other alloying elements are classed as stainless steels. A vast majority of these steels must contain carbon below 0.12%. A large proportion in fact contains 0.07-0.08% C and the Ni-Cr series is made with only 0.03% C.

In the early days stainless steel was made by the addition of low carbon Fe-Cr to a refined plain carbon steel bath to make up the entire chromium specifications. It was thus a very costly method. Stainless steel scrap could not be dead melted without carbon pick-up from the arc furnace electrode. As a result the scrap could not be reused economically. It piled up to such an extent that it became imperative (for the Ni and Cr values in the scrap) to develop a process of melting the scrap to produce low-carbon stainless steels. By 1931 a process was developed wherein carbon was oxidized and whatever chromium oxidized concurrently was later on reduced back to the metal phase. The process soon came to be known as the rustless process, which is still being used when oxygen gas is not available for refining. The furnace bottom and side walls to a height just above the slag line were made of chrome brick to withstand the chemical action of superheated iron oxide-rich slags. Chrome ore used for fettling the walls soon become a major source of chromium.

Typically the charge consisted of such ingredients that one-third of the specification chromium was obtained from high carbon ferro-chrome, one-third from chrome ore, and the remaining one-third from the stainless steel scrap. Chrome ore is evenly charged on the bottom and along the banks. Light scrap is charged first on the charge-chrome ore and then the heavy scrap followed by iron ore, anything up to 15-20% of the heat weight. High carbon Fe-Cr is charged above the iron ore followed by a little of light scrap. The reason for adding a large excess of iron ore will be clear from the following:

The free energy temperature relationship of the reactions

$$\frac{3}{4} <\mathrm{Cr}> + \{\mathrm{O_2}\} = \frac{2}{3} <\mathrm{Cr_2\,O_3}> \qquad \ldots(12.1)$$

and

$$2 <\mathrm{C}>\{\mathrm{O_2}\} = 2\{\mathrm{CO}\} \qquad \ldots(12.2)$$

indicate that they are at equilibrium at 1220°C, i.e., the free energy change of the reaction (avoiding fractional powers)

$$3\,\{\mathrm{CO}\} + 2 <\mathrm{Cr}> = <\mathrm{Cr_2\,O_3}> + 3 <\mathrm{C}> \qquad \ldots(12.3)$$

is zero at 1220°C. The equilibrium constant is

$$\mathrm{K}_{(12\cdot3)} = \frac{{}^{a}< \mathrm{Cr_2O_3} >{}^{a^3}\cdot< \mathrm{C} >}{{}^{a^2}< \mathrm{Cr} >\cdot{}^{p^3}\{\mathrm{CO}\}}$$

If steel containing low carbon is required, the value of $\mathrm{K}_{(12.3)}$ should be very much lower than unity. The equilibrium contents of carbon and chromium in steel in contact with pure $\mathrm{Cr_2O_3}$ at $p_{\mathrm{CO}} = 1$ atoms, at various temperatures are shown in Figure 12.2. During refining the electric arc furnace slag is almost saturated with chromium oxide and the ambient atmosphere can be assumed to be of CO at one atmospheric pressure. The condition in an arc furnace thus is close to those governing Figure 12.2. The data

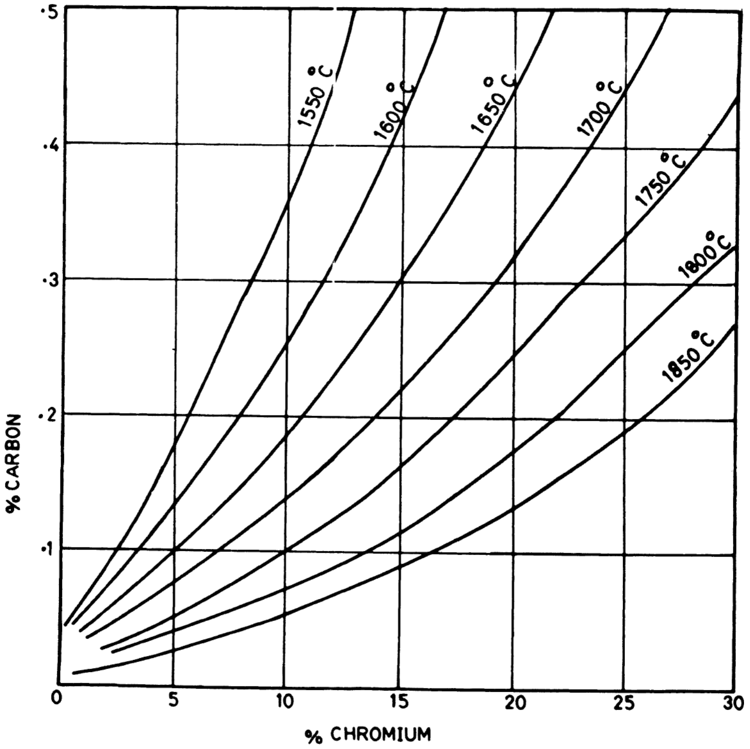


FIGURE 12.2. Equilibrium carbon and chromium contents of steel in contact with pure Cr_2O_3 under carbon monoxide at one atmosphere at various temperatures. (after Hilty[16]).

predict that at 1600°C, for a Fe-C-Cr melt containing 10% chromium, the equilibrium carbon content is 0.22% which is reduced to 0.05% at 1800°C. The carbon in the melt can, therefore, be preferentially oxidized from a Fe-C-Cr bath only at much higher temperatures.

The liberal use of iron oxide in the charge in the rustless process oxidizes not only carbon during refining, but it also oxidizes a lot of chromium from the bath to the slag. This develops the necessary heat to raise the bath temperature. When carbon is down below the specification level, the oxidizing conditions are replaced by reducing conditions to win back chromium from the slag. Fe-Si and Al are added to the bath and it reduces Cr_2O_3 as

$$(Cr_2O_3) + \frac{2}{3}[Si] = 2[Cr] + \frac{3}{2}(SiO_2) \qquad \ldots(12.4)$$

Taking equilibrium constant and rearranging the terms

$$\frac{{}^{a}(\mathrm{Cr_2O_3})}{{}^{a^2}[\mathrm{Cr}]} = \frac{1}{\mathrm{K}_{(14\cdot4)}} \cdot \frac{{}^{a^{3/2}}(\mathrm{SiO_2})}{{}^{a^{3/2}}[\mathrm{Si}]} \qquad \ldots(12.5)$$

It may also be approximately written as

$$\frac{(\%\mathrm{Cr})}{[\%\mathrm{Cr}]} \propto \frac{{}^{h}(\mathrm{SiO_2})}{{}^{h}[\mathrm{Si}]} \qquad \ldots(12.6)$$

In basic slags the activity of silica is very low and hence the chromium distribution is favored in the direction of the metal bath. Kinetically chromium transfer across slag-metal interface is quite rapid and hence much of the chromium from the slag is reduced back to the metal phase. Fe-Si addition is very effective in the case of basic slags. In acid slags it has hardly any effect, and hence a stronger deoxidizer like Al is used when slag is acidic in nature.

Being a basic oxide the activity coefficient of (Cr_2O_3) is much higher in basic slag than in acid slag, and hence Cr gets distributed more in favor of metal under a basic slag than under an acid slag. In fact the higher the basicity, up to a certain limit, the better is the chromium recovery in metal phase.

After the preliminary analysis is known, final adjustment of the chemistry of the bath is done, using low carbon Fe-C. The heat is tapped after final temperature adjustment.

In practice it is aimed at obtaining the desired carbon content of the bath by the time the charge is melted down. Rapid melting and liberal iron oxide in the charge are useful in this regard. This is a very cheap process as compared to the early method of making stainless steel because cheaper sources of chromium are used for obtaining the necessary alloying content.

12.5. Scope of the Process

Electric furnaces, both the arc and the induction types, are best suited for the production of medium and high alloy steels from any quality of all solid charges. These are the most economical methods of utilizing the alloy steel scrap. In steel foundries electric furnaces are practically the only choice for making varieties of plain carbon or alloy steels in small or large quantities for founding. Although oxygen steel making processes are fast becoming competitors in the field of alloy steel making on large scale, the electric furnace will still remain unrivalled for alloy steel making from all solid charge, more particularly, on a small scale.

The advent of oxygen steel making has revolutionized the electric arc furnace practice, and because of this, when open-hearth process is on its way out, it can even compete with other processes in bulk steel production particularly from all solid charge.

Many of the old open-hearth shops have been replaced by the electric furnaces to deal with all cold charges. There is not going to be any reversal now to the old style open-hearth practice. The electric furnace practice can be suitably modified to include the secondary steel making technology, not only for increasing the productivity of the shop, but also improving upon the quality of steels. These modifications will be described later.

CHAPTER 13

APPROACH OF MODERN STEEL MAKING

In This Chapter

- The Paradigm Shift
- Philosophy of Achieving Modern Steel Specification
- System Approach
- External Desiliconization
- External Desulfurization
- External Dephosphorization
- Concurrent Removal of Si, S, and P
- Basics of Modern Steel Making

13.1 The Paradigm Shift

Until the end of the 19th century, the scientific concepts, developed primarily by Newton and R. Descartes were so dominating that everything else was, at least attempted to be, explained in terms of those basic concepts or under their shadows. These ideas were essentially based on a linear interpretation of everything viewed in a mechanistic way, using deductive methods. This interpretation accepted that the parts of this universe, and therefore of everything else therein, were discrete and not interdependent in any way. This dominant approach influenced the technology in general as consisting of discrete processes for ultimate production of the required product, and steel production was no exception. Steel making was therefore viewed as consisting of discrete steps of ironmaking and steel making, coupled with rolling, for the production of desired quality of steel at the end.

The situation altogether changed with the acceptance of the Einsteinian model of the universe, from the beginning of the 20th century, wherein everything was considered to be intimately interrelated to each other, in the form of a network. This *holistic* approach to science was a paradigm shift in scientific thinking and it had its profound influence on technology as well. Although the Newtonian-Descartesian influence continued to haunt even thereafter, World War II proved to be the turning point. The technological perception drastically changed thereafter and the holistic approach dominated the production activities. The total production process invariably came to be viewed as a single system, to be carried out in one or more interdependent process steps, to ensure better economy and productivity of the final product.

As a result, iron and steel making and subsequent rolling for producing steel semis were viewed as a single system of production from the given raw materials like iron ore, coal, and the like. The individual processing steps could thus be increased by differentiation of existing activities, or decreased by their amalgamation to finally produce the desired steel product, most efficiently and economically. Modern steel making has evolved out of this basic fundamental approach. The earlier simple process of ironmaking as one stage and steel making as another, thus became more and more complicated and intricately interdependent, with additional interpolated activities, while still becoming more efficient. The complexity increasingly became raw material-specific and demand-specific. The modern steel industry exhibits all these characteristic approaches.

13.2. Philosophy of Achieving Modern Steel Specification

The introduction of oxygen steel making after World War II, once again revolutionized steel making after Henry Bessemer's first revolution way back in the 1850s. Although the changeover did not occur at once, it rapidly evolved into a very different strategy of steel making. One of the dominant reasons that contributed to such paradigm change was the increasingly lower impurity levels, like the contents of S, P, Si, C, Mn, gas contents, and so on, required in final steel specifications. The economic considerations for achieving these contributed immensely to the evolution of these modern steel making practices. Broadly only two routes were left for steel making over the time for producing steel from the iron ore. One route was the traditional route of BF and BOF/OBM, and the other was the DRI and EAF/IF route.

During this period, besides the earlier known S and P impurities, Si and even C, and gases like H, N, and O were also categorized as impurities, to be removed to the specified lower levels of specifications. The refining strategies had to be altered to suit these end requirements. To the earlier list of impurities in steel, additional elements were not only added but their individual actual levels were lowered to improve upon the properties of steel as desired for their applications and better performance.

During this time the sulfur specifications, in general, were lowered down from earlier 0.05% to around 0.005% and in special cases to as low as 10 ppm. A blast furnace cannot be run to produce hot metal with low enough sulfur, for finally producing such low sulfur specifications. It is neither economical nor easy to try to remove sulfur under the usual oxidizing conditions of steel making to such low levels. Similarly phosphorus cannot be readily reduced from the earlier 0.05 to 0.01% without compromising the productivity and economy of steel making.

Oxygen steel making could readily reduce carbon to 0.03-04%. Further reduction was however possible only by adoption of vacuum decarburization, and for which several techniques had to be developed. Silicon is readily oxidized in oxygen steel making to a level of 0.05% because of high stability of silica in basic slag. The real problem however was posed by the precarious nature of both S and P and also because of their peculiar thermodynamics, as well as kinetics of refining in steel making. Because of their extraordinarily high interaction with iron, they preferably remain with iron, unless specific efforts are made to remove them. The lower their specifications, the more difficult it is to meet them, and the operation also becomes costly.

The acceptable levels of gases like H, N, and O could not be met even by the oxygen steel making processes alone. Vacuum treatments had to be evolved to meet their specifications. A very low level of oxygen could be obtained only by adoption of strong deoxidizers, but not without their attendant cleanliness problems, and for which additional treatments had to be devised to produce clean steels.

13.3. System Approach

Previously steel was made broadly in two steps, viz. iron making, and steel making. Sometimes the steel making was further subdivided to carry out refining in two or more stages like in duplexing and triplexing, to take care of the extra silicon content of the hot metal or to improve the productivity of slower hearth processes of steel making. On the whole, modern more stringent steel specifications could not be met by these established routes alone. The newly developed concept of the *systems approach* to iron and steel making is now increasingly being adopted. The systems approach gave rise to splitting the total operations into as many sub-steps as needed, achieving at the same time, overall better productivity and economy without impairing the desired quality of the final product, in a given situation. This approach could be thought of as splitting the refining, in more than one stage, to achieve the desired end specifications.

Since the quality of hot metal primarily, or the charge materials as such, finally decides the efficiency and economy of steel production, the stringent steel specifications could only be met by introducing one or more prior treatments of hot metal, for making it more suitable for such modern steel making practices. Similarly the DRI or the steel scrap is to be processed prior to steel making. The hot metal pre-treatments and their objectives are broadly:

(*a*) to remove sulfur from hot metal to nearly the required level of steel specifications thereby nearly eliminating the need for sulfur removal during actual oxidizing refining.

(*b*) To remove silicon to reduce its load in steel making to economize on flux consumption and for reducing the slag volume to reduce the iron losses during steel making.

(*c*) To remove phosphorus, under more idealized conditions, to finally meet its lower specifications and in this process to produce much lower slag volume and to achieve lower iron losses in slag.

(*d*) To ultimately aim at a *slag-less* or *lime-less* steel making operation which is bound to decrease slopping, flux consumption, and iron oxide loss and to improve productivity.

(*e*) To ideally aim at only decarburization during steel making to obtain 100% end point chemistry and temperature at once thereby improving productivity.

This has given rise to the development of the following independent additional refining operations, in the overall ore to steel processing route, in the form of external treatments of hot metal as:

(*a*) External desiliconization

(*b*) External desulfurization

(*c*) External dephosphorization

And secondary steel making processes including vacuum treatment, etc., for further subdividing the overall process of steel making to finally produce the desired quality of steel. These three treatments form the subject matter of this chapter; the secondary treatments will be described later under secondary steel making.

13.4. External Desiliconization

The problem of high silicon in hot metal has already been dealt with earlier while describing the *duplexing* and the flush slag practices in open-hearth. An increasing proportion of silicon requires an increasing proportion of flux to maintain the required basicity of slag during steel making. This also requires extra thermal energy for melting of the flux to become a part of slag. This is a substantial economic factor when the electric furnace is employed for steel making, wherein the power consumption goes up. This increases the slag volume with its attendant disadvantages, in particular the increased loss of iron as oxide in slag, for maintaining the required oxidizing power. Silicon can be readily removed in a separate vessel at the expense of much less flux and iron oxide loss in slag. Oxygen gas or mill scale can be used for such oxidizing refining. This process can be carried out in the following alternative ways:

(*a*) In a *blast furnace runner* while the hot metal flows from the tap hole to the ladle.

(*b*) In the traditional open top ladle designed for this specific purpose.

(*c*) In the torpedo ladle.

Silicon can be oxidized in the runner while the hot metal flows from the tap hole to the ladle either by introducing oxygen through consumable pipes or the addition of mill scale.

This process is usually carried out in the hot metal transfer ladle (not torpedo) itself. It does not take more than –20 minutes of actual operation. Refining can be carried out by using even consumable oxygen lance. The carry-over slag from B.F. gets neutralized in this operation and whatever slag still gets carried over into the steel making, after the desiliconization, has a less adverse effect on subsequent steel making. Silicon oxidation raises the temperature of the metal by nearly 50-100°C, depending upon the amount of silicon oxidized. This is helpful in the subsequent main steel making operation. The low silica slag, made later in the primary refining, has lower corroding influence on the basic lining and therefore improves lining life of the main steel making furnace. It also helps to form highly basic slag early in the refining, and thereby improves refining time. The total overall iron loss in slag produced during desiliconization and in steel making is reduced. It results in lower consumption of ferro-alloys and at the same time better recovery is achieved. It makes the subsequent oxygen refining process more amenable to automatic on-line control and arriving at the correct turn-down conditions.

Despite all these advantages, except its earlier use, in conjunction with open-hearth formerly, it has not been widely adopted because of the excessive attack of the less basic slag, generated during the desiliconization, on the lining and also because of its general handling problems. Hence every effort is made to produce as low a silicon containing hot metal as possible in the blast furnace itself without bothering about its sulfur content. This is the acid-burdening practice as described earlier in Chapter 5, which is the common mode of BF operation, now universally adopted to obtain hot metal with silicon below 0.5%. This is obtained by keeping basicity close to 1.0 in the BF operation. It does increase the sulfur content of the metal, which is dealt with by external desulfurization.

13.5. External Desulfurization

The basic philosophical approach to adopt external desulfurization was already discussed in Chapter 5 while discussing the raw materials for steel making. It has now become a common practice to adopt external desulfurization. It means the blast furnace will be run under more or less acid burdening conditions, i.e., the basicity will be maintained close to one and not very high, when desulfurization inside the blast furnace will not be effective. This practice decreases the silicon content of the hot metal to its minimum, which is often around 0.5% and which is generally considered quite ideal. The hot metal however contains extra sulfur, depending upon the total sulfur load in the charge. This hot metal is subjected to external desulfurization, before charging in the steel making furnace.

Sulfur removal from steel is basically a slag-metal interfacial reaction. The slag part can be either solid or liquid. The basic thermodynamics and kinetics of desulfurization reaction were already discussed earlier in Chapter 4, Sections 4.2.5 and 4.3.1. The basic requirement of lowest possible oxygen potential for effective removal of sulfur is well known. That is the reason why every attempt is made to avoid sulfur removal under the oxidizing conditions of steel making. The desulfurizing efficiency is very high in external treatment. Soda ash is very effective since its sulfide capacity is very high. But it creates other hazardous pollution problems and hence is discouraged.

The next best reagent is Mg metal granules and CaC_2 or an admixture of the two in some proportion. It produces a very dry slag that does pose problems in its raking and separation. The addition of limestone is to generate gas, by way of its calcinations, from within for effective stirring and dispersion, which is essential for hastening the reaction. The addition of carbon is to neutralize the oxidizing action of the evolved CO_2 into a net reducing atmosphere, so essential for desulfurization. The chemical composition of the reagents used in shown in Table 13.1.

The actual amount so injected depends upon the amount of sulfur to be removed, particle size of the reagent and temperature of the hot metal. It is usually in the range of 4-5 kg/t and fed at the rate of 40-45 kg/min.

Table 13.1
Chemical Composition of Reagents Used [Tata Search 1996, p43]

Compound/element, %	CAD-80	CAM-20
Calcium carbide	58-60	75
Magnesium	—	20
Calcium carbonate	17	—
Calcium oxide	20	—
Carbon	3-5	5

This reagent is injected deep inside the hot metal pool via a neutral carrier gas. There is no effective alternative system of bringing the reagent intimately in contact with the melt. The area of contact of the reagent/melt depends on the size and the specific amount of reagent added. This brings the sulfur level to 0.010-0.025% S.

The neutral carrier gas generates a plume wherein the gas bubbles are present. They lift up the reaction product as they are pumped into the plume and are separated. It has been estimated that the desulfurization reaction is first order reaction and the rate constant is nearly proportional to the square root of the gas flow rate. Hence maximum amount of gas, as is otherwise permissible, needs to be blown. But increased gas flow leads to higher temperature drop and only a certain drop is permissible.

13.6. External Dephosphorization

Nearly 85-90% of the P in the charge in the blast furnace joins the metal phase. If this is high and if low P steels are to be produced ultimately, phosphorus is also removed externally like silicon or the sulfur. The conditions of P-removal have already been discussed earlier in Section 4.2.2. These conditions have to be met for effective dephosphorization of steel externally. The reagent that gives the best possible dephosphorizing capacity is to be used to achieve this. The dephosphorizing ability of lime is well known. Barium-based compounds have been found to be more effective.

The external dephosphorization has not found much application because not many steel specifications ask for so low a phosphorus content. Occasional low P specifications are somehow met by manipulating slag regimes in the primary steel making process itself or by using multiple slags, forgoing a bit of productivity. This is yet considered adequate by most steel makers.

13.7. Concurrent Removal of Si, S, and P

Normally these are not removed one after the other from the same hot metal bath. This is fraught with excessive loss in metal temperature, abnormal generation of slag, i.e., reaction product and its handling problems and often on account of its unfavorable techno-economics. For similar reasons they are not removed concurrently.

13.8. Basics of Modern Steel Making

Depending upon the varieties, volumes of various products and their final specifications, on one hand, and the quality of iron ore and coke available or likely to be used on the other, the total integrated iron and steel making process design is worked out. For ordinary cheaper constructional grades of steels, therefore, the process design includes only one-stage ironmaking and only one stage steel making process to produce the final required product. A typical example of this is the presently adopted coal-based DRI production, coupled with the induction furnace, for its melting, and traditional ingot mold casting and rolling, to produce the desired cheaper commercial grades of steels.

As the final product requirements become more and more stringent, in terms of the variety, quality and volume, this simple process route gets more complicated, with additional steps in iron and steel production to achieve the end requirements at better economy and productivity.

The ironmaking then includes, in addition to the blast furnace, the additional treatments as mentioned above. Similarly the steel making gets divided into primary steel making and one or more of secondary steel making processes. These additional treatments have been developed over the years, out of practical experience, supported by theory and R & D inputs. They have proved to be more economical, in spite of being more complicated and elaborate.

The laboratory research and pilot scale R & D efforts are matched in due course of time, by the development of required mechanical electrical equipment designs, for their adoption in practice. Once this phase is over their computer control and automation comes in. This is how the steel making has progressed and is under progress presently. It should be remembered that all these efforts need lot of initial investments before the process gets modified. These are very capital intensive activities.

The modern steel making operations or current steel making practices that are described in the chapters to follow shall reveal this approach in action.

13.8.1. Overall Quality/Economy/Productivity

Steel making is nothing but one of the enormous economic activities. It is carried out to finally make profit. The process is therefore carried out to achieve maximum productivity of quality products by the most economical route. A given set of equipment and a given processing route can give a certain maximum productivity, while maintaining the required quality standard of the products.

Modern steel making processes are very fast. The efficiency of steel making lies in carrying out the refining in such a way that the end conditions in refining are achieved in one go and without any trial and error approach in individual heats. In order to achieve this, elaborate automatic process-control has been evolved and is used extensively in practice. Any lack of such control on process route is bound to decrease the efficiency.

From the same equipment, if the process route is improved higher productivity may be achieved. Similarly using the same process route modifications in equipment or even replacement of equipment, partially or fully, can improve the productivity without impairing the quality. The modern steel making therefore takes into account the intricate interrelationships of capabilities of the equipment, efficiency of the process route adopted and the requirements of quality standards of the products and, from them only, evolve the best possible steel making practices. These relationships can be changed for better economy through required R & D efforts only. At times productivity is increased, for earning more profit, without necessary R & D input. This invariably ends up with quality as its first victim.

CHAPTER 14

MODERN ELECTRIC ARC FURNACE PROCESS

In This Chapter

- Introduction
- Trends in Modern EAF Practice
- Design Improvements
- Process Modifications
- Charge Modifications
- Modern Arc Furnace Practice
- Stainless Steel Making
- Electric Arc Furnaces in USA (2017)
- Concluding Remarks

14.1. Introduction

Until recently, factors like the small size of the furnaces and the high cost of electric power restricted the use of electric arc furnaces for the production of only costly alloy steels. The cost of electric power vis-à-vis chemical fuels steadily moved in favor of the former and the applicability of arc furnace improved accordingly. The relatively cleaner working conditions of electric furnace process contributed to this success. The demise of open-hearth, on account of its cumbersome design and construction, energy inefficient operation, and slow production rate eventually led to increasing adoption of arc furnace, as perhaps the only alternative steel making process to be adopted for utilizing solid steel scrap as charge on large scale. The sixth and the seventh decade of the previous century saw the cost pattern of electric and chemical energy to move in favor of the latter. Simultaneously the electrical engineering aspects of design and construction of electric furnaces steadily led to bigger and bigger furnaces being adopted in practice. Furnaces of 70 t capacity were in use by 1950. It rose to 150-200 t capacity by 1960. The largest furnace of 800 t capacity was built and it was powered by 200 MVA transformer and was rated to produce over a million tons of steel per year. The operation was also modified to rapidly charge, melt and refine steel such that the process could compete with other processes of bulk steel production under certain conditions.

The change in steel making technologies over the years has been so drastic that now broadly only two processes of steel making are essentially left on the horizon, i.e., the electric furnace process and the BOF processes including their general modifications. The electric furnace is no longer restricted to the use of cold scrap charge alone. It can take a large proportion of DRI and hot metal in its charge, and can thereby compete with any other process even for bulk steel production. The conventional process routes of BF/BOF and the DRI/EAF have now been combined in the present electric process to take up DRI and HM in the charge to produce any variety of steel with equal case. For this reason, very large capacity DRI plants have been put into commercial practice as a part of integrated steel plants.

14.2. Trends in Modern EAF Practice

EAF process has been developed in multiple directions in the last few decades to improve upon:

1. Process flexibility in terms of charge acceptance.
2. Larger capacities for taking advantage of production economy.
3. Higher transformer capacities to increase rate of power input for improving productivity.
4. Better quality product, better cleanliness, etc.
5. Improved furnace design to decrease tap-to-tap times.
6. Judicious use of chemical energy to supplement the electric power inputs.

7. Improved process designs to improve process efficiency.
8. Maximum automation to improve consistency of operation.
9. Maximum use of process heat for steel making.
10. Minimum environmental damage.

The recent trends in arc furnace practice can therefore be summarized under the following sub-heads:

14.2.1. Design Improvements

(*i*) Rapid melting technology
(*ii*) Water-cooled panels
(*iii*) Eccentric shell with bottom tapping
(*iv*) Emission and noise control
(*v*) Switch gear improvements
(*vi*) Automatic alloy feeder
(*vii*) Coated and water-cooled electrodes
(*viii*) Oxygen lance and its manipulator
(*ix*) Effective off-gas treatment/recovery of fuel value
(*x*) On-line control through exit-gas analysis
(*xi*) Use of jet-box or co-jet for enhancing melting
(*xii*) Top-lance oxygen blowing besides the side jets
(*xiii*) Direct-arc furnaces

14.2.2. Process Modification

(*i*) Foamy slag practice
(*ii*) Scrap preheating
(*iii*) Hot tundish
(*iv*) Coupling with ladle furnace
(*v*) Stirring by purging gas
(*vi*) Process automation

14.2.3. Charge Modifications

(*i*) Use of DRI/sponge iron/HRI to substitute scrap

(*ii*) Use of hot metal as regular charge material

14.2.4. Improved Management Philosophy

Virtually aligned continuous process with ultra-high efficiency to produce quality steels with effective quality control at every stage is the hallmark of this philosophy.

It has, on the whole, led to develop and adoption of the most up-to-date technology for very cost effective steel making. The emphasis is on achieving the highest productivity from a given setup using given raw materials. This needs continuous assessment of equipment, raw materials, process technology in the light of cost of production vis-à-vis market demands and price-recovery of the products. It is a continuous non-stop process of evaluation and improvements accordingly. It needs a courageous management outlook and philosophy. Through all these improvements it has been possible to produce more than 3.5 million tons from four 200 t furnaces with hot metal, scrap, and DRI charge.

14.3. Design Improvements

14.3.1. Rapid Melting Techniques

The rate of melting of solid charge in a way determines the total heat time, and hence the productivity of the furnace. It is directly proportional to the power input in terms of kVA/t capacity of the furnace. It has now been agreed by the Committee on Technology of the International Iron and Steel Institute that arc furnaces would be classified in terms of their transformer ratings as:

(*a*) Low power (LP)	100-200 KVA/t capacity
(*b*) Medium power (MP)	200-400 KVA/t capacity
(*c*) High power (HP)	400-700 KVA/t capacity
(*d*) Ultra high power (UHP)	more than 700 KVA/t capacity

Earlier the low and the medium together used to be called regular power (RP). It is obvious that if regular power is converted to high or ultra high power by changing the transformer for the same furnace, the rate of power input would increase and thereby the melting rate and reduction in heat times. This however needs additional changes in the design of the furnace as a prerequisite. The electrical configuration was earlier dictated by the refractory lining wear, but this limitation no longer exists because of the use of water-cooled lining-less panels. It is therefore possible to operate the furnace with higher voltages and lesser current in secondary, and with high reactances in secondary circuits gives more stable arcing conditions. The reduction in electrode currents improves electrode life. Melting of sponge, which has lower thermal and electrical conductivities as compared to steel scrap, is easier in the modified secondary circuit designs. Feeding of high or ultra high power needs water-cooled cables, water-cooled holders of electrodes, special electrodes, and so on to become effective. The net time saving in using higher power is in proportion to the increase in power per metric ton capacity. Some typical power ratings for different capacity furnaces are shown in Table 14.1 to illustrate the variations in ratings under different power systems.

It has been estimated that the use of UHP in place of regular power supply results in saving in melting and refining times of the order of 30% and 10% respectively. The significance of UHP has gradually evolved into the concept in terms of more efficient materials handling and increased furnace availability, with power on, and which should reach 65-70%.

Performance data of four different UHP furnaces are summarized in the following Table.14.2.

The electric arc furnace is a very efficient melting machine during the melt-down period. The availability of the furnace can be further improved if it is essentially used for melting, and the subsequent refining is carried out in another furnace powered with a relatively lower capacity transformer in the form of a ladle furnace. Such an arrangement needs at least two electric arc furnaces, one big with perhaps UHP supply and the other with a small transformer, operated under controlled synchronized conditions.

Table 14.1 [17]

Furnace capacity	*Regular power (HP) KVA*	*High power (HP) KVA*	*Ultra high power (UHP) KVA*
5	3000	5000	6500
10	5000	7500	10,000
50	17,500	25,000	30,000

Table 14.2 [17]

Item	*Units*	*1*	*2*	*3*	*4*
Shell diameter	mm	3350	4100	5200	6700
Tap weight	t	22-20	27	57	140-145
Transformer rating	MW	13	20	34	80
Max. melt-down Power	MW	9.5	13	25-26	48-53
Approx. power factor	-	0.70-0.75	-	-	0.67-0.69
Melt-down currents	KA	3 × 26…27	3 × 30…42	45-50	-
Electrode dia	mm	350	400	500	6.00
Electrode length	mm	1830	1830	-	-
Energy consumption	KWH/t	510-530	-	49 0/t HM	510-540
Electrode consumption	Kg/t	5	-	5.5	4-4.75
Tap-to-tap time	hr-min	-	1.30 to 1.55	2.15	2-20 to 2-30

Many do not prefer this and adopt two same sized furnaces to ensure independence of operation and control.

The use of coal injection along with oxygen and the use of foamy slags are two important modifications that have been adopted even on low and medium powered small capacity furnaces very recently to improve upon the electric energy efficiency of arc furnace operation. Coal is injected even during melting, immediately after a small pool of molten metal is formed, to hasten melting. The thermal efficiency is further improved by allowing a foamy slag to form around the electrodes. This acts as an insulating blanket around the arc, and the arc heat is thus transferred to the metal for melting more efficiently. It also helps to expedite the slag metal reactions during the oxidizing refining, as there is no necessity to wait until melt-out (all charge is molten) to begin oxidizing refining.

14.3.2. Water-cooled Panels. In view of a favorable relation of diameter and height of the steel bath and due to construction difficulties, the furnace diameter cannot be increased at liberty. The refractory consumption, as a result, exceeds the acceptable values for large capacity furnaces particularly if high or ultra high powered. The use of water-cooling systems for the walls and the roof becomes absolutely necessary for economical operation of these furnaces.

Water-cooled panels have been developed as a substitute for refractory lining for upper side walls. These panels are made up in segments and arranged around the upper side walls as a buffer against the high thermal input of the furnace. These panels are designed with some form of studs or cups on their hot face so that they may retain a slag coating. This slag cover is deliberately given to reduce thermal conductivity through this zone.

Water-cooled panels are suggested even for the LP and MP furnaces in operation as a part of modernization of their designs to improve their efficiency. They can be fitted either from the interior or exterior of the shell.

As a result, the side wall of EAF now is a water-cooled cage enclosing the water-cooled panels and nothing else. This allows replacement of an individual panel when damaged with a minimum of down-time. The water-cooled cage does not allow any thermal expansion, thereby avoiding thermal stresses and gaps for air leakage. It allows extra power input and thereby expedites melting, using UHP and the like.

The EAF is normally operated under slightly negative pressure and air, therefore always gets sucked in through the door or the spout. It will also get in through the gaps in wall panels if formed. Hence the panels are laid very tightly without gaps, as far as possible.

14.3.3. Eccentric Shell with Bottom Tapping. The bottom of the modern EAF is made eccentric or oval shaped opposite the door side, and the tap hole is located in this bottom area. The eccentric part is suitably covered by an adjunct to the cylindrical shell, which also becomes eccentric in place of cylindrical on one side only. A small hole is located in the eccentric shell cover, vertically above the tap hole, such that granular refractory material could be dropped to fill the tapping hole to close it. The tapping hole in the bottom eccentric part is also closed from outside the bottom steel plate by a refractory sliding gate, similar in design as is used for teeming ladles. For tapping, the sliding gate is pulled apart and metal flows out without much difficulty by removing the refractory grog earlier put in to close the hole. The eccentric shape of the modern arc furnace along with the water-cooled panels are shown in Figure 14.1.

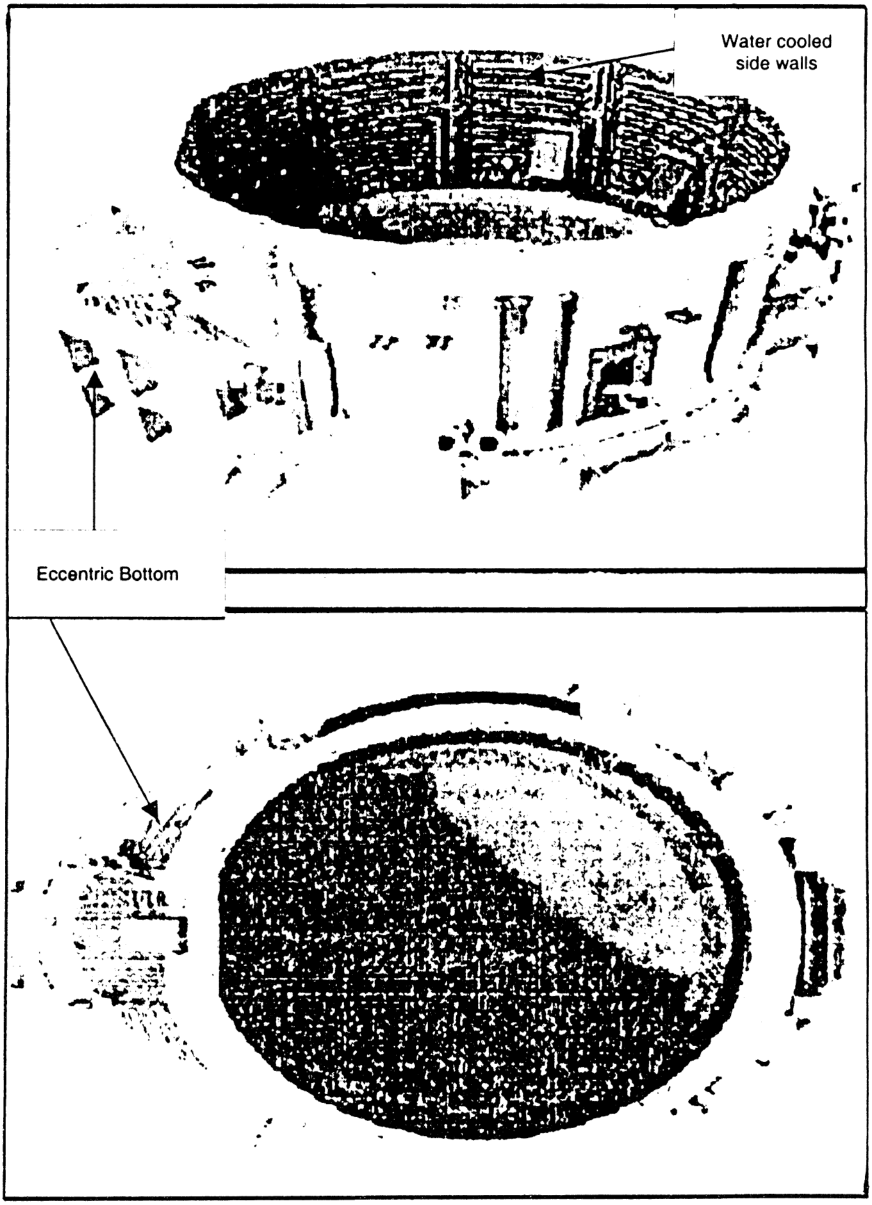


FIGURE 14.1. Eccentric EAF showing the design and the water-cooled panels.[18]

The chief reason for designing this eccentric tap hole is to ensure nearly slag-free metal tapping in the teeming ladle or the ladle furnace. It also results in less loss of temperature of the bath during tapping. This facilitates the retention of liquid heal for the next heat. This design leaves the crane free for other activities. Otherwise it is required to move the ladle during tapping. The other benefits include faster tapping, resulting in saving of a few minutes of heat time and more compact stream of metal flow, which is less susceptible to reoxidation and nitrogen pick-up. A possibility for shrouding the stream exists, to minimize it still further.

The eccentric bottom need not be tilted that much as is needed for spout type furnace for tapping, and thereby allows the side-wall water panels to be installed to still lower levels.

Usually granular material mainly MgO containing nearly 2-3 % FeO is used to fill the tap hole from inside. This material is not compacted in any way, but molten bath still covers it from inside. The looseness is helpful in getting it open easily when the bottom cover is pulled aside for tapping.

14.3.4. Emission and Noise Control. The anti-pollution legislation and necessity for better working conditions, particularly in the case of modern large arc furnaces, makes it almost imperative to have an effective exhaust system on the furnace for emission control. Popular on large furnaces is direct evacuation or a fourth hole, as it is termed, where a duct tap is made directly into the roof and slight interior negative pressure is present inside the furnace to evacuate smoke and fumes. This coupled with fugitive emission control at the spout and door using curtains or canopy in the roof truss area above the furnace makes up a complete evacuation system for the melt shop.

A relatively new entry is the complete enclosure or dog-house. This not only accommodates the effluent but gives some relief to the noise problem as well. In the early days the size of off-take in cubic feet per minute was nearly three to four times the transformer rating in KVA. It is now taken as approximately seven times and increases to nine to twelve times if fugitive emission control is required.

During the melt-down the ambient noise impact around large UHP furnaces may reach levels of 95-110 decibels. As of necessity noise attenuation using protective headgear or ear plugs seems to be the standard equipment. The enclosed control cabin can provide at least a 20 decibel reduction in sound. The current furnace designs have been able to bring the noise level down to a low level of only 80 decibels.

14.3.5. Switch Gear Improvements. An electric arc furnace is switched on and off more frequently than any electrical appliance of its type and size. This repetitive switching of up to 150 times a day or 45,000 switchings per year is a sizable task for switchgear. Early furnaces were operated by oil-circuit-breaker (OCB) and the tar had to be poured out of the switch tank each week. Then came the air-magnetic and air-blast gear, which worked well but was not mechanically sound.

Now the Joslyn vacuum interrupter is a standard piece of equipment for this job. The basic switch is a 15 KV, 600 amp assembly that can be arranged in series-multiple combinations for desired electrical load.

14.3.6. Automatic Alloy Feeder. With increasing size of the arc furnace, the net additions have increased in proportion. For meeting stringent specifications, the additions have to be made more accurately. For these reasons automatic weighing of additions as per the auto-control directions are made and are added via automatic alloy feeders. The actual mechanism of such varieties of additions is a complex arrangement of conveyor systems leading to the feeder via weighing machines. This not only saves time but also makes the overall process more accurate to meet the specifications more precisely.

14.3.7. Coated and Water-cooled Electrode. Electrode consumption cost currently may be around 10-15% of the conversion cost in most arc furnace plants, and therefore the electrode performance is of immense importance. The electrode is consumed in three ways. The tip consumption amounts to nearly 50% of the total electrode consumption. The surface consumption is around 40% and the remainder 10% may be due the mechanical breakage. The tip consumption is directly dependent on electrode current, hence modern practice is to use low current high voltage secondary. The tip is also consumed by dissolution in liquid steel if it is to be often lowered down to strike arc and such operation. The surface consumption is reduced by coating the electrode. The mechanical damages can be reduced by using an electrode automatic regulator. Use of sponge iron definitely improves the performance of electrode tip and the surface. The increasing use of sponge iron in the charge may lead to develop electrodes more suitable for this rather than the conventional one designed for steel scrap charge.

The UHP furnaces carry heavy current through electrodes and hence need water-cooled electrode holders. Similarly for high sponge charge in large capacity furnaces, hollow electrodes have been recommended.

14.3.8. Oxygen Lance. Use of the oxygen lance for refining in arc furnaces is now a routine practice even for small furnaces. Use of the consumable lance has been quite popular on old furnaces. But modern large capacity furnaces are provided with the water-cooled lance as in LD process. These are inserted through the roof from the top by providing an additional hole besides the three holes for electrodes. The consumable lance is inserted through the slagging door or an additional charging door if provided. Those using water-cooled lances have to be provided with hoods to carry fumes, and these may be provided with induced draught and fume cleaning systems. This has enabled the arc furnace to make even bulk steels on competitive basis. This of course needs additional innovative adaptations as well for effective economy.

14.3.9 Effective Off-Gas Treatment/Recovery of Fuel Value. The modern pollution control laws demand that the exhaust gases from EAF are treated and then only let off into the open atmosphere. The gases leave the furnace at the furnace working temperature and hence they must be cooled to almost room temperature. They also contain very fine iron oxide, formed as a result of vaporization of iron in the arc space, and which must also be recovered before the gases are let off into the atmosphere. Generally 0.8-1.5% of the iron charge is evaporated during the heat in an EAF.

The EAF is therefore provided with an elaborate gas treatment plant as shown in Figure 14.2. At the furnace working floor the off-gas line is called the elbow because of its shape. The CO contained in the gas must be either burned fully in the elbow or is utilized for its thermal recovery by burning it elsewhere usefully. The dust collected from the gas is recirculated in the sinter plant.

14.3.10 Jet-Box /Co-Jet. Initially jet-box designs were developed to burn fossil fuels inside the furnace during melting of the scrap to improve melting rate. These were later improved into Co-Jet designs that work at much higher pressure such that the injector nozzle delivers a three to seven-foot laser-like jet of oxygen at supersonic speed onto the charge. It results in very fast melting of scrap. The characteristic of conventional supersonic jet vis-à-vis the coherent jet can be seen in Figure 14.3. Mathur [19] sums up the advantages of Co-jet as follows:

1. A coherent jet maintains its original velocity, diameter and gas concentration for distances in excess of 70 nozzle diameters, significantly longer than a conventional supersonic jet.

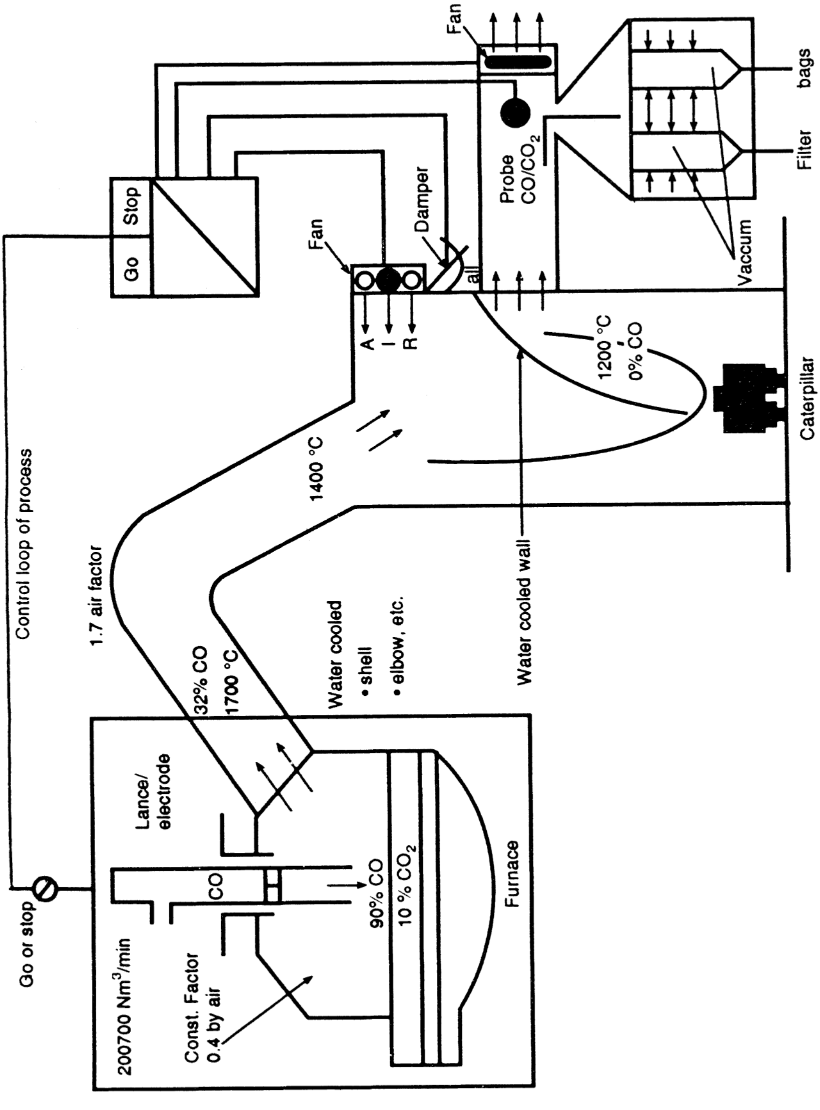


FIGURE 14.2. Shows the schematic of the off-gas treatment plant associated with EAF.

2. A coherent jet entrains less than 10% of the mass of ambient gas entrained by a conventional supersonic jet.
3. The rate of jet spreading and the jet decay is much lower for a coherent jet.
4. The jet force number (or impact pressure) is significantly higher for a coherent jet at any distance from the nozzle.

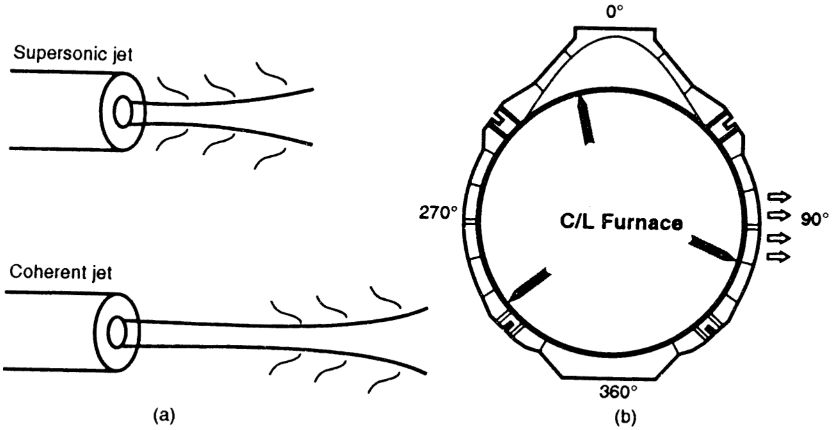


FIGURE 14.3. Shows in (a) the penetrating nature of the co-jet vis-à-vis the usual supersonic jet and in (b) the arrangement of how they are mounted in the shell [19].

5. There are staged firing profiles provided such that it can be operated in any desired mode from piercing oxygen jet on one end to a mere spray of oxygen at the other end. This gives a bushy flame for preheating the scrap to a penetrating flame for almost instant melting and any combination in between.

The key feature of this technology is that a fixed injector is provided in the furnace sidewall, which can carry out effective bath decarburization, although the tip of this co-jet is far above the still bath surface. The decarburization rate of 0.03% /min is achieved by blowing 1200-1400 cu m/hr of oxygen at NTP. It is estimated that almost all the oxygen injected penetrates the bath and reacts with it. It also helps foaming formation. It equally helps post combustion of CO inside the furnace.

These co-jets are mounted in fixed position in the side wall of the EAF. Normally three such jets are provided. They are located well above the normal bath/slag level and hence do not get damaged during operation. There are various rings of holes of fine diameter in the nozzle, to give the projected performance. They are operated individually as per the requirements.

14.3.11. Top Lance Oxygen Blowing. These days EAFs are provided with LD-like top oxygen blowing water-cooled lance to supply oxygen with supersonic speed for refining. This is to expedite the refining process in spite of the fact that the bath is quite flat in shape resembling open-hearth furnace. This is one of the major ways to compete with BOF steel making processes in producing usual variety of steel products. The design of such lances is quite similar to those used in BOF processes. There is a delta provided to hold the lance in vertical blowing position. This has its own disadvantages, but is widely adopted in spite of them.

14.3.12. Direct Arc Furnaces. Lately in place of alternating current electric arc furnaces, direct arc (DC) electric furnaces have been designed and put into commercial practice. The design includes an additional graphite electrode put in the hearth of the shell from below to act as one terminal, the second being the electrode inserted from top as usual. As compared to the AC-EAF the DC-EAF claims the following advantages:

1. Low electrode consumption
2. Low energy consumption
3. Low refractory consumption
4. Uniform and efficient heat distribution
5. Better metallurgical control
6. Working possible even with weak electrical input

However the only disadvantage is that the DC graphite electrode is relatively expensive and the cost of bottom electrode is an additional expenditure.

14.4. Process Modifications

14.4.1. Foamy Slag Practice. Foam has been defined as a medium consisting of gas bubbles dispersed in a liquid phase, e.g., soap bubbles in water. The stability of foam therefore depends on surface tension, density and viscosity of liquid phase involved. Foam dies out if the liquid film between the bubbles drains out into the bulk. The lower the viscosity faster will be the collapse of the foam and vice versa. Molten slags with high viscosity and high surface tension readily form foam whereas metals do not. Foam also dies down if the gas phase is withdrawn. In steel making the generation of CO during the refining can lead to foam formation if the slag is viscous enough.

Formation of foam was noticed in open-hearth during limestone calcination if it rose to slag-metal interface because of faulty practice. It was not beneficial because it hindered heat transfer from gas to metal in open-hearth.

This was avoided in open-hearth practice. However in an electric arc furnace the insulating nature of slag foam has been beneficially used for preventing radiations from arc to furnace walls by getting the foam to surround the arc area during the melting-refining period in the early part of the heat. This is commonly known as foamy slag practice, and it effectively decreases power consumption in arc furnaces. The same thing is used in submerged arc ladle furnace to generate a foamy slag.

Since viscosity of a slag depends on chemical composition of the slag, foam stability in a given situation will be affected by the nature of the slag. Foam is deliberately generated by blowing oxygen in the small liquid metal pool below the arc without waiting for the entire solid charge to melt. The CO generated thereby develops and maintains foam around the arc. The slag is kept viscous for this purpose. The foam stability is given by Fruehan and others as[14]

$$\text{F. S.} = K\mu/(\rho\sigma)$$

where ρ is the density of slag, σ is the surface tension and μ is the viscosity and, K is a constant depending upon the experimental configuration. A higher proportion of CaO or Al_2O_3 increases the viscosity and tends to stabilize the foam whereas addition of CaF_2 is bound to decrease the foam stability. A moderate rate of CO evolution tends to stabilize the foam. A slow argon purging in ladle furnace may also behave likewise. But faster CO evolution and faster purging is surely counterproductive.

The insulating nature of foam thus has been beneficially used in arc furnace operation by suitably adjusting the related parameters. In a ladle arc furnace, the situation can be similar but in argon bottom purging formation of foam can be problematic.

14.4.2. Scrap Pre-heating

It is an established fact that if a preheated solid charge is added to an arc furnace, it is going to reduce the time requirement for melting, thereby improving the economy. Attempts have been made to utilize off-gases during refining to pre-heat the subsequent charge, but they add to the problems in designing the flues and handling of the hot scrap. It is therefore common to preheat the scrap with the help of oxy-fuel burners inside the furnace as such. This is economical only if the cost of chemical fuel is favorable as compared to the cost of electricity. This has led to the development of the energy optimizing furnace as a modification of arc furnace design. This will be described later. The arc furnace practice has been developed to accept hot sponge directly as charge but this is not without various associated problems.

14.4.3. Coupling with Ladle Furnace

The power requirement during melting is much higher than during refining. Now ladle furnace is increasingly being adopted for better quality product. Coupling of these two have been suggested to effect production economy as well as capital economy. In this coupled arrangement the main UHP furnace is used only for melting and the steel melt is transferred to the ladle furnace for final and leisurely refining. The net total transformer rating in the coupled condition is much lower.

14.4.4. Stirring by Purging Gas

It is now a universal practice to adopt bottom purging in EAFs of reasonable sizes. No quality steel can be produced without stirring the slag and the steel bath in a furnace during refining, in general and in ladle furnace in particular. The detail of this practice is described later under secondary treatments of steel in Chapter 21 under the section of stirring treatments.

14.4.5. Process Automation

Like the BOF process automation advancements, the EAFs have also been developed with respect to their automatic control. Like BOF it also relies on the continuous off-gas analysis, which is interpreted in terms of the chemistry of the bath inside the furnace at any moment. The key components of this system are [20]:

Water-cooled probe.

Customer specific continuous gas analyzer.

Supervisory control and data acquisition computer.

It intermittently but continuously draws gas samples during the refining and analyzes it for CO, CO_2, H_2, and O_2, using generally infrared-based sensors for CO and CO_2, thermal conductivity for H_2 and electrochemical analysis for O_2. The whole control system is based on assessing the gas contents of the exhaust gases. The philosophy being *what is not measurable cannot be controlled.* It also measures [21] various rates of:

1. off-gas leaving the furnace
2. air-in leakage in the furnace

3. CO generation from decarburization reaction and combustion

4. water entrained in the gas phase

5. energy losses via the gas phase

Based on these measurements the dynamic mass/energy balance is established to know the furnace operation according to the energy input to the charge, via both electrical and chemical sources.

14.5. Charge Modifications

The electric furnace charge is scrap and there is no control on its nature. The scrap charge is increasingly being prepared to define its quality and thereby exert auto control on its use in arc furnace. Some of the details have already been described in Chapter 5.

Steel scrap is shredded in heavy disintegrators by taking advantage of the fact that below a certain temperature (transition temperature of ductile-brittle fracture), steel is brittle enough for easy disintegration. This cryogenic breaking is widely adopted on large capacity scrap processing. Separation of non-ferrous materials in this case is far more easily done.

Use of sponge iron is yet another way of obtaining a feed material of well defined properties for effective auto-control of steel making in arc furnaces.

14.5.1. Sponge/DRI as Charge Materials

Electric furnaces were developed in the beginning to convert primarily steel scrap into finished steel. They were soon found to be best suited for converting scrap to finished alloy steels more economically. Steel scrap is therefore still the dominant charge material in electric furnaces. Cast iron scrap or pigs may be included in the charge to the extent of 15-20% of the total charge weight on economical ground. But this addition means addition of impurities from the cast iron or pigs and will have to be removed during refining. Any excess of these shall prolong the refining period, hence the heat time. As far as possible, electric furnaces are run to produce the same chemistry as the charge so that a minimum of ferro-alloys would be required to meet the chemical specifications.

The paucity of steel scrap led to the development of alternative iron-bearing materials in the form of directly reduced iron, popularly known as DRI/sponge iron, as the charge material for electric furnace steel making. The electric furnace process since then developed on a different direction to compete the existing route of BF/BOF process for the production of steel on large scale. The furnace sizes therefore continuously increased over the years and the emphasis slowly shifted to production of all those steels made by any other process.

With the same enthusiasm, the process was developed to take even hot metal as the charge with equal ease with any proportion of solid charge and produce all variety of steels.

A sponge iron and electric furnace combination for producing steel is estimated to be 30-40% cheaper in capital cost than an equivalent blast furnace and oxygen steel making combination in the lower range of production capacities. Such a plant is shown in Figure 1.1(c) and in Figure 2.2; around 1.0-1.5 million t capacities are in operation successfully. The consistent quality of DRI as compared to the varying nature of steel scrap can be used for effective automation of electric furnaces readily, thereby producing more consistent quality of steels.

The electric furnace practice using DRI and hot metal as the charge is the current practice of the day and is extensively adopted all over the world. It has therefore been described in greater details in the form of state-of-the-art electric furnace practice in Chapter 19.

Here the practice of using only the solid charge of scrap and /or DRI has been described in the form of modern practice.

14.5.2. Characteristics of DRI

Sponge iron has the following characteristics that affect its metallurgical behavior inside the furnace:

1. It is a porous material and is susceptible to furnace oxidation readily unless due care is taken to minimize the same during melting.
2. The gangue constituents of the original ore from which the sponge is produced remain with the sponge as impurities. These will be separated only on melting, when these gangue oxides join the slag phase. The acidic/basic characteristics of these gangue oxides therefore dictate the flux quality and quantity requirements during steel making.
3. Sponge does not contain impurities like Cu, Sn, As, Ni, Cd, etc. It means that it can be suitably blended with scrap containing such tramp elements and yet produce steels with required tramp specifications.

4. Sponge iron does not contain impurities like Si, Mn, S, etc., as are invariably present in steel scrap. However it does contain 0.6-1.2% oxygen, and about 3 to 4% carbon is allowed to get in to balance the sponge such that during melting the two interact to reduce the oxide and produce iron. Melting and refining of sponge therefore go together such that the conventional refining time after melt-out is practically nil. It is therefore necessary to have consistent oxygen and carbon in the sponge to control the melting process effectively.
5. The phosphorus content of the sponge should be preferably below 0.05% and sulfur below 0.03%. These are readily achievable.
6. Sponge iron can form a part of the charge and can be charged as scrap up to 15-25% of the total charge.
7. Use of 100% sponge charge in HP and UHP furnaces is a viable proposition, but continuous charging facilities are a prerequisite to avoid oxidation of sponge during charging.
8. The volume of slag produced in an electric furnace depends on proportion of sponge in the charge. It increases with the increase in sponge proportion and also with the gangue percentage in the ore from which sponge is made.
9. In continuous sponge charging, melting and refining should match in such a way that there is no need to carry out refining separately after the sponge is melted. This can be achieved by adjusting the feed rate in relation to power input into the furnace. The recommended rate is 27-30 kg /min/MVA power input and the material is fed at an angle of 35-50° to the horizontal bath surface.
10. Charging of hot sponge iron at around 650°C has been established to obtain better thermal efficiency.

When 70-80% or more of the charge is sponge iron, it is better to maintain hot heel of about 20-25% from the previous heat and the sponge iron is fed into this hot heel continuously at a uniform rate as shown in Figure 14.1. The standard contimelt process design for continuous charging and melting is quite popular. A hot heel of 20% of the charge is maintained to introduce the sponge below the bath surface to avoid oxidation of the charge.

14.5.3. Melting-Refining of Sponge Iron

The precise charging and melting practice of arc furnace depends on the proportion of sponge in the charge. This proportion is dictated by the

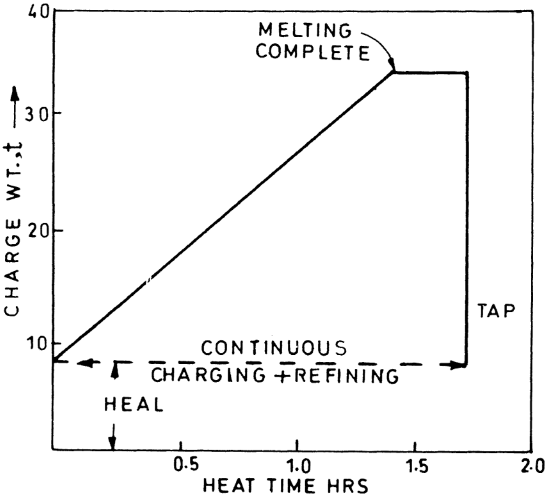


FIGURE 14.4. Arc furnace practice using high proportion of sponge in the charge and hot heal.

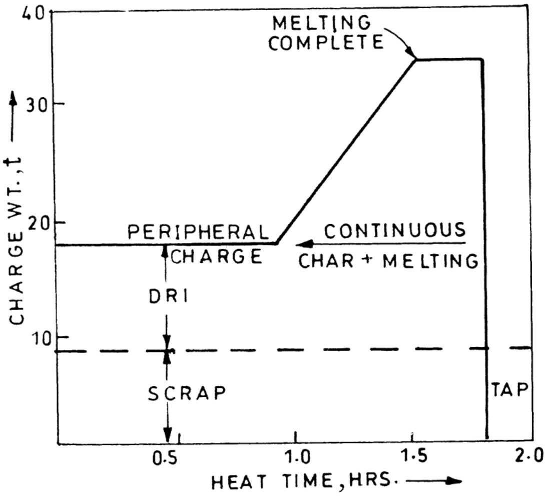


FIGURE 14.5. Arc furnace practice using scrap and DRI in the peripheral charge followed by continuous feed of DRI all along.

availability and cost of scrap vis-a-vis sponge. A small proportion, such as up to 30% sponge, can be dealt with as if it were scrap. It is charged in between two steel scrap charges and melted and refined as scrap is dealt with.

As the proportion of sponge in the charge increases the productivity goes on improving till around 30% sponge charge, that is heat time decreases, and then it again decreases with increase in sponge proportion. The productivity increase has been found to be 40-45% on the tap-to-tap basis. Not only are the heat times considerably reduced but they become more reproducible and predictable. This maxima in the productivity curve is due to the fact that after 30% sponge charge the amount of slag accumulated inside the furnace goes on increasing and it offsets the productivity increase. As compared to all scrap charge the productivity is always higher even up to 100% sponge in the charge.

For a given gangue, the productivity curve attains a maxima and then decreases and, the maxima is reached earlier the more is the gangue impurities in sponge iron.

With metallization more than 92% the heat time is not affected by the proportion of sponge in the charge but below it the heat times rapidly increases with fall in metallization.

The more is the gangue in sponge the more is the power required to produce a unit of steel with increasing proportion of sponge in the charge. The optimum results can be obtained by matching the feed rate with that of power input. The more homogeneous nature of sponge helps in getting a more stable arc.

The more gangue, the more will be the flux requirement and more will be the slag volume. It has been found that for every 45 kg rise in slag weight energy consumption goes up by 40 kWh/t of steel.

The continuous nature of sponge charging requires full power on for melting and refining to go on together and hence gives better power factor and better utilization of transformer capacity as compared to all scrap charge.

Electrode consumption is found to be decreased by about 4-5% when a high proportion of sponge is used.

Refractory consumption has been found to be decreased while using high proportion of sponge to the extent of even 40% in some critical areas.

14.5.4. Prospects of DRI-EAF Combination

A sponge iron and electric arc furnace combination for producing steel is estimated to be 30-40% cheaper than an equivalent blast furnace and oxygen steel making combination in the lower range of steel production capacity. Such a plant as shown in Figure 1.1(*c*) of 1.5 million metric tons capacity has been in operation. Several such plants have been or are being set up in India recently and they contribute a substantial fraction in the overall steel production. The degree of consistency in terms of metallurgical properties of sponge iron is far better than that of scrap, hence, extensive automation of arc

<table>
<tr><th colspan="2"><i>Period</i></th><th><i>Operation</i></th></tr>
<tr><td colspan="2">Repair</td><td>← Lime</td></tr>
<tr><td colspan="2">First charge</td><td>← Power on</td></tr>
<tr><td colspan="2">Melting</td><td>← Additional charge(s)
← ....
← Lime</td></tr>
<tr><td>Oxidising</td><td rowspan="3">Refining</td><td>← Lime at melt-down
← Oxygen or ore
← Spar</td></tr>
<tr><td>Slag off</td><td>← Fe-Mn, Fe-Si, lime and spar</td></tr>
<tr><td>Reducing</td><td>← Fe-Si, lime coke and spar</td></tr>
<tr><td colspan="2">Tapping</td><td>← Al to ladle</td></tr>
</table>

FIGURE 14.6. Scheme of making plain carbon steel in an arc furnace.

furnace operation has become readily possible with the use of sponge iron. This was rather difficult earlier because of the variable nature of steel scrap.

14.6. Modern Arc Furnace Practice

The existing arc furnace plants are being rejuvenated with respect to the above mentioned innovations to improve upon their economy to the extent possible in terms of capital investment required vis-à-vis economic advantages accruable.

A Greenfield arc furnace installation, however, will have to adopt the modern furnace design, sponge iron charge and modified practice to obtain desired economy of production even in the case of bulk steels.

A typical process of making low carbon steel from scrap and sponge iron charge is shown in Figure 14.6 (low sponge charge).

The average material and energy consumption for a mild steel heat are shown in Table 14.3.

Table 14.3

	Consumption per metric ton of liquid steel	*Note*
Scrap	1035-1100 kg	Depends on quality of scrap.
Fe-Mn(75% Mn)	6 kg	
Fe-Si (75% Si)	3 kg	
Al	0.5 kg	
Lime	40 kg	
Coke	2 kg	
Spar	12 kg	
Oxygen	8 m^2	
Dolomite	30 kg	Equivalent ore is 15 kg for fettling.
Roof bricks	3 kg	For roof life of 100-150 heats.
Wall bricks	4 kg	
Electrodes	5.5 kg	
Electricity	550-700 kWH	Depends on furnace size and quality of steel made.

Relative operating costs are as follows:
(Figures are approximate)

Scrap	65%
Other charge materials	5%
Electric Power	9%
Electrodes	4%
Mold, stool, teeming etc.	4%
Refractories	3%
Labor	7%
Misc.	3%
Total	100%

A typical log-sheet depicting the practice of making low alloy steel at the Durgapur Alloy Steel Plant is shown in Table 14.4.

Table 14.4
Log-sheet of Low-Alloy-Steel Heat [22]

Heat No. 5—5457	*SAE 8620*	*Specifications*		
		C 0.13/0.23 *(Aim 0.18/0.22)* *Mn 0.70/0.90*	*S 0.05 max.* *P 0.035 max.* *Cr 0.40/0.60*	*Mo 0.15/0.25* *Ni 0.40/0.70* *Si 0.20/0.35*
Time	*Operation*	*Remarks*		
23.45	Power on	Charge:		
		Turnings and borings	: 5000 kg	30,000 kg.
00.15	Recharge	Heavy scrap	: 10,000 "	
		Revert scrap	: 6000 "	
		Light/Medium scrap	: 9000 "	
02.25	All Melt:			
	C-0.67			
	Mn-0.45			
	S-0.022	Recharge:		

	P-0.027	Turnings & borings:	2000 kg						
	Cr-0.43	Light Medium scrap:	3000 "						
	Mo-0.10	Revert scrap:	15,000 "						
	Ni-0.39	Lime stone:	3000 "						
02.30	Oxygen blow 14 minutes. Total scrap charge = 50,000 kg.								
03.30	Slag off								
		Reducing Slag Making							20, 000 kg.
03.55	Alloy test:	C	Mn	Si	Cr	Mo	Ni		
		0.13	0.53	0.06	0.40	0.16	0.41		
		0.14	0.54	0.07	0.40	0.17	0.42		
04.30 to 04.40	Adjustment of ferro alloys	100 kg H.C. Fe-Mn, 172 kg Fe-Si lump 20 kg H.C. Fe-Cr. 200 kg Fluorspar.							
05.00	Tapped	Tapping temperature 1580°C							
Pit analysis:		C	Mn	Si	P	S	Cr	Ni	Cu
		0.23	0.81	0.025	0.010	0.30	0.47	0.17	0.42

14.7. Stainless Steel Making

Oxygen lancing of stainless steel bath for decarburization was first introduced in 1939. Originally it was meant to remove carbon below the required level without regard to oxidation of chromium. Oxygen lancing helps to raise the stainless steel bath from its usual temperature of 1600 to 1800°C at the expense of nearly 3% Cr loss to the slag which was earlier thrown out. The process was modified at a later date to incorporate a reducing period.

The slag was not removed until chromium was won back in the bath. The use of oxygen, thus revolutionized the stainless steel making practice in an arc furnace. Two typical log-sheets from the Durgapur Alloy Steel Plant Practices are shown in Tables 14.5 and 14.6.

In this practice oxygen blowing commences even before the heat is fully molten. The exothermic heat of refining is made use of in raising the temperature of the bath. Blowing continues till carbon comes below the required level. The rates of oxidation of Cr, Mn, and Fe increases with increase in blowing rate up to nearly 30 Nm^3/hr and thereafter it remains

Table 14.5
Log-sheet of a Stainless Steel Heat [22]

Heat No. 5—5090	*AISI 304*	*Specifications*						
		C 0.08 max *(Aim 0.07 max)* *Mn 1.0/1.40* *Si 0.50/0.80*	*S 0.030 max Ni 9.5/10.5* *P 0.045 max (Aim 9.5/10.5)* *Cr 18.0/19.0 (Aim 18.0/18.5)*					
Time	*Operation*	*Remarks*						
08.25	Power on	Charges:						
10.20	All Melt:	18/8 Scrap	: 2200 kg	Total scrap charge:				
	C 0.32 Mn 0.91	Revert scrap (Cr-Mo-Ni)	: 15500 “					
	S 0.030	Nickel	: 2200 “	22,850 kg				
	P 0.085	H.C. FeCr	: 1400 “					
	Cr 10.60	Lime Stone	: 1500 “					
	Ni 12.08							
10.30/11.05	Oxygen blow 37 minutes							
10.05/11.20	Coolant and reduction							
		18/8	: 6000 “	14,200				
		Fe Cr Si	: 2000 “					
		Fe-Si	: 300 “	37,050 kg				
		L.C. Fe Cr	: 5900 “					
11.50	Slag off							
12.15	Alloy tests:							
	C	Mn	Si	Cr	Nil			
	0.06	1.03	0.75	18.01	9.03			
	0.06	1.05	0.72	18.10	9.09			
12.50/13.10	Adjustment							
13.30	Tapped.	Tapping temperature 1540°C						
Pit analysis:	C	Mn	Si	P	S	Cr	Ni	Cu
	0.66	1.22	0.79	0.017	0.015	18.19	9.75	0.036

Table 14.6
Log-sheet of a Stainless Steel Heat 20 [22]

Heat No. *3-6100*	*AISI 304 L*	*Specifications*					
		C 0.03 Max *Mn 1.10/1.40* *Si 0.40/0.70*	*S 0.03 Max.* *P 0.04 max.*	*Mo 0.05 max* *Ni 10.0/11.0* *Cr 18.0/19.0*			
Time	*Operation*	*Remarks*					
15.10	Power on All melt:						
17.30		C-0.36	Borings and turnings	: 4000 kg			
		Mn-0.38	Revert scrap (Cr-Mo-Ni)	: 30,000 "			
		Cr-0.44					
		Mo-0.06	Nickel	: 5000 "			
		Ni-11.69	Lime stone	: 2000 "			
17.50/18.40	Oxygen blow	50 min					
18.45	Fe-si	220 kg					
18.50	Slag off						
18.55/19.40	Charge:	Fe-Si 700 kg. Mn 600 kg (Electro) ELC FeCr 1600 kg. Ni 750 kg					
20.40	Alloy tests:	C	Mn	Si	Cr	Mo	Ni
		0.025	1.33	0.85	17.55	0.07	9.63
		0.026	1.36	0.83	17.65	0.07	9.78
21.00/21.30	Adjustments		ELC Fe-Cr 1600 kg Nickel 750 kg				
21.50	Tapped Pit analysis:		Tapping temperature 1535°C.				
		C	Mn	Si	Cr	Mo	Ni
		0.025	1.27	0.80	9.00	0.07	10.50
		S	P	Cu	Al		
		0.014	0.030	0.025	0.009		

nearly constant. The higher the initial chromium content, the higher is the final temperature of the bath at the end of refining.

This method of raising the bath temperature is much more efficient than oxidizing the chromium by excess iron oxide as in the rustless process, because the iron oxide takes part of the exothermic heat of chromium oxidation for its own dissociation. The bath can be decarburized to 0.06% C and whatever chromium is oxidized, is reduced by Fi-Si and/or Cr-Si addition during the reducing period.

This is almost a universal practice that successfully takes any proportion of stainless steel scrap in the charge and for this reason the demand for this scrap exceeds its supply.

14.7. Electric Arc Furnaces in USA (2017)

Source: U.S. Geological Survey, Mineral Commodity Summaries, January 2017

Events, Trends, and Issues: U.S. iron ore production decreased in 2016 owing to decreases in steel produced from basic oxygen furnaces, which consume iron ore, and an overall decrease in U.S. steel production from 88.2 million tons in 2014 to 78.9 million tons in 2015 to an estimated 80 million tons in 2016. Continued increases in the share of steel produced from electric arc furnaces, which use steel scrap and DRI, offset steel production requiring iron ore. Despite modest increases in the price of seaborne iron ore during the year, major price declines in recent years have not been offset. In October 2016, the spot price for iron ore fines (62% iron content) imported into China (cost and freight into Tianjin port) was $58.02 per ton, an increase from $52.74 in October 2015, but still well below $80.09 and $132.57, prices seen in October 2014 and 2013, respectively.

Salient Statistics—United States:	2012	2013	2014	2015	2016[e]
Pig iron production[2]	30.1	30.3	29.4	25.4	23
Raw steel production	88.7	86.9	88.2	78.8	80
Basic oxygen furnaces, percent	40.9	39.4	37.4	37	33

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[e] Estimated

Continuously cast steel, percent	98.6	98.8	98.5	99	99
Shipments:					
Steel mill products	87.0	86.6	89.1	78.5	78
Steel casting [e,3]	0.4	0.4	0.4	0.4	0.4
Iron castings [e,3]	4.0	4.0	4.0	4.0	4.0
Imports of steel mill products	30.4	29.2	40.2	35.2	30
Exports of steel mill products	12.5	11.5	10.9	9.0	9.0
Apparent steel consumption	98	98	107	99	94
Producer price index for steel mill products (1982=100)[5]	208.0	195.0	200.2	177.1	168
Steel mill product stocks at service centers, yearend [6]	7.8	7.6	9.0	7.5	8.0
Total employment, average, number:					
Blast furnaces and steel mills[5]	92,600	90,900	91,000,	87,000,	87,000
Iron and steel foundries[5]	70,500	69,400	67,600	66,000	66,000
Net import reliance 7 as a percentage of apparent consumption	11	12	30	22	16

14.8. Concluding Remarks

The chief utility of an electric arc furnace was and still is in its ability to convert steel scrap into finished steel, and in this respect it is matched only by an open-hearth furnace. The two are, therefore, compared below:

Electric arc furnace process	*Open-hearth process*
1. It has now transformed itself into a process of making all types of steels rather than an alloy steel making process alone.	It can make plain carbon and low alloy steels readily but high alloy steels readily but high alloy steels can be made only in an acid open-hearth and that too from a well graded scrap.

2. Wide variety of charge materials is acceptable even for high alloy steel making.	It is acceptable but for the production of bulk steels only.
3. Metallized pellets can also be used as a raw material.	Metallized pellets cannot be used as it will get reoxidized again in the oxidizing atmosphere of the furnace.
4. The recovery of alloying elements of scrap is high.	Recovery of oxidizable alloying elements in scrap is poor.
5. Yield is nearly 91%.	It is nearly 89% for the same charge and product.
6. Furnace availability is 96%.	It is 91% maximum in the most mechanized shops.
7. Compact and efficient layout.	Clumsy and inefficient layout particularly in old shops.
8. Favorable cost pattern of electrical energy even in foreseeable future.	Less favorable cost pattern of fuel oil used in heating modern furnaces.
9. Wide range of furnace capacities and can match even open-hearth sizes.	Much higher as compared to usual electric arc furnace sizes. Not available in small sizes.
10. Flexibility in terms of intermittent operation, for irregular production, is good.	It is not so good.
11. Relatively clean working conditions.	Not so clean working conditions.

The electric arc furnace is, therefore, bound to be preferred to open-hearth for future installations for the production of even bulk steels from solid charge of practically any composition. Its superiority regarding high alloy steel making remains unchanged in spite of the fact that the BOF processes are rapidly bridging this gap. This is due to the fact that BOF processes need virgin hot metal whereas electric arc furnaces can run on scrap charges alone. Many of the old open-hearth shops have been dismantled and electric arc furnaces have been installed in those places. The installation of a very large number of scrap-based mini-steel plants in India is also a pointer indicating the use of the electric arc furnace process.

The electric arc furnace practice can be suitably modified to include secondary steel making technology, not only for increasing the productivity of the shop, but also for improving upon the quality of steels. These modifications will be described later.

CHAPTER 15

LD PROCESS——PLANT AND EQUIPMENT

In This Chapter

- LD Shop
- LD-Vessel
- The Oxygen Lance
- The Hood and the Waste Gas Treatment
- Vessel Lining and Wrecking Accessories
- Materials Handling and Storage Facilities
- Instrumentation

The refining of iron by oxygen lancing was first tried by Professor R. Durer in Switzerland, but it was at Linz and Donawitz in Austria that the pilot plant trials on 2-5 t vessels were successfully carried out to cast refined steel for the first time on June 25, 1949. A 10-15 t capacity vessel was estimated to be an economically viable unit, and after thorough testing of the product on a commercial level, the first LD plant was installed at Linz. The first commercial cast was made on November 27, 1952. Some time later, a second plant was installed at Donawitz. The first plant outside Austria was installed at Hamilton in Canada in 1954. LD process was first adopted in India by Hindustan Steel Ltd. at their Rourkela plant in 1956. It started production in 1960.

The name LD stands for Linz and Donawitz where the process was born. Coincidently it also stands for the nozzle process of Professor Durer in German.

From the initial 250,000 t annual production at Linz, the production of LD process reached nearly 16 million metric tons in 1962, nearly 200 million metric tons in 1970 and, nearly 400 million metric tons in 1980, and is very rapidly increasing its percentage share in total world steel production.

15.1. LD Shop

The layout of an LD shop is quite different from those of the conventional processes because of the specialties of this process. The rapid rate of production requires almost an automatic and very efficient system of material transport and weighing. A multi-floor shop is, therefore, required in spite of the increase in shop height. Tall shop is also required to raise and lower the lance in the vessel. An elaborate gas cleaning facilities are required to meet the pollution control legislation. The number of vessels in a shop may be generally two or three and one out of two or two out of three vessels operate at a time. The relining facilities required to maintain a uniform high production rate should always be more than adequate. A rapid process like this requires an efficient process control strategy using computers and automatic spectro-chemical analytical methods.

The general floor arrangement in the shop was already shown in Figure 7.4 as a plan view. The vertical elevation is shown in Figures. 15.1 and 15.2. A very simplified elevation view of the Rourkela Steel Plant is shown in Figure 15.3. The plant design very much depends upon whether an eccentric or concentric vessel is used. If an eccentric vessel is used the charging and tapping is carried out on the same side. It need not be so if a concentric vessel is used. In general hot metal is brought to the furnace by rail cars in a pit and the ladle is lifted by an overhead crane for pouring the metal in the vessel. The mixer, therefore, is located on the ground floor unlike in open-hearth shops. The scrap is stored in the scrap yard and is supplied in required lots in special charging boxes. The scrap, ferro-alloys, etc. are charged from the working floor. The flux, ore, etc., are fed from upper floors via a chute.

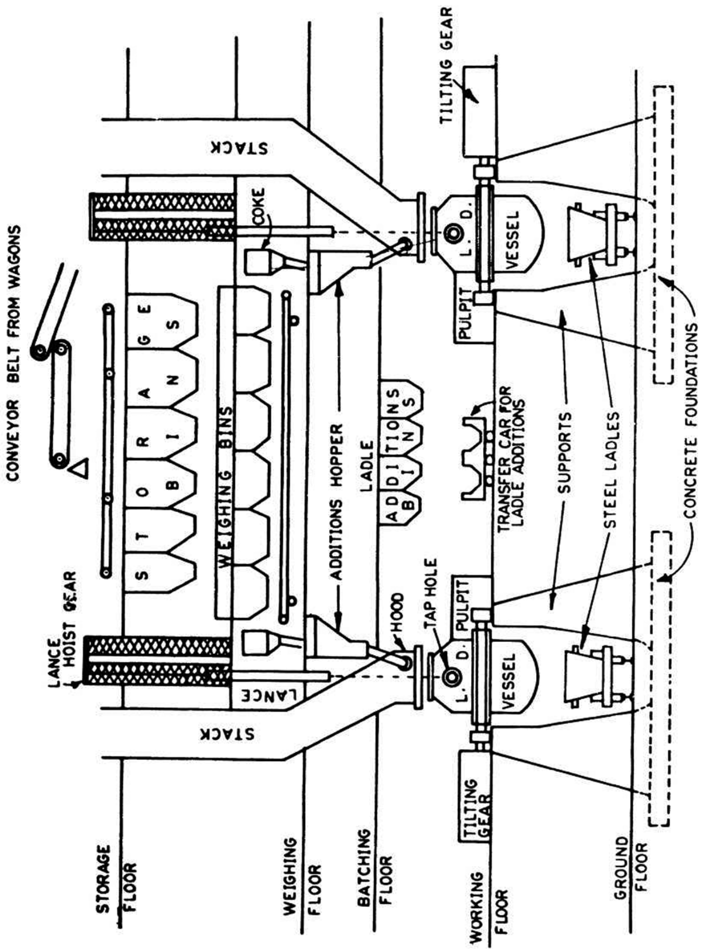


FIGURE 15.1. Vertical section through a LD steel making shop showing the relative locations of the main equipment.

The shop is provided with separate cranes for handling hot metal, refined steel, scrap and slag. In the Rourkela LD shop with 3 × 40 t and 2 × 60 t vessels the following cranes are provided:

Hot metal and steel cranes	…two 100/32 t
Scrap handling	…four 12/5 t
Slag pot handling	…two 50/12 t.

The size of this shop is 354 × 41 m for three converters and 135 × 41 m for two converters; it is nearly 47 m tall.

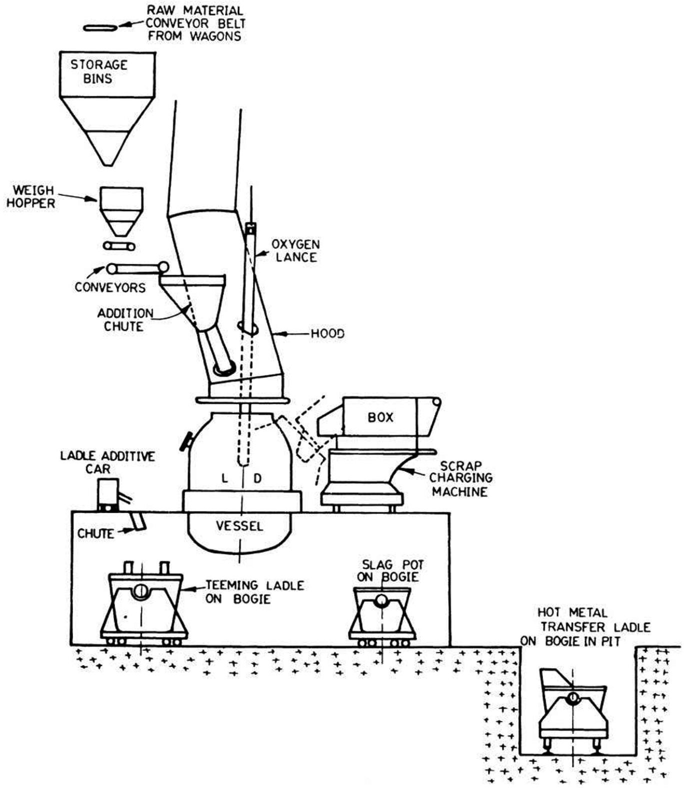


FIGURE 15.2. Vertical section through a LD steel making shop showing the relative locations of the main equipment. This section is perpendicular to that shown in Figure 15.1.

Being a very rapid process the LD shop is always provided with more than adequate facilities for material handling to avoid any possible delays.

An LD plant consists of the following major constituents:

1. The vessel including foundations, rotating gears, etc.
2. The lance including its auxiliary gears.
3. The hood and the waste gas treatment plant.
4. The material handling and storage facilities.
5. Instrumentation and control pulpit.
6. The vessel lining and wrecking accessories.

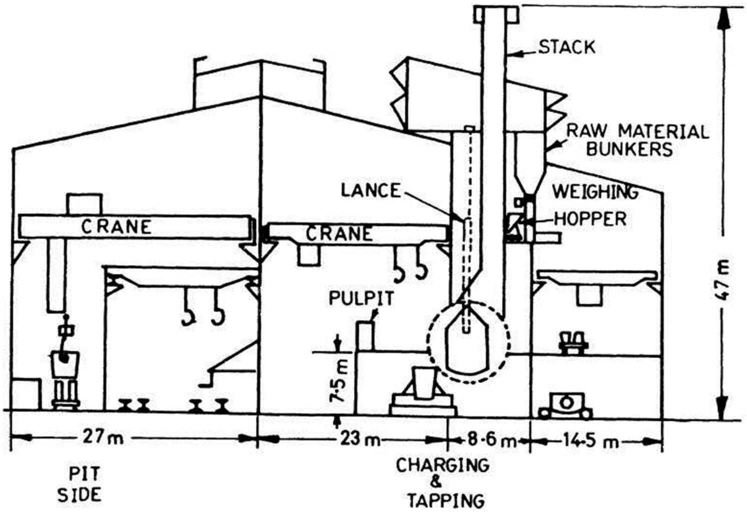


FIGURE 15.3. Simplified cross-sectional view of the LD shop in Rourkela Steel Plant (after Trenkler [23]).

15.2. LD-Vessel

The vessel in which refining is carried out in an LD process is nearly similar in shape to the Bessemer converter. It is either referred to as an LD vessel, an LD converter or a basic oxygen furnace (BOF). The vessel is divided into three segments viz. the spherical bottom, the cylindrical body (or shell) and the conical top. Each of these is a welded construction of non-ageing steel plates. In the earlier designs the bottom was detachable to help cooling and relining after the campaign was over. The modern vessels are, however, without joints, i.e., the segments are welded to form one single piece. A vessel without joints is much safer, warps very little, and is cheaper to construct than one with joints. The segments are welded to achieve smooth continuity at the joints to help maintain the form of the vessel. The top may be concentric or eccentric to the rest of the body. The eccentric shape is adopted for the following reasons:

1. The slag, as well as metal removal, is easy.
2. The gases, ejections, etc., come on one side and regular cleaning of only that side is needed.
3. The construction of hood and insertion of a lance is easy.
4. The lance is protected and controlled in a better way since the hood is inclined.

It however suffers from the following disadvantages:

1. The lining is not symmetrical and hence it is more difficult to lay.
2. The shop is congested since the charging and tapping are carried out on the same side.

In spite of these disadvantages the eccentric shape is still almost universally adopted because of the above-mentioned advantages.

The vessel top is a truncated frustum of a cone staggered by nearly 8-10° to the main axis of the converter and is obliquely attached to the cylindrical body. A steel ring is welded from inside at the junction of the bottom and the body to support the permanent lining of the vessel. In the early designs the converter was tapped from the month. In the modern designs a tap hole is provided at the junction of the cylindrical body and the conical top, opposite the shortest wall height of the conical top. A detachable wear plate is attached to the flange of the tap hole to support its lining.

The heavy weight of the vessel, including those of the shell the refractory lining and the charge, is uniformly taken over by a steel ring which is either welded or attached, in a detachable fashion, to the shell in the plane of its center of gravity. Two trunions are attached to the ring to support the vessel freely in split bearings. The pedestals of the bearings are firmly anchored in the concrete foundations. The tilting gear is attached to one of the trunions. The vessel is capable of rotation through 360° but in practice it rarely exceeds 220°. It is rotated by an electric motor through a system of reduction gears. Small vessels have two pinion drives whereas the big vessels are provided with four pinion drive tilting arrangements. The trunion may be coupled with or directly mounted on the pinions.

15.2.1. Vessel Design

For a given capacity vessel the bath depth is suitably chosen in the range of 110-180 cm depending upon the capacity. The depth should be maximum to prevent damage to the bottom during lancing. The trend is that the height to diameter ratio of about 1.5 is preferred. The bath area increases with increasing capacity since the depth is not increased in the same proportion. The bath area is generally 50% of the corresponding value in an open-hearth. The specific volume of the vessel was earlier just over one cubic meter per metric ton capacity but it was reduced down to 0.75 m^3/t capacity. The modern trend is towards one m^3/t again. The total lining thickness depends upon the quality of lining. It however varies between 600-1000 mm in thickness. The wear lining thickness is decided in terms of wear at the rate of 1-2 mm thickness per heat to obtain the designed life. The height of the vessels varies from 7 to 10 m. The nose diameter and angle are chosen with reference to problems of heat loss, erosion, skulling, stability of nose lining, charging ease, etc. and are usually around one-third the shell dia and about 67° respectively.

The early vessels had capacities around 30-50 t but the modern vessels are up to 400 t capacity. As the vessel capacity increases, the diameter of the vessel is increased since bath depth is nearly the same for vessels.

The major dimensions of few modern different capacity vessels are shown in Table 15.1.

Table 15.1
Nominal Dimensions of LD Vessels of Different Capacities

The dimensions are quite approximate and are only meant to have some relative idea about the sizes.

Vessel capacity, t	30	75	200	300
Height of shell, m	7	7.5	9	9
Dia of shell, m	4	5.5	6.7	8.5
Dia of bath, m	2.5	4.0	5.0	6.5
Bath depth, m	1.1	1.3	1.5	1.8
Dia of nose, m	1.2	1.65	2.3	3.5

Table 15.2
Some of the Important Dimensions of Vessels at the Rourkela Steel Plant

Capacity	40/50 t	60/66 t
Shell height, m	7.98	7.5
Shell dia, m	4.6	5.0
Bath dia, m	2.85	3.60
Vessel volume, m^3	40	53.84
Nose dia, m	2.60	2.80

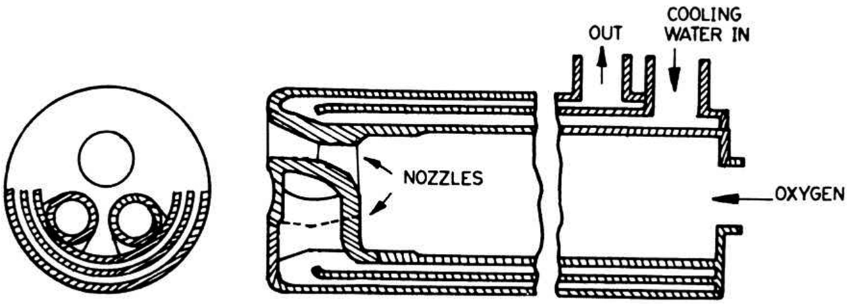


FIGURE 15.4. Oxygen Lance.

15.3. The Oxygen Lance

Oxygen gas—the refining agent—is fed to the furnace through a water-cooled lance as shown in Figure 15.4. The lance is made of three concentric steel tubes to circulate water around the central tube and pass oxygen through the innermost tube. Since the tip of the lance is exposed to a very high temperature, water cooling is made more effective over thereby using a copper tip which is welded to the steel tubes.

The lance is nearly 8-10 m long and its diameter varies with furnace capacity in the range of 20-25 cm. Water requirements are around 50-70 m^3/hr at a pressure of 5-7 kg/cm^2. The lance is suspended by a wire rope and can be inserted in or withdrawn from the furnace by means of an electrically operated lance gear. Jigs are used to hold the lance in a fixed blowing position. Safety devices are employed to automatically withdraw the lance, should the cooling water temperature rise above a maximum specified safe value (≅ 40°C). Extra holding devices are provided for emergency, should the usual arrangements fail. An additional lance is provided as a stand-by to immediately replace a faulty one, to continue the blow. It takes a few minutes to replace a faulty lance so that the heat can proceed without any harm done to it.

15.3.1. Lance Design

The LD process was developed using a lance with a cylindrical nozzle. The physics of a jet issuing from such a nozzle was hardly understood at that time. The drawbacks of using a cylindrical nozzle for steel making were, therefore, unknown. The successful development and commercial adoption of the LD process later led to the study of the physics of supersonic jets, and thereby the development of a proper lance design.

It is now known that the supersonic jet issuing from the nozzle of a lance in an LD process should penetrate the bath adequately and that the area of its impact on the bath should be maximum. These conditions are essential chiefly for efficient refining, i.e., for decarburization as well as dephosphorization.

The static pressure in a jet from a cylindrical nozzle, as it emerges into the ambient atmosphere, is more than the atmospheric pressure. It, therefore, interacts with the atmosphere generating shock waves, and the velocity of the jet decreases with damped fluctuations. This affects the bath penetration as well as area of impact adversely.

Much of these drawbacks are eliminated if a *convergent divergent* Laval-shaped nozzle is used. The static pressure in a jet from a laval-shaped nozzle disappears within a short distance from the nozzle tip and hence it does not interact much with the ambient atmosphere. The velocity of the jet decreases more uniformly with fewer damped fluctuations, if the inside and the outside diameters of the nozzle are properly designed. The velocity at any point in the stream is more than at the corresponding point of the stream from a similar size cylindrical nozzle under similar conditions of blowing. The resultant bath penetration is more in the case of Laval-shaped nozzle than that due to cylindrical nozzle. The Laval-shaped nozzle is, therefore, universally adopted.

Oxygen is generally blown at 8-10 atmospheres pressure through a Laval-shaped nozzle so that the jet issuing at the nozzle exit is supersonic and generally has a velocity between 1.5-2.5 times the velocity of sound (mach). The characteristics of a supersonic jet, as emerging from a Laval nozzle and impinging on the liquid metal bath are shown in Figure 15.5. The jet has characteristically a potential core, a supersonic core and a subsonic region. As the jet travels, its velocity is retarded due to the ambient atmosphere, the supersonic surrounding zone expands radially. The potential core may normally extend, to a length of about 15 times the diameter of the nozzle from the nozzle-tip. The velocity of the supersonic core gradually decreases until at a distance of about 30 times the nozzle diameter from the nozzle tip, the jet becomes wholly subsonic. This point marks the end of supersonic core and the development of a fully expanded jet. The velocity of the jet decreases hereafter more rapidly.

For a given size nozzle, the length of the supersonic core depends on the blowing pressure and the ratio of the densities of the jet-gas and the ambient atmosphere. Although the density of the ambient atmosphere in the LD process changes during the blow, an average value is assumed to calculate the length of the supersonic core.

During the blow the jet should be expanded to obtain maximum impact area at the bath surface. At the same time, it should also penetrate the bath surface to a maximum extent. The depth of penetration of a jet in a metal bath varies inversely with the impact area at the bath surface. The requirements, therefore, can only be met at the optimum.

In the blowing position the lance height from the still bath level has to be more than the length over which the supersonic core extends in the jet, since the jet is not fully expanded until that point. In actual practice the proper height would be around 40-50 times the diameter of the nozzle.

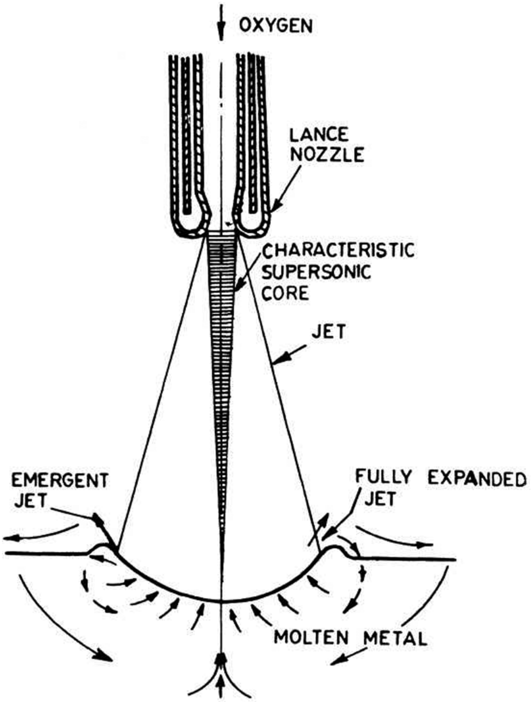


FIGURE 15.5. The effect of supersonic jet on bath circulation. The bath circulates outwards on the surface and upwards on the vessel axis. The characteristic supersonic core is also shown.

The depth of penetration of a jet in a bath can be assessed in terms of the jet force number as:

$$\text{JFN} = \frac{\text{Gas pressure} \times \text{Nozzle throat dia}}{\text{Height of nozzle}}$$

It may be mentioned here that decarburization is faster for greater values of JFN and dephosphorization is faster for reverse conditions.

The gas flow rate from a nozzle can be calculated by assuming a frictionless and adiabatic flow through the nozzle. The jet behavior does not alter adversely even if the actual flow rate deviates by ± 20% from this nominal value.

In the case of supersonic oxygen jet the exit temperature may be lowered by as much as 100-120°C because of the Joule-Thomson effect. Circulating water cools the lance up to the nozzle throat and the nozzle itself is essentially cooled by this Joule-Thomson effect, i.e., it is gas cooled.

15.3.2. Multi-Nozzle Lance

A single-nozzle lance was adequate for the early small size vessels. Modern big vessels need a multi-nozzle lance as show in Figure 15.4. The blowing action of a multi-nozzle lance is quite different from that of a single-nozzle lance. The axes of nozzle in early multi-nozzle lance are inclined to the vertical axis of the lance by nearly 10°, so that they do not interact with each other. If the angle is increased, the depth of penetration decreases and hence 10° inclination is used to obtain non-interacting jets without appreciably lowering the penetration power. It is now increased to 20° to vertical. The impact area is considerably increased if a multi-nozzle lance is used as shown in Figure 15.6. Therefore the multi-nozzle lance has become fairly popular for big furnaces and is claimed to possess the following advantages:

The situation has since then improved further and seven- and eight-hole lances have been used for vessels of large capacities like 350-400 metric tons. A survey on lance designs used in European steel plants reveal that for converters from 50-380 metric ton capacity 46% used six-hole lances, 32% used five-hole, 20% used four-hole and only 2% have less than four-hole tips. The effectiveness of using more nozzles can be understood from the example of Tata Steel. They replaced their SMS 2 by using 2 vessels of 130 t capacity each, in 1983, adopting a three-hole lance. As late as 1993,

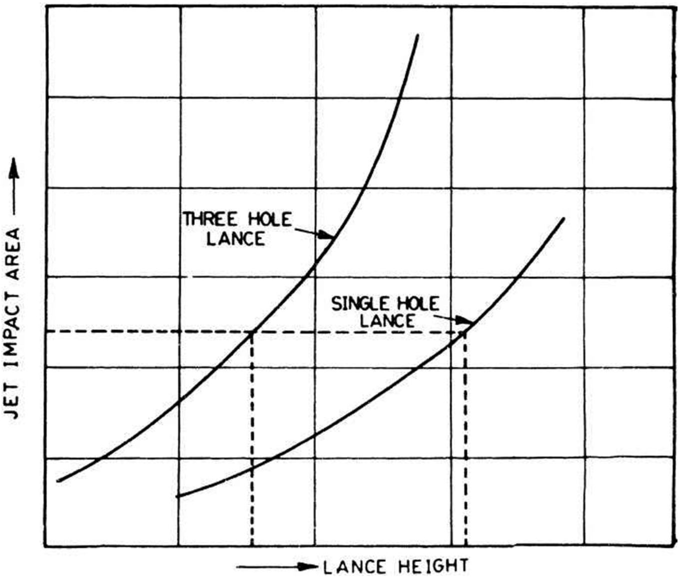


FIGURE 15.6. Comparison of performance of single- and three-nozzle lances. (after Smith [25])

while installing another 2-vessel, 130 t, LD shop in their SMS 3 similar three-hole lance was still preferred. But now in both of these shops three-hole lances have been replaced by six-hole lances because of their proven better performance. The nozzles are at 17° inclination to vertical axis.

A detailed comparative study of three- and six-hole lances carried out by them proved the following beyond doubt [24]:

(*i*) increase in total throughput oxygen without any adverse effect, at the same pressure

(*ii*) improvement in jet spread on the metal bath. These two lead to:

(*a*) less slopping and spitting and thus fewer mechanical losses, in turn better yield

(*b*) improved mixing of slag and metal, thereby better mass transport and hence better rate of refining

(*c*) less danger of burning vessel bottom in spite of increased oxygen blowing rate

(*d*) better gas recovery and improved lining life (*e*) better thermal balance and hence more of coolant scrap or ore is required

(*f*) improved slag basicity from around 3 to 3.5

(*g*) much improved turndown %P, from earlier 0.034 % to 0.017%

(*h*) high residual Mn in the bath so that less of Fe-Mn is subsequently required for deoxidation.

Albeit along with other improvements, the use of a six-hole lance has improved the lining life beyond expectations. Lining life of more than 1200 heats for tarred dolomite, quite unheard of anywhere in the world, has been achieved in Tata Steel [26]. The slopping is practically absent.

Lance life is usually determined by the life of the nozzle. Failures of a lance may be due to faulty cooling, manufacturing defects, internal stresses, differential expansion of tip and steel tube, etc. Although as high as 5000 heats were claimed to have been made by the same lance, the usual life does not exceed a few hundred heats.

15.4. The Hood and the Waste Gas Treatment

In the area where the oxygen jet impinges on the metal bath during refining, an intense hot zone is developed due to the exothermic reactions. The hot zone temperature is estimated to be over 2500°C and as a result nearly a per cent of the metal charge is blown in the gases as vapors. When the gas containing iron vapors comes in contact with oxygen from the atmosphere, iron vapor forms red-brown and dark-brown solid oxide particles. About 80% of the particles have sizes less than 0.8 microns and at least 20% are in the range of 0.1-0.3 microns. The gases in which fine dust like this is suspended appear as dense brown fumes. In old plants these gases are led through a hood into the stack and then let off in the atmosphere.

The hood and the stack are steel plate shells which are lined from inside with a course of fire bricks and then plastered by clay. The hood is water-cooled by a complex cooling system. A water-cooled sleeve is provided for inserting and guiding the lance in the vessel through the hood. One or two water-cooled chutes are fitted in the hood for charging solids when the vessel is in vertical position. Access doors are provided for cleaning the hood. The hood is sealed at the bottom by two water-cooled removable dampers to prevent infiltration of air, while the converter is not being blown (inclined position).

The hood is supported clearly off the vessel in such a way that enough atmospheric air is available for burning the carbon monoxide at the mouth of the vessel. An explosion may occur if CO is not entirely burned in the lower part of the hood.

The dense dark-brown fumes were let off in open atmosphere in the early designs, although these are quite hazardous for public health. The problem of fume formation did worry the originators of the process in spite of all the advantages that accrued from oxygen blowing from top for refining iron. The problem has now been solved to the satisfaction of all concerned. Necessary gas cleaning systems have been developed to remove over 95% of the dust in the gases. A million metric ton steel plant may blow and then recover nearly 10,0000 t of iron oxide dust which could be recycled into the ironmaking process.

A side-by-side waste heat recovery system has also been developed since the gases do possess a substantial amount of sensible heat. Many plants have installed waste heat boilers but it has been observed that they add to the problems. The intermittent nature of blowing necessitates either synchronizing of blowing operations or installation of secondary burners to raise steam during the idle period between two blows. Boiler cleaning is a big problem because of the dust-laden gases. The whole economy of waste heat recovery depends on how best the steam that is generated is utilized.

The gas cleaning consists of preliminary gas cooling by water sprays and dust collection by wet or dry methods as shown in Figures 15.7 and 15.8.

The Venturi scrubber is quite a popular wet dust collecting system. The dust-laden gases are accelerated through a Venturi throat, and water jets are directed on the gas at right angles. The dust particles get wet and become heavy, and are separated by cyclonic or centrifugal action in a separator. The wet slurry is further thickened and filtered, if necessary. There is no danger of explosion and the cost of installation can be much less if wet slurry can be disposed of. The Venturi system requires less ground space but substantial water.

An electrostatic precipitator can clean the gas in dry or wet condition. The gas is cooled by water sprays to 300°C in flues and in spark boxes to eliminate large dust particles. The gases enter the electrostatic precipitator at 130-150°C. Due to the Corona discharge in the tubes solid particles get

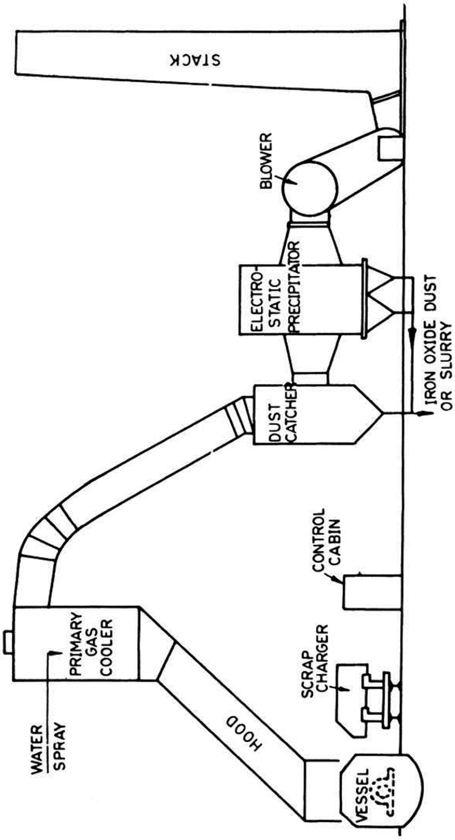


FIGURE 15.7. LD gas cleaning equipment with primary cooling by water sprays followed by wet or dry electrostatic precipitators.

charged and move to the neutral electrode. The dust collected on the neutral electrode is periodically scrapped and collected for disposal. The gas velocity during precipitation is very low. Wet electrostatic precipitators are also in use. Flowing water film over the neutral electrode washes down the dust in the form of a slurry. The operating cost of electrostatic precipitator is low and cleaning efficiency is high. The dust, being rich in iron oxide, can be readily pelletized and charged in a blast furnace.

In conclusion, it can be said that if full utilization of steam is possible, a full boiler, electrostatic dust collector system is a best choice. If water is available in abundance and steam is not required, a Venturi scrubber may be used. A half boiler system is recommended if steam demand exists.

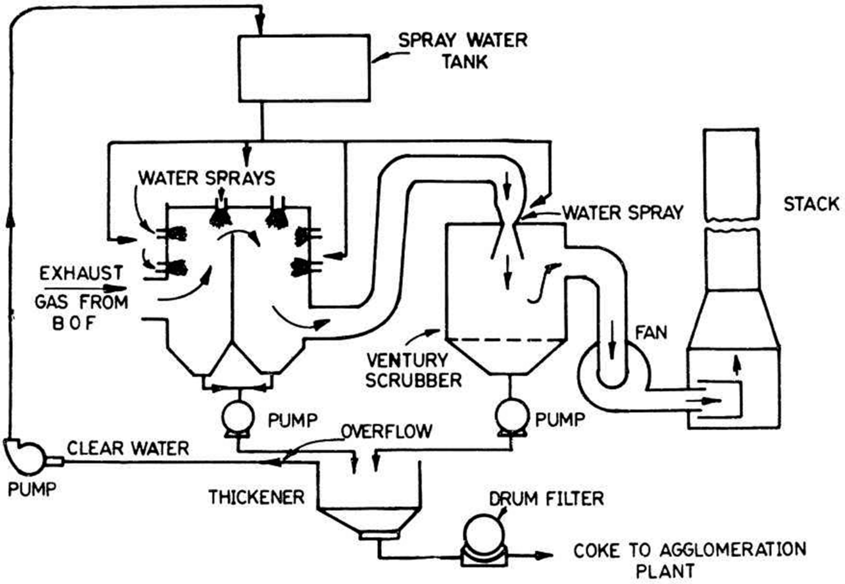


FIGURE 15.8. LD wet gas cleaning equipment with a Venturi scrubber.

15.5. Vessel Lining and Wrecking Accessories

As compared to conventional processes of steel making, the refractories used in lining BOF vessels are expected to withstand more severe chemical and mechanical action. The attack of molten metal and slag is severe in LD vessel if silicon and phosphorus contents of the iron are more and/or if dead soft steels are to be produced. The exhaust gases laden with high concentration of iron oxide fumes are very corrosive. The mechanical damages, from solid and liquid charges, due to violent liquid movements during refining and due to vessel movements are quite severe. The lining is also subjected to thermal shocks and spalling because of frequent changes in the state of the vessel.

15.5.1. Refractories

The century-long tradition of using tarred dolomite for basic steel furnaces led to its extensive use in the LD vessel lining as well. It is increasingly being replaced by dolomite enriched with magnesite, natural magnesite, or even synthetic magnesite prepared from sea water. Whatever material and in whatever form it is used, all contain tar when delivered to the furnace for making the lining.

The forms are:

(*i*) tar bonded green bricks,

(*ii*) tar bonded and tempered bricks to improve hot strength on initial heating and,

(*iii*) fired bricks, tar impregnated after firing.

The necessity of tar will be clear if the mechanism of wear of the lining as shown in Figure 15.9 is properly understood. Tar first gets oxidized and disappears. The decarburized zone reacts with slag and forms the fluxed zone, which is slowly worn out. The fluxed zone is always separated from the tar bonded portion by a thin layer of decarburized zone. The lining, therefore, wears out slowly. New fluxed zones and decarburized zones are continuously formed. These are generally each 1 mm in thickness. The rate of wear is thus determined by the rate of formation of the decarburized zone.

Several different materials are used in lining an LD vessel. These are broadly as follows:

1. An air settling ramming mix to level-out surface between the shell and the lining.
2. Tar-impregnated fired bricks for semi-permanent lining.
3. Becked tarred bricks for wear-lining.
4. Tap hole sleeve brick embedded in castable magnesite.
5. Tar impregnated fusion cast or rebonded bricks at the top of the nose.
6. Tarred ramming mass to ensure solid (with minimum of pores) construction of brickwork.
7. Dry powder to level-out the brickwork rings during relining.

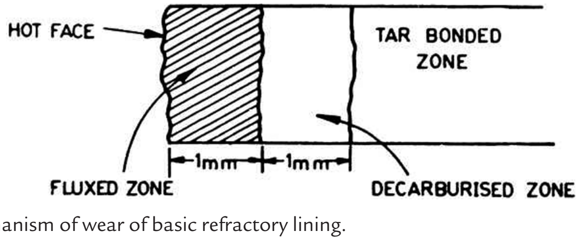


FIGURE 15.9. Mechanism of wear of basic refractory lining.

In general the total weight of refractory material required for lining is nearly three times the nominal capacity of the vessel for steel making. Two-thirds of this weight is of the wear-lining and is to be replaced after the campaign is over.

The life of an LD vessel lining varies considerably with its own chemical composition, method of making the bricks, lining design vis-à-vis vessel design, care exercised in laying the lining, charge material composition, vessel operation, blowing conditions, fluxes added, slag chemistry, temperature inside the vessel during blowing, quality of the end product, and so on.

The lining is subjected to erosion due to various reasons such as chemical attack, thermal stresses and mechanical wear. Slag chemistry and its actual temperature are chiefly responsible for dissolution of lining in the slag. The oxidation of carbon in the lining due to high oxygen pressure of blowing also erodes the lining. Water leakage may lead to spalling of the lining. The lining is always subject to variations of temperature during its use for steel making; this can cause spalling. The more the variations, the more the spalling. Charging of scrap, hot metal and other additions can lead to mechanical wear. Dust-laden gases with high velocity may be damaging to the cone-lining in particular. Every attempt must be made to obtain the highest lining life. Over the years many such improvements have been introduced and substantial improvement in lining life has been obtained. This directly affects the economy of steel making.

The earlier lining life of dolomitic lining was approximately 100-200 heats and that of dolomite enriched with magnestic was little over 200, of only magnesite 200-300, of burned and tar impregnated magnesite 300-350, and of fusion cast magnesite was over 600 heats. In general 3-5 kg of refractory material was consumed per ton of steel produced in an efficient plant.

These all have been improved a fair bit in the last decade. Lining life of 1200-1400 heats has been obtained with prefused material. Introducing many other innovations, the Tata Steel [27] has been able to achieve over 1200 heats from tarred dolomite lining, quite unheard of anywhere in the world. The details of this achievement would be discussed later while discussing the LD refining reactions.

The most important criterion about lining life is that a uniform production rate should be achieved when more than one vessel is provided in the shop. If one out of two vessels is to operate at a time, while the other is being relined, the lining life should be more than the relining time required under the conditions provided in the shop. If two out of three vessels are to operate at a time, while the third is being relined, the lining life should be more than the lining time required for lining two vessels. This is provided in the original design but this timetable becomes inaccurate with the passage of years as the plant gets old, because of the inefficiency developed in the plant. The main reason for this is the wear and tear of the equipment with time. Any substantial increase in the lining life later is able to offset this disturbed timetable of relining. This has been observed by the Tata Steel. Their improvement of lining life from designed life of 160-180 heats to over 1200 heats has facilitated even major repairs of equipment in the shop without hampering production, because even one vessel can keep the production going uninterrupted for a much longer period. The increased lining life directly reduces the refractory consumption per metric ton, and thereby the cut in production cost. The indirect benefits of increased lining life are much greater in terms of keeping the production uninterrupted.

Because of this exceptional lining life, Tata Steel is considering the proposal to operate three vessels out of four in the two different two-vessel-SMS shops in an integrated way at a time while repairing the fourth one to improve upon the productivity without additional investment. Each of the four vessels, in turn, and one from each shop alternatively, will be down for repairs [26].

Hot repairs of LD vessel lining, as in BOH, in order to improve its life is not very popular because of loss of production and heat content of the vessel from the previous heat. These losses are generally not offset by way of correspondingly increased life of the lining.

15.5.2. Lining

The lining of an LD vessel at the Rourkela Steel Plant is being described here as an illustration. The lining is shown in Figure 15.10. The permanent lining consists of magnesite and magnesite water glass mortar. Cardboard pieces are inserted at certain regular intervals to provide room for expansion of brickwork on heating. Next to the spherical bottom shell a layer of fire bricks is used before magnesite lining. Specially shaped bricks are used at the joints of the body shell to the bottom and to the nose. The semi-permanent lining of the body rests on the horizontal steel ring welded from inside. The wear-lining is made of tar bonded dolomite bricks and tarred-dolomite ramming mass in between the two brick linings. No mortar is used. The tap hole is made from cold setting magnesite ramming mass,

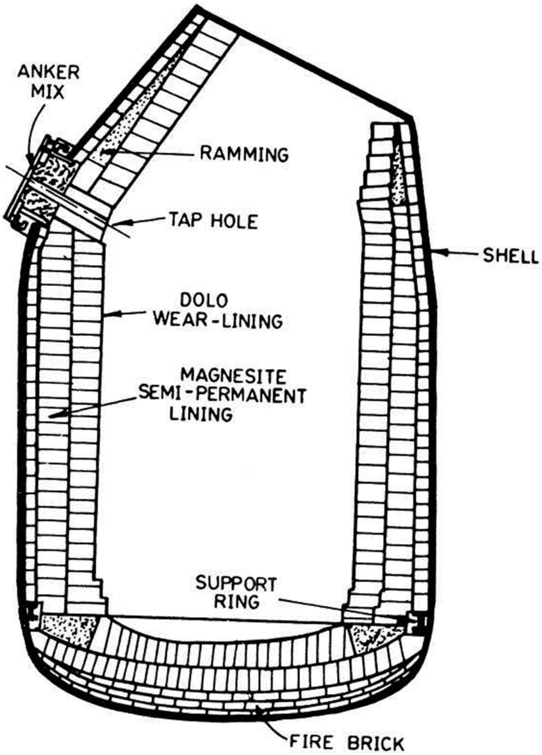


FIGURE 15.10. Lining details of a LD vessel.

Table 15.3
The Details of LD Vessel Lining at the Rourkela Steel Plant

Vessel part	*Type of lining*	*Thickness of lining, mm*
Bottom	fire brick	1 × 75
	magnesite brick	2 × 65
	magnesite brick	1 × 250
	tarred dolomite brick	1 × 350
		805 total thickness
Side walls	magnestic brick	1 × 125
	tarred dolomite brick	2 × 350
		825 total thickness

Nose	magnesite brick	1 × 125
	tarred dolo mass	1 × 190 max
	tarred dolomite brick	1 × 350
		665 max. and 475 minimum thickness
	Materials required for lining a 50/60 t converter are	
	Total No. of dolo bricks	3821
	Wt. of each dolo bricks	31.2 kg
	Total wt. of dolo bricks	12.4 t
	Wt. of ramming mass	9.0 t
	Wt. of magnesite lining	57.9 t
	Less waste of bricks during laying	6.0 t
	Total wt. of refractory	194.3 t

the trade name of which is the Anker-mix. The size of the new tap whole is 90 mm and tapping time is 4.5 minutes. When the tapping time comes down to 3 minutes, it is changed.

The average lining life at Rourkela is 100-120 heats. Such a low lining life is due to the high silicon content of the iron and poor quality of other raw materials used in the plant. The consumption of refractories is, therefore, 2-3 times that in any other good plant. Of late, the LD practice has been modified to incorporate bottom inert gas purging wherein argon is bubbled through the bottom, as in the case of Bessemer, while blowing oxygen from the top in the usual manner. This will be described later in detail. It will suffice here to know that the LD lining design has to be changed to incorporate this feature.

15.5.3. Wrecking and Relining the Vessel

After the last heat is tapped the lining is cooled for few hours and then the wrecking operation begins. A special air cooler is used for this purpose. All the remaining wear lining is removed. The semi-permanent lining is repaired wherever damaged and the wear-lining is entirely built up a new. Usually one to two shifts are required for wrecking and about 7-8 shifts for relining. In a two-vessel plant dolomite lining is considered satisfactory, but in a three-vessel plant magnesite may be necessary for wear-lining.

For wrecking and relining, a platform is lowered in the vessel for the crew to stand on and work. Material movements are carried out by cages. Covering of the roof is provided to protect the crew from falling pieces.

A relined vessel is preheated before starting the heat. Coke fire is prepared in a perforated drum and when white hot, the coke is pushed on the charging chute and from there it is dropped in the vessel. A coke bed of nearly 200 kg is prepared in the vessel prior to dropping the burning coke. Oxygen lance is then lowered in to supply oxygen for burning the coke. Initially the oxygen pressure is one atmosphere. After 3 hours it is raised to 2 atmosphere when the lining is expected to be red hot. Heating is continued for two hours more. More coke is added regularly during the 5 hours of heating period. By this time the vessel becomes white hot, i.e., at 1650°C and is ready to receive the charge the first heat.

15.6. Materials Handling and Storage Facilities

The hot metal is stored in a mixer located suitably in the shop. The metal is brought in open top ladles on bogies and with the help of overhead cranes it is charged in the vessel by tipping the ladle. Scrap is stored in the scrap yard and is supplied in required lots in special charging boxes. Overhead cranes are used to lift the boxes and charge scrap in the vessel. The scrap is nearly poured in the vessel to keep the time of charging to a bare minimum. Alternatively charging machines are used that tip the scrap in the vessel in place of overhead cranes, which are free for other jobs. Refined steel is tapped in teeming ladles and slag pots. The shop is provided with one or more hot metal charging cranes, slag pot handling cranes, scraps handling cranes (machines), ladle cranes, etc.

The other solid charge materials like lime/limestone, ore, mill scale, spar, bauxite, silica, etc., are stored in a series of overhead bunkers. The materials are usually transferred from wagons to the bunkers by a belt conveyor system. The materials are discharged in weigh hopper and then on to the conveyor belt to transfer them to the hopper on the top of the converter. In some earlier designs cars are used to transfer the material to converter hoppers. The materials are thus automatically weighed and charged through a system of hoppers and conveyor belts as shown in Figure 15.2. Weights of these charges are automatically communicated to the control pulpit.

Storage bunkers at the Rourkela Steel Plant:

Vessel capacity	*40/50 t* (*2/3 operation*)	*60/66 t* (*1/2 operation*)
2 Bunker for burnt lime	58 t each	80 m^3 each
1 ” ” limestone	52 t	47 m^3
1 ” ” ore	90 t	47 m^3
1 ” ” bauxite	20 t	19 m^3
1 ” ” scale	12 t	19 m^3

15.7. Instrumentation

The measuring, recording and controlling instruments for the variables in a LD shop are located in a control pulpit located in front of the vessel across the working floor. The process is controlled by the operator from this pulpit. A typical set of such instruments provided in the Rourkela Steel Plant is described below as illustrations:

1. Depth indicator for the oxygen lance. … range scale } 0-12 m and other 0-100 cm.
2. Control knob for lance position.
3. Control knob for oxygen flow. … manual
4. Oxygen on-off valve.
5. Converter tilting control.
6. Lance cooling water. … less/more valve
7. Stack cooling water inlet and outlet temperature measurements and its pressure indicator. … range: 0-150°C and 0-10 kg/cm^2
8. Lance cooling water temperature and pressure indicators. … range: 0-70°C 0-12 kg/cm^2
9. Oxygen pressure indicator. … range: 0-30 kg/cm^2
10. Oxygen flow indicator recorder. … range: 0-125 Nm^3/hr × 100
11. Lance cooling water flow recorder. … range: 0-70 m^3/hr

12. Charge materials weight indicator and control.
13. Lance grip control.
14. Bath temperature indicator/recorder.

The extent of manual or automatic control of the process variables varies as per the design. The plants are increasingly being controlled automatically to the extent that several plants are fully automatically controlled. Even the early plants with least automation were provided with computers to calculate the weight of charge materials and the total amount of oxygen to be blown during refining.

The control of steel refining and arriving at the correct turn-down chemistry requires the knowledge of oxygen dissolved in liquid steel. This information helps to calculate the ferro-alloys to be added for required deoxidation more correctly to finish the heat more appropriately. Sensors based on using solid electrolytes and measuring the EMF generated at the galvanic cell so formed gives the level of dissolved oxygen content of the bath. These are disposable tips that are dipped in the bath for such measurements.

Automation of SMS has been standardized in terms of Levels 1 to 5 and is adopted starting from Level 1 depending upon the requirements and their applicability. Level 1 is the lowest level of automation adopted and progressively the 5th level is the highest level of automation as standardized world over. Computer software is available accordingly for use.

It is almost essential to carry out the preliminary analyses by quantitative automatic spectrochemical methods. A *quantovac* or a spectrometer may be suitably located in the plant. Pneumatic channels are laid to carry samples to the laboratory. The analysis of several elements is accomplished in a few minutes and the report is sent back to the shop floor by teleprinters. The final analysis may be done later for preservation of records.

The bath temperature is measured by an immersion thermocouple and is registered in the pulpit. Consumable Pt/Pt, Rh thermocouple junction is used.

CHAPTER 16

LD PROCESS—DESIGN AND PRACTICE

In This Chapter

- Charge Materials
- Heat
- Characteristics of LD Process
- Process Control
- Process Economies
- Output
- Slag Characteristics
- Yield
- Product Characteristics

16.1. Charge Materials

The charge consists of molten metal, cold pig iron, steel scrap, lime/limestone, dolomite, fluorspar, bauxite, iron ore, mill scale, gaseous oxygen, etc.

16.1.1. Hot Metal

The classic LD or the modified LD process, both because of oxygen refining, are often used to make soft steels for flat products. This, coupled with general requirements for extra low sulfur specifications, has necessitated a constant reduction in sulfur contents of the final steel products.

The adoption of continuous casting also requires that the sulfur should be below 0.02% to produce a transverse-crack-free product. In special steel plates, the normal sulfur specification is 0.01% and there is demand for extra low sulfur steels with as low as 0.001% (10 ppm) coupled with very low carbon, for example pipeline steels.

Due to speed in refining, the LD process does not allow enough time for effective sulfur removal during refining. The oxidizing conditions of LD process are not at all conducive to desulfurization. Any attempt to bring about sulfur removal in LD refining necessitates high basicities, for which extra lime must be charged for making the slag, which is detrimental.

In modern LD practice sulfur removal during refining is not the goal. This results in reduction of lime load, otherwise required for sulfur removal. This also saves 3-4 min of blowing time, thereby affecting the production economy merely by deleting the sulfur problem solution inside the vessel.

This, however, necessitates that sulfur in the charge iron must be close to the final specification level. Under the conditions of ironmaking in general, such low sulfur levels cannot be obtained during ironmaking. The external desulfurization of molten iron in transfer ladles is the only answer. This is increasingly being adopted all over the world. The major steel plants in India also resort to external desulfurization. In Tata Steel, calcium carbide and magnesium base mixtures are injected separately or together to obtain sulfur contents below 0.001%.

Such molten irons are ideally suited as a charge material for the LD process. The silicon content of such iron must be as low as possible, preferably below 0.5%, usually below 1.0%. Every extra bit of silicon requires extra lime charge to make slag of the required basicity, hence slag volume goes up with silicon content even if it is possible to tackle it during refining.

The LD process is capable of removing more than 90% of the phosphorus present in the iron. The final phosphorus is often required in the range of 0.025%. The usual phosphorus level in Indian irons is around 0.2–0.3%. From this level it is to be brought down to 0.025% for average applications and for special specifications the requirement is for much less %P.

A certain minimum manganese content was considered necessary for hot metal charge in BOH practices. It results in enough manganese in the bath to prevent over-oxidation of the bath. In an LD process however, particularly for producing soft steels, the manganese has to be removed to a very low level, at times even by adopting slag flushing which unnecessarily prolongs the blow and reduces the yield. High manganese in slag also tends to retard dephosphorization of the bath. Manganese content in the range of 0.5-1.1% is tolerated but 0.7% is considered better.

The usual carbon level of 3.5-4.5% is considered all right. The analysis of iron used in the LD process is as follows:

	Required %	*Range use %*
Carbon	4.1-4.3	4.00-4.50
Phosphorus	0.1-0.25	0.05-0.45
Silicon	0.50-0.85	0.65-1.40
Manganese	0.5-0.8	0.40-2.50
Sulfur	0.02-0.03	0.02-0.06

The temperature of the hot metal at the time of charging is usually around 1250-1300°C. The proportion of hot metal in the charge varies in the range of 75-90%, i.e., 10-25% scrap depending upon its availability.

16.1.2. Cold Pig Iron

Under certain circumstances it may be necessary to include cold pig iron in the charge to consume the excess of pig iron produced in the plant. Big pieces of pigs up to a total of 3-4% of the charge weight can be included along with the scrap. Granulated cold pig iron however, tends to settle down at the bottom and it does not come up until towards the end of refining. It causes a reaction to take place while tapping the vessel that may even necessitate a small reblow.

Cold pig iron, in general, tends to prolong the blow by disturbing the usual blow characteristics of the scrap plus hot metal charge, hence is neither preferred by operators nor recommended. However, if granulated cold iron is charged while blowing it should be possible to absorb all the iron in an integrated steel works to balance iron and steel production.

16.1.3. Fluxes

Thermal energy available during the heat is best utilized for melting scrap and at times for disintegration of iron ore, hence is not recommended to be utilized for calcination of limestone added as flux. Use of limestone as a flux is, therefore, discouraged. There is still another reason to prefer lime as a flux instead of limestone. Lime is immediately available for fluxing and formation of slag. If limestone is added as a flux, lime will be available only after calcination, i.e., the slag formation will be delayed. Efficiency and economy of the process very much depends upon the formation of thin, oxidizing and basic slag as early as possible during the blow.

The use of bauxite, silica-sand, and fluorspar as fluxes brings lime quickly in solution. Some plants use dolomite to help early formation of slag, but the operational results do not conclusively prove its utility in this respect. The use of dolomite as a flux may be useful in safeguarding the lining, but it adds one more item to be handled on the shop floor.

Besides the available lime content of lime, its reactivity is an important parameter to judge its quality as a first approximation. The reactivity can simply be assessed, in terms of its particle size. A lumpy lime is slower to dissolve in slag than fine powdered lime. Conversely fine powder tends to fly off. A small granular form of lime is best. It is estimated that the use of lime in the form of fine powder may successfully deal with iron containing more than the usual maximum limit of 0.4% P, as in the Q-BOP process.

Lime consumption varies around 2 to 5% of the weight of metal charge. The amounts to fluorspar, bauxite and silica, added as flux, very much depend on the local conditions, but do not exceed more than a few kg per metric ton of steel made.

The flux addition depends upon the amount of Si and P to be oxidized from the charge during the refining. The right quality of the slag is made by the addition of lime and/or dolomite in calcined form. The amount added is often empirically calculated as:

$$\text{Kg of burnt lime/t} = 19 \times [\%\text{CaO}/\%\text{SiO}_2] \times \%\ \text{Si (in the charge)}$$
$$\text{(of metallic charge)}$$

and $\{\text{kg of burnt lime/kg of burnt dolo}\} = \{2 + 0.3 \times (\%\text{CaO}/\%\text{SiO}_2)\}$

16.1.4. Scrap and Ore

Scrap and ore are used as coolants to best utilize the excess heat energy available during refining. The use of scrap as a coolant is a must since the home scrap generated in the plant has to be recycled back into the process. The LD process can take up to 25% of the metal charge as scrap.

Since oxygen gas is used as an oxidizing agent, iron ore is not required for the same purpose. Some operators, however, use it to hasten the formation of an early slag. Iron ore can act as a coolant but is not preferred because the available excess thermal energy can be economically used in melting maximum load of circulating scrap in an integrated steel plant. If scrap is not available, iron ore is added as a coolant to the tune of up to 100 kg per metric ton of steel. In its cooling effect 1% ore in the charge is nearly equivalent to 3.5% steel scrap.

Early formation of an effective slag is beneficial in many ways, and hence may be preferable to add scrap plus a little ore as coolant. After all, this is a matter of policy and economics.

In the Rourkela Steel Plant, being one of the early LD plants, only 15% scrap is used as coolant. The proportion has been increased in the expanded shop.

16.1.5. Oxygen

It has been observed that the nitrogen content of the bath increases with decreasing purity of oxygen. During the decarburization of the bath from 3.0% to nearly 0.12% C, the nitrogen level remains the same for a given purity of gas. But below 0.12% C the nitrogen level increases if oxygen purity is less than 97%, whereas it decreases if purity is above 99%. Hence oxygen of 99.5% purity is always desirable to keep the nitrogen level of finished steel below 0.0016%. It finally gives nitrogen content below 0.003% in the steel ingots.

The consumption of oxygen per metric ton of steel made varies with the proportion of scrap and ore added as coolants, and also with single- and multi-hole lance designs. The early figures, for 25% scrap as coolant, were about 45 m^3 for a single-nozzle lance and the modern figures are 55-60 m3 using a three-nozzle lance. If iron ore is added as the sole coolant, the oxygen consumption comes down by about 4-5 m3/t of steel.

16.2. Heat

The sequence of operation is as follows:

1. After the previous heat is tapped and slag is drained, the vessel is brought to an inclined position to receive scrap. The scrap is virtually poured in the vessel from the charging chute in one lot. It is kept waiting to minimize delay. The lining is inspected. (3-5 min.)
2. The vessel is rocked to gently drop the scrap on the bottom and is brought back to a horizontal position to receive hot metal. The metal is kept waiting in the ladle for being charged. The metal is charged. (3-5 min.)

3. The vessel is rotated to vertical position, the lance is lowered to its first blowing position, and the oxygen is turned on. (1-1.5 min.)
4. As soon as ignition takes place, part of the lime is charged along with other fluxes to form early slag. The rest of the fluxes are added at predetermined intervals but in the early part of the blow. Some prefer to add all fluxes in one lot after good ignition.
5. The blow continues for nearly 15-25 minutes during which period the lance height may be altered as per established practice. The oxygen is put-off and the lance is raised above. (15-25 min.)
6. The vessel is rocked back to the inclined position, and slag and metal samples are taken out for analysis. The temperature of the bath is measured by immersion thermocouple. Analysis is received. Alloying additions in heavy lumpy form can be added here and 2-3 minutes of waiting time is allowed for homogenization. (5-8 min.)
7. If the analysis and tapping temperature are correct it is rocked in tapping position and steel is tapped in the ladle already waiting for it. Deoxidizers and alloying additions are made in the ladle in the same way as described for open-hearth practice. (4-7 min.)
8. As soon as slag just appears to come out, the vessel is rocked further to tip out the slag.

This gives a tap-to-tap time of 30-50 minutes. Delays due to tap hole repair, cleaning of tip, etc., may increase heat time by a few minutes. If the ancillary equipment and materials are not available in time, the operation is further delayed. The most efficient plants have an average tap-to-tap time of 30-33 minutes, whereas 40-50 minutes is considered a good average heat time for most of the plants. The heat time of high carbon and alloy steels is little more than that for soft steels.

If the analysis is not as it should be, the vessel is reblown for a short duration which may be about 1-2 minutes. For several reasons, the most important being loss in production, everything at command should be adjusted so that as far as possible the necessity for reblow does not arise.

Opinion is divided on retaining a part of the previous slag for the next heat to help form early slag. Those who oppose this believe that it leads to sloping during blowing in the initial stages.

In some plants the lance is located at the highest distance from the bath level and is slowly lowered to required positions during the course of the blow. The lance height depends on the vessel shape and size, raw materials used, and blowing conditions. Some other plants favor a single blowing position while the oxygen pressure is varied during blowing to achieve required refining conditions. The choice is dictated more by experience rather than personal fancy. After the first addition the remaining lime, ore, spar, etc., is added in batches in order to control the bath temperature. If the converter does not have a tap hole, a dam of slag or lime is prepared before tapping. For convertors with a tap hole, a nearly slag-free tapping of the refined steel can be achieved using a device popularly known as a *float.* A large number of such patented designs are available. This is a ceramic device available in various shapes as per tap-hole designs and requirements. The characteristic feature of this float is that its density is in between those of the slag and the metal. The slag density is around 2.5 and that of steel is around 7.5 g/cc, and that of the float is in between these two values i.e., around 4.5 g/cc. As the steel starts flowing out of the tap hole this float is floating in the steel bath and not obstructing the metal flow. When the slag comes along with the steel, the float prevents the slag from entering the tap hole by positioning itself at the slag-steel interface. When all the steel is tapped it will settle down to plug the tap hole, and thereby prevent slag from flowing out along with the liquid steel. This is the device that prevents carry-over of oxidizing slag from the steel making furnace into the ladle or the LF. This carry-over slag, being oxidizing, consumes unnecessarily the ferro-alloys that are added for deoxidation, thereby increasing unnecessarily their consumption and affecting the economy adversely.

16.2.1. Practice at Rourkela Steel Plant

The special feature of the LD operation in Rourkela is the adoption of a two-slag practice. The hot metal analyses:

> C = 4%, Si = 1·0 −1·3%, P = 0·25−0·35%, Mn = 0·8−1·1% and s = 0·04 − 0·08%.

In a 50 t converter nearly 48-49 t of hot metal and 6-8 t of scrap are charged when the converter is in a tilted position. The actual practice along with the details of additions, lance height, etc., are shown in Figure 16.1. The (*a*) part refers to the earlier practice, whereas the present practice is shown in (*b*). Blowing commences with oxygen at 11 atmosphere pressure. The second blowing is carried out at 8-9 atmosphere pressure and at lance height of 110 cm as against earlier 130 cm. After all the lime additions, the lance is further lowered to 90 cm and the blow is completed at 11 atmosphere pressure.

The lance is withdrawn, the vessel made horizontal, and slag is removed until the predetermined amount is left in. Samples of slag and metal are taken for test. The bath temperature is measured by an immersion pyrometer. A slag dam is formed to retain the slag in the vessel for the next heat. Dam formation consists of shoveling old, sized slag prices near the converter mouth, where it gets fritted to form the dam. Liquid steel is tapped by

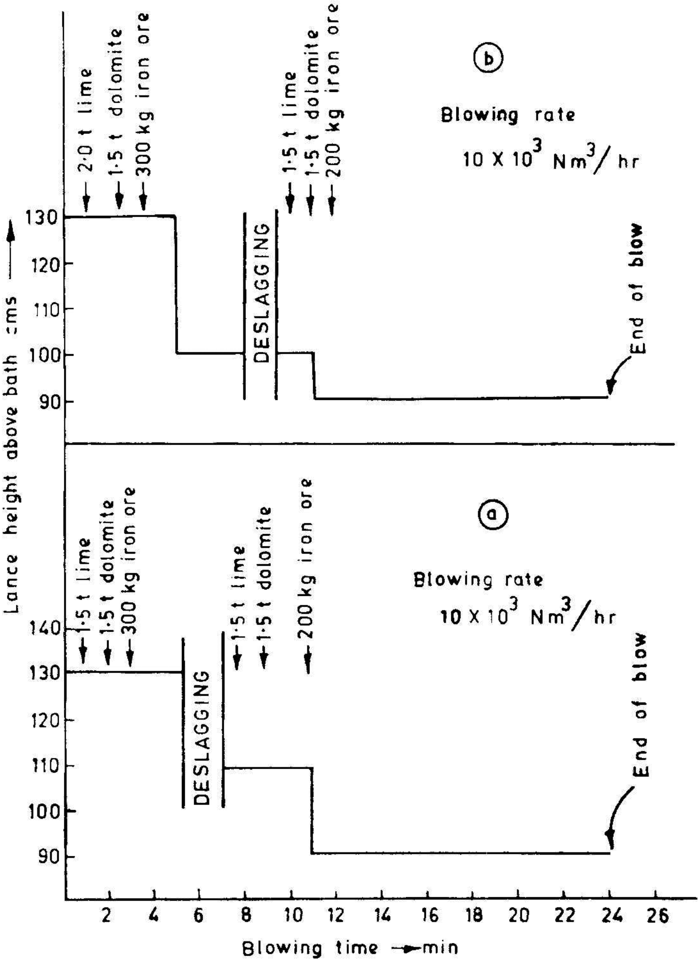


FIGURE 16.1. Details of LD blowing practice at the Rourkela Steel Plant.[23]

piercing a hole in the slag dam (no tap hole is provided on this converter), which holds back the slag. Although the double-slag practice is mostly used for high phosphoric irons, this practice is adopted at Rourkela because of the high silicon content of the hot metal, which is detrimental to the lining life of the converter.

The double-slag practice does decrease the production rate; however, it has advantages. The main portion of the blow is conducted with less slag volume of higher basicity, with a consequent benefit of reduction in sloping tendency, less wear on lining, and a better control (FeO content) of slag resulting in cleaner steel.

Details of blowing:

Blowing time -1st blow: 5 - 6 min.

- 2nd blow: 15 - 18 min.

Oxygen pressure: 9-11 atmos.

Oxygen flow rate: 10×10^3 Nm^3/hr

Oxygen consumption: 60 Nm^3/t of steel

Lance height: 90-130 cm

Bath temperature: 1600-1620°C

Slag volume: 150-170 kg/t

Rate of production: 50-60 t/hr.

Tap-to-tap time: 40-49 min.

Refractory consumption: 17 kg/t of steel produced.

16.3. Characteristics of LD Process

The LD process in its classical form or the modified bottom agitated form is a unique process; the way it functions and carries out the refining is very different from the earlier hearth or the pneumatic processes. In order to appreciate the process, the following phenomena need to be appreciated.

1. As the supersonic oxygen jet travels through the vessel gases, some ambient medium is carried along. This entrainment is much less in the supersonic core but is substantial in the sonic portion of the jet. The composition of the jet, at the point where it strikes the bath,

depends on the blowing rate, height of the lance, etc. It could even be only 60 % O_2, as has been estimated under certain conditions of soft blowing, and the remainder being practically CO and CO_2.

2. As the jet strikes the bath surface it results in stirring of the bath. It has been conclusively proved that the bath circulates radially outwards on the surface and upwards on the vertical axis as shown in Figure 15.5. This stirring, coupled with exothermic refining reactions, helps to dissolve the scrap, particularly in the early part of the blow. The rate of dissolution of scrap depends primarily upon its size, blowing conditions, vessel design, temperature, and composition of the hot metal, etc. Deeper baths are inferior to shallower baths in this respect because of lesser turbulence in a deeper bath, under identical conditions.
3. In spite of using a supersonic jet, the kinetic energy of the jet is not enough to cause enough mixing of the slag and metal inside the vessel; consequently the bath suffers from significant gradients in temperature and composition, which is somewhat responsible for periodic ejections and slopping, and thus loss of yield.
4. It has been found that in the short blow of 15-20 min all lime is not dissolved in slag, in spite of the slag being high in iron oxide content. High reactivity of lime is better in this respect.
5. The refining takes place by oxygen either directly by reaction with carbon, or through formation of iron oxide first and then iron oxide oxidizing the impurities in iron. Considering that the solubility of oxygen in iron is only around 0.25%, and all that required oxygen, roughly about 3 wt% of the bath weight, is to be dissolved in 15-20 min of the blow, much of the refining occurs via iron oxide.
6. Oxidation of iron is quite exothermic and hence generates a temperature of the order of 2000° C at the jet impact zone. It has been known as a *hot spot,* and the high refining rates of LD were once erroneously attributed to this high temperature. In fact, being in large bulk and having a large impact area of jet, the oxidation of iron is very fast and iron oxide is available in great abundance for further refining. In fact, the refining process is conditioned by the rate of supply of oxygen.
7. The sequence of elimination of impurities during the blow is shown in Figure 16.2. It reveals that the rate of silicon oxidation is very

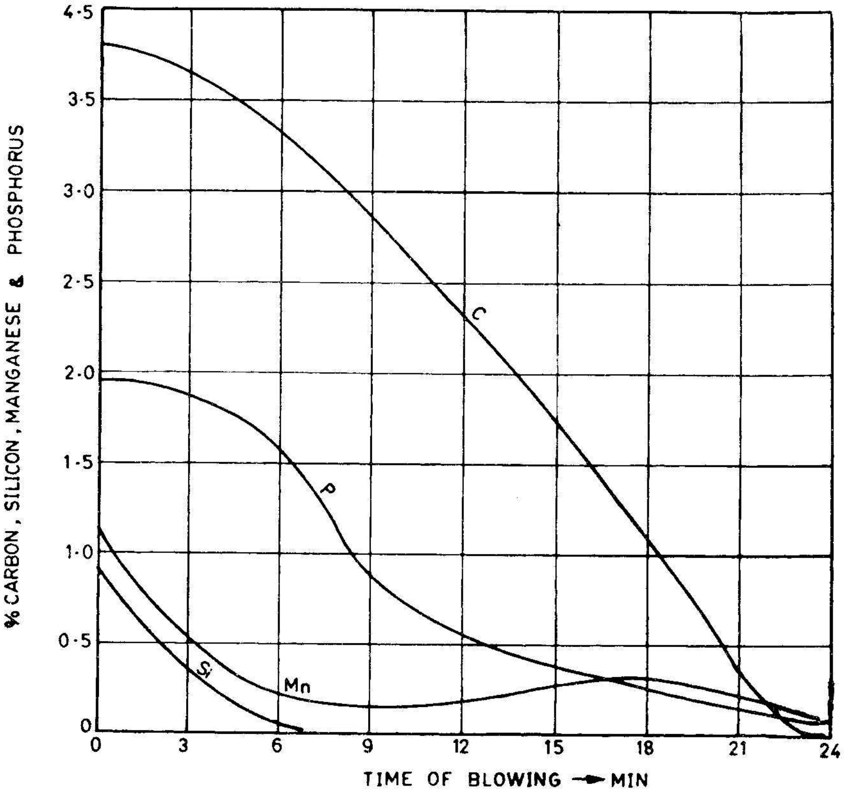


FIGURE 16.2. Sequence of elimination of impurities in an LD blow (schematic) (after Trenkler [23]).

fast, as expected. It is completed in about the first 3-5 min of blow. Manganese comes down concurrently with silicon up to a certain point and thereafter it may remain the same or revert to some extent towards the end of the blow. Carbon oxidation starts only after a few minutes of the blow and phosphorus is oxidized concurrently with carbon, but is completed well before carbon oxidation is over.

8. The high velocity jet tears off small droplets of metal as well as slag from the bulk, and throws them up in the vessel atmosphere where the iron oxide contained therein reacts with the carbon, generating CO gas. In a way, the jet action creates a situation akin to the formation of bubbles from soap solution when air is blown through it. The soap bubble is an emulsion wherein an immiscible thin soap solution layer covers the individual bubble. In LD there are two immiscible liquids such as metal and slag and these may cover the gas bubble.

In fact this is an emulsion of slag, metal, and gas. The stability of this emulsion is ensured from within by the CO gas formation owing to the iron oxide reacting with carbon in the metal droplets. The emulsion therefore is stable as long as the carbon reaction is going on. The initial high viscosity slags, before complete lime dissolution, are helpful for the formation of the foam. The interfacial area of contact of slag-metal-gas is thus tremendously increased.

It could amount to a total slag-metal interfacial area of nearly 100-200 m^2 /t of steel in reality. The metal droplets in the emulsion may have sizes of around 0.1 to 5.0 mm or equivalent area of 10-20 cm^2 /g of steel. The average residence time of the droplets in the emulsion may be around 1-2 min. The enormous refining rates of the order of 10-12 wt% C/hr, as observed in LD practice, are due to this increase in interfacial area of slag and steel, and not due to the hot spot formation. It is equivalent to a decarburization rate of nearly 0.25 % C per min at the peak time.

It has been estimated that the actual slag-metal interfacial area in a 100 t ID vessel may be around 7500 m^2, where the apparent bath area may be around 12-14 m^2 only.

9. It means that there are two distinct zones of refining in an LD vessel, viz. the reactions in the emulsion and in the bulk phase. The contribution of bulk refining, i.e., refining in the impact area is dominant at the beginning when emulsion is yet to form and towards the end when emulsion collapses. It is also believed that substantial decarburization does occur via refining of droplets directly by the jet gas.

As the emulsion builds up, the refining attains a dominant role. The bulk phase refining dominates again towards the end when emulsion collapses.

10. The early exothermic oxidation of Si and Mn, particularly in the emulsion, raises the temperature of the slag and helps dissolve lime.

As the volume fluidity of the slag increases, the volume of metal droplets in the emulsion increases and thereby the decarburization of the droplets, as well as the generation of CO from within, which stabilizes the emulsion from within. It is a case of *mutual compound acceleration,* i.e., the decarburization rate is increased due to increased emulsion formation, and increased emulsion formation increases the decarburization rate. As this continues, the rate of decarburization attains a limiting value with respect to the rate of supply of oxygen for refining, rate of mass transport of oxygen to the reaction interface. It has been seen in practice that the rate of decarburization increases with rate of oxygen blowing, other conditions remaining the same.

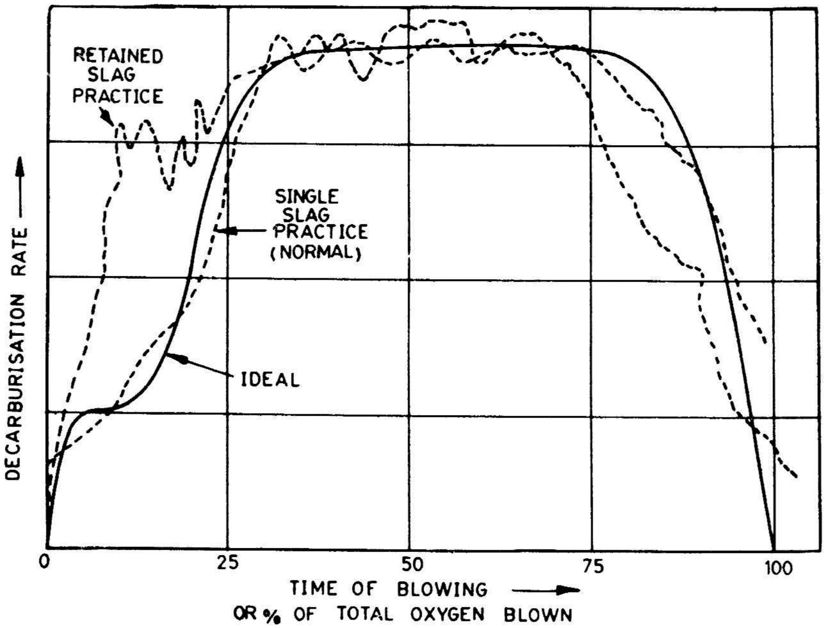


FIGURE 16.3. Decarburization rate as a function of time of blowing (after Mayer [28]).

11. The residence time of the droplets in the emulsion is estimated to be nearly 2-3 minutes halfway through the blow. It is slightly more in the beginning due to the high viscosity of the early slag, and is slightly less towards the end of refining since the slag is thin. The particle is thrown up and after its residence it falls back to the bath. The cycle is repeated as long as emulsion is stable. The droplets undergo refining as per its own impurity level, slag, and gas composition in contact with it. In the early part of the blow, therefore, silicon manganese is oxidized in some droplets, whereas in some others decarburization may have commenced.

The decarburization of droplets is delayed due to lack of supersaturation of dissolved oxygen necessary for the reaction. By and large, silicon manganeses are almost completely oxidized in the 4-6 min. of initial blowing. This period is marked by several other important happenings in the vessel. First, slag is formed due to the exothermic oxidation reactions and fluxing action of the oxide product, in particular silica. Second, the amount of metal droplets increases in the slag phase and attains a peak value. Third, the decarburization of droplets increases in increasing proportion as is evident from the buildup of the decarburization rate curve as shown in Figure 16.3. The ideal curve for decarburization rate is also shown in the same figure for the sake of comparison. The steady state rate of decarburization is matched by the rate of supply of oxygen. The necessity for the early formation of slag to expedite refining, therefore, hardly needs any more emphasis.

12. Toward the end of refining, the carbon concentration of the droplets in the emulsion approaches a value such that the rate of decarburization is no more controlled by the rate of supply of oxygen; instead it is controlled by the carbon content, which is low. The decarburization rate curve begins to fall since not enough CO is formed to sustain the emulsion from within. The drop in the decarburization rate towards the end is quite rapid since now it is a case of *mutual compound deceleration.* This *tail* of the decarburization rate curve is quite reproducible and forms the basis for *dynamic control* of the process in the production of soft steels.

13. Conditions for dephosphorization are that the slag should be basic, thin and oxidizing and, that the temperature should be low. Dephosphorization, therefore, does not take place efficiently until such a slag is formed. Such a slag is formed in LD process only after the initial 4-6 minutes of blowing. The rate of dephosphorization picks up concurrently with the rate of decarburization. For efficient decarburization as well as dephosphorization the slag should, therefore, form as early as possible in the process. If a preformed slag is present as in a double slag practice wherein the second slag is retained in the vessel in part or full, the decarburization rate curve rises more steeply in the beginning as shown in Figure 16.3. In an LD practice the conditions are adjusted to obtain early slag conducive to effective P removal.

14. Dephosphorization is very rapid in the emulsion because of the increased interfacial area and efficient mass transport. Phosphorus should, therefore, be fully eliminated before the emulsion collapses. If this is not achieved the heat will have to be kept waiting for dephosphorization to take place, and in the bulk phase, it is extremely slow as compared to that in the emulsion. In general dephosphorization should be over by the time carbon is down to 0.7-1.0%, i.e., well ahead of the collapse of emulsion which begins at around 0.3%C.

15. The relative rates of dephosphorization and decarburization, when the latter is at its steady state, are controlled to achieve the desired objective. This can be simply done in a given practice by adjusting the lance height or by adjusting the flow rate of oxygen. Raising the height of the lance or decreasing the oxygen pressure decreases the gas-metal reactions in the emulsion (i.e., decarburization) and vice versa. The dephosphorization reaction is thus relatively increased by the above change and vice versa. Towards the end when temperature is high the danger of phosphorus reversion does exist but it can be prevented by maintaining a high basicity of the slag.

16. The blowing strategy is decided in advance to obtain effective dephosphorization well ahead of decarburization. Some operators prefer to alter the height of the lance while maintaining the oxygen flow rate constant. Others keep the lance height constant during the entire blow but vary the oxygen pressure to obtain the desired conditions of blowing.

17. The decarburization paths in practice have been observed to be different even with similar blowing practices. The physical distribution of scrap inside the vessel, in its effect to cause baffles and delay the ignition, may be responsible for such observed variations. But it is believed that even a minor change in the chemistry of the process leads to a major difference in the refining paths.

18. As in a basic open-hearth, a basic and thin slag coupled with large slag-metal interfacial area in the emulsion favors desulfurization to the extent of 40-50% of the total sulfur in the metal at the level of 0.04% S. Desulfurization is definitely favored by the increased (FeO) content of the slag.

19. Blowing conditions have a major effect on manganese distribution. The initial rate of oxidation of manganese is around 0.25% per minute but as the (FeO) content of the slag builds up towards the end it is reverted at the rate of around 0.03% per minute. Its content decides the oxygen level of the bath at the end, and as such is of economic importance.

20. The efficiency of refining in the LD process primarily depends on how rapidly a thin oxidizing and highly basic slag is formed for dephosphorization and, possibly desulfurization, to start. This in turn depends on how quickly lime, the flux, is dissolved to form the required slag. Initial iron oxide and silica formed have to assimilate lime to form the right type of slag. This was not the problem in the earlier slow process of steel making like open-hearth. In the faster LD process however, this became a crucial problem. The mechanism and kinetics of lime dissolution in basically a FeO-SiO_2 type slag initially, and in FeO-SiO_2-CaO type ternary slag and high order slag, later is of interest. It is known that lime dissolution is a mass transfer controlled process. It essentially would be solid(CaO)-liquid(FeO-SiO_2) interfacial phenomena.

Porous lime particles are surely better in this respect. The formation of a solid dicalcium-silicate layer initially in this process can create problems. FeO and MnO are good solvents for this solid layer. The slag composition in LD continuously changes in the initial part of the blow. With all the care LD slag does contain solid undissolved lime particles to the extent of 5-10% of slag weight.

16.4. Process Control

The process is always designed for a certain reference, local conditions with respect to the composition and amount of charge materials (input variables) and, operating variables like lance height, flow rate, pressure of oxygen, etc. The dimensional changes in the furnace lining and the lance nozzle due to the usage should also be taken into account by the operator. Although care is taken to adhere to the input variables, the parameters do change and the process needs to be controlled by changing the operating variables suitably. The term *process control* means the ability of adjusting the operating variables in order to obtain the desired *turn-down conditions* (desired composition and temperature on the bath) despite the variations in the quality and quantity of the charge materials.

The conventional processes like open-hearth and electric furnaces were slow enough to carry out analyses of slag and metal during refining, adjust the slag composition and, thereby finally obtain the desired product. The LD process is too fast to adopt similar leisurely control strategies. Traditionally the LD process control has much relied on the experience gained from blows of nominally similar charge compositions to produce steels of identical specifications. The best working conditions are established more by trial and error. This procedure often results in off-temperature heats which lower the productivity, since corrective measures such as reblowing to increase the temperature or adding coolants to decrease the temperature have to be adopted. It moreover deteriorates the furnace lining.

The process control of LD is, therefore, of utmost significance because in the absence of such a control much of the superiority of extremely fast rates of refining in a LD process, as well as process economy in refractory consumption, will be lost if the desired turndown conditions are not reached when the blow is stopped.

The control problem is becoming more and more acute as the specifications are getting more and more stringent when wide variations in raw material characteristics are to be tolerated. A positive and predictive control, which is flexible and versatile enough for various raw materials, operational characteristics and desired end product is prime requisite. Initially static control, based on thermal and material balancing of the charge, prior to the blowing, was developed to attain desired turn-down conditions. This is a predictive type of model and using mainly off-line computer decides the optimum weights of the materials to be charged and the amount of oxygen to be blown to obtain desired end product. It may include, the prediction of a reblow or cooling additions, if any, required at the end. Although it improves the control, it ignores the dynamic features of the process that require an ability to sense, judge, and initiate control action within seconds.

The operation is, therefore, increasingly being controlled automatically to improve upon the production rate by reducing the percentage off-grade heats. A simplified scheme is shown in Figure 16.4. Several mathematical models have been developed to simulate the process to work out the computer control program.

There is one redeeming feature of this process, which allows control of refining without any catastrophic consequences, even if the process variables are not very accurately adjusted to the process requirements. If the lance height is less than the correct value, more metal is thrown in the

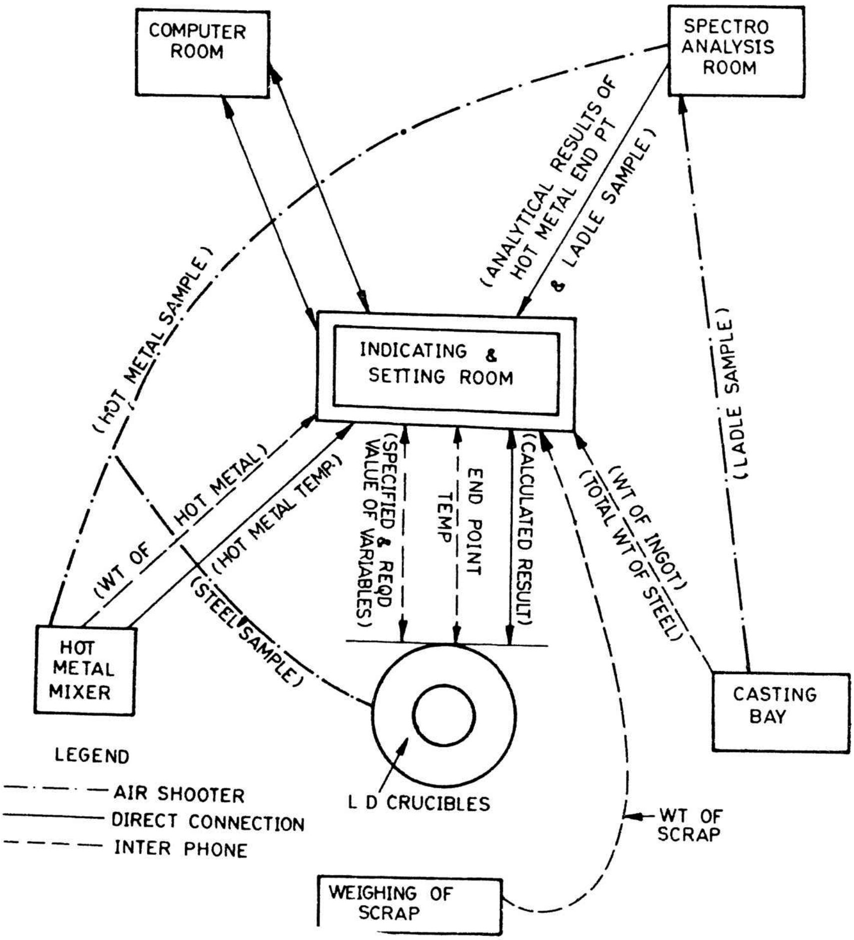


FIGURE 16.4. Simplified scheme of LD process control using computers and automatic analyzers.

emulsion and thereby, the true height tends to attain the correct value; the converse is also true. The same will be observed if the pressure of oxygen varies at any fixed position of the lance. This is, no doubt, possible only within certain narrow limits. This built-in and self-correcting balance allows variations around the standard state without affecting refining adversely, thereby make the process control of otherwise a dynamic process a relatively less difficult job.

16.5. Process Economies

The following points, besides the production rate factor, need to be looked into, to improve upon the process economics:

1. Ejections from the vessel, i.e., sloping should be kept to a minimum. It not only leads to loss of metal but also affects smooth operation. It is caused by lower lance position during blowing for long. Build up of (FeO) in the slag when carbon is high causes violent and intermittent reaction resulting in sloping. If Si and Mn contents of the bath are high, if ore and spar additions are more than necessary, if vessel is over-charged and all such factors contribute to sloping. It can be minimized by early formation of a proper slag.
2. Lowering the lance too close to the bath cause metal particles to be ejected from the mouth in the form of sparks. These ejections can damage the lance. Early formation of slag is beneficial in minimizing sparking.
3. Being a basic process, more silicon content in the metal leads to increase in the slag bulk which affects the lining life adversely. A very high silicon content needs double slag practice. Similarly if phosphorus is more double slag practice will have to be adopted. Such double slag practices increase the heat time, decrease the production rate and decrease the yield.
4. Spar or bauxite should be used sparingly to improve lining life. Its use adds to the cost of steel making.
5. The lance should be absolutely vertical in blowing position. Inclination on any one side causes the gases to escape in some preferential direction resulting in excessive erosion of lining on that side.
6. The scrap pieces should remain fully submerged in the bath or else it may deflect the jet in preferential direction causing unnecessary heating and wear of lining.
7. If scrap is wet or greasy it should be held in the vessel for a while, before charging the hot metal, to drive-off the moisture, otherwise the product may not be of required quality with respect to its hydrogen content.

8. Blast furnace slag should not flow in along with the metal since it will alter the basicity of refining slag. It complicates the weighing of metal charges.
9. The ejections do form a skull at the mouth of the vessel. It may protect the lining but excessive skull formation reduces the mouth area. If it is mechanically removed it may cause erosion of lining at the mouth. It is generally cut into two rings by a flame so that the lower part falls in due to its own weight and the outer one is gently removed.
10. Extra superheat at turndown unnecessarily affects lining life adversely.
11. Surplus oxygen supply may burn CO to CO_2 at the mouth and cause lining wear.
12. Delay in tapping a heat contributes to lining wear. This is particularly serious in the production of soft steels when final (FeO) content of the slag is high [see Section (4.1)].
13. Extra holding time of a heat due to other delays reduces lining life.

LD lining wear should be studied to improve the lining design. A balanced lining wears uniformly.

16.6. Output

The blowing times in LD vessels are kept nearly the same irrespective of the sizes of the vessels. The tap-to-tap time, therefore, increases with vessel size due to the increased time of charging, tapping, etc. One of the fastest vessels of 150 t capacity has a tap-to-tap time of just 30 minutes in which lancing time is 16 minutes. The lancing time can still be decreased but then there is not enough time for slag formation and, adequate dephosphorization and desulfurization may not take place before carbon is brought down to the required level. A two vessel plant of 150 t capacity each, with one working at a time, produces 2.5 Mt of steel per annum. Although these are a bit exceptional values, production of 2.25 Mt from a 200 t vessel and 3.0 Mt from 300 t vessel in a two vessel shop are considered to be good performances. The production rates obtainable under various conditions are shown in Figure 16.5.

For expansion of an existing shop some prefer to add a third vessel in the existing shop so that two work at any time out of three. This can increase the production rate by nearly 80%. Production rate has been increased by 60% by working both the vessels in a two vessel shop on week days and carrying repairs of one over the week ends. The organization of the shop in this case is rather difficult.

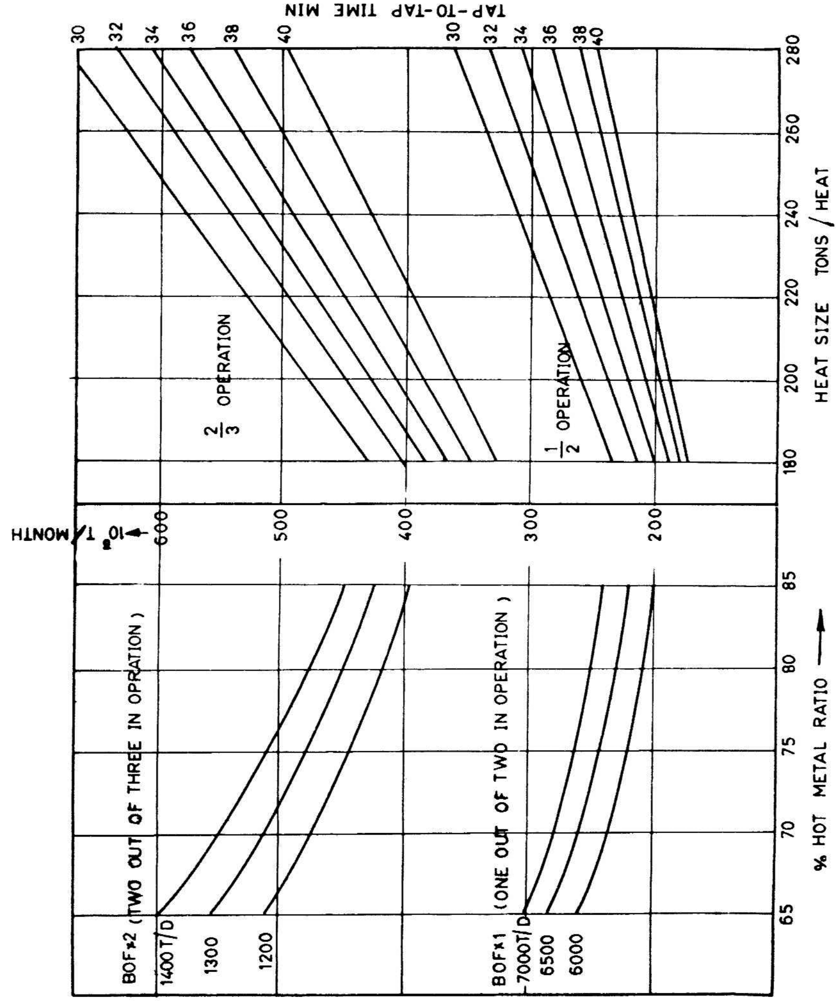


FIGURE 16.5. Production rates of LD shops as a function of number of vessels operating at a time, percentage hot metal in the charge, vessel size and tap-to-tap time. The output per day and per month is indicated on the abscissa (after Yamaguchi and Takahashi [29]).

For expansion of an existing two-vessel plant some operators strongly advocate the installation of an additional two-vessel plant instead of adding a third vessel to the existing plant. The following table illustrates the output in Indian plants:

Name of plant		*Vessel size t*	*Number of vessels*	*Output Mt/year*
Rourkela	SMS 1	45/50	3	1.0
	SMS 2	60/66	2	0.8
Bokaro	SMS 1	100	2	1.7 first stage
	SMS 2	250	3	2.3 expansion
Total Steel	SMS 1	130	2	1.5
	SMS 2	130	2	1.5

16.7. Slag Characteristics

The slag is under very careful watch during refining in an open heath and electric furnaces. It is periodically tested to control refining. The situation is different in LD process. The slag in a LD process is rarely tested during the blow and the final analysis, if carried out, is more for reference or checking. In general the basicity is on the high side in most LD plants. Excess lime prolongs lining life and ensures complete dephosphorization and maximum desulfurization without the danger of their reversion. The slag emulsion all around the *hot spot* protects the lining from radiations from this spot. It, therefore, improves thermal efficiency of the process unlike in an open hearth practice.

The slag composition in a LD vessel varies continuously during the blow. A typical set of basicity data in LD steel making at the Bokaro Steel Plant are shown below:

Percentage of blowing time	Basicity of slag
0 (mixer slag)	0.65
12	1.14
25	1.20
37.5	1.23
50	1.70
100	3.70

A typical composition of slag from a heat of dead soft steel at the time of tapping is as follow:

CaO.......... 47-52%
MgO......... 2-5%
P_2O_5.......... 1-3%
SiO_2.......... 10-16%
MnO........ 3-5%
Fe............ 16-22%

The mass of slag formed in any steel making heat is normally not weighed to know its actual eight. It can however be estimated in the post-mortem of the heat by knowing the slag analysis and the actual lime and/or dolomite added in the heat. The SiO_2 that is formed and that joins the slag comes from the total Si contained in the charge and the SiO_2 present in the flux. It is empirically found as:

$$\text{Mass of slag formed kg/t of steel} = \frac{\text{Total \% of CaO} + \text{MgO} + \text{SiO}_2}{90 - (16.6\ \text{Basicity} - 30)}$$

Normally the CaO or CaO plus MgO content of BOF slag is adjusted to around 45-50% of the slag. It means mass of slag generated is approximately double the value of flux added as actual weight of the base oxides. In other words empirically

$$\text{Mass of slag generated/t steel} = [\{\text{kg of CaO added}\}/\text{Charge wt}\ ] \times (2 \times 95)/100$$

Wherein the 95/100 factor is to take care of other oxides present in the flux and other errors involved, if any.

The iron content of the slag is on the lower side if high carbon steel is made. If phosphorus in the charge is more intermediate slagging off is carried out at nearly two-thirds of the way through the blow. The first slag in this case is less limey and oxidizing and, the second slag much more basic and oxidizing as compared to that in the single slag practice. A part of the second slag, if made, is generally retained in the vessel for the next heat.

16.8. Yield

The metallic yield is often around 90%. Ingot yield of 87-88% is considered to be a good practice. The Japanese plants have achieved ingot yield of 91-7% which is due to the relatively low metalloid content in their hot metal. The yields under Indian conditions are:

Rourkela	...	85-86%
Mysore	...	86-87%
Bokaro	...	86-87%

More of phosphorus and silicon contents in the hot metal generate large slag volume and the yield decrease due to increased loss of iron in the slag. Production of dead soft steel involves high iron oxide content of the slag at tap than when high carbon steels are made and hence the yield is low if soft steels are produced. Excess of lime charge also results in a high iron loss in slag.

16.9. Product Characteristics

LD process was earlier meant to produce only soft steels (C]>0.15%) but now it has successfully been used to make even high carbon and alloy steels including stainless steels. However the dead soft steels, for the production of flat products, is still the major production from all LD plants. LD steels are characterized by their low phosphorus, nitrogen and carbon contents. Phosphorus can be reduced from 0.15-0.20 to 0.012-0.017% is one slag operation. With a two slag practice it can be further reduced but at the expense of some metallic yield.

Nitrogen content of the order of 0.002% is readily attained to finally have only 0.003% in the ingots.

Carbon can be reduced to 0.04%, which is not readily possible in an open hearth practice.

The contents of residual elements are very low in LD steels and that the steels are in general cleaner than BOH steels.

CHAPTER 17

FURTHER DEVELOPMENTS IN LD PROCESS

In This Chapter

The LD process was originally designed to refine basic pig iron containing less than 0.4% P, to consume around 25% of the metal charge as scrap as coolant, and to produce soft steels. Once it was established that even the most modern open-hearth shops could not compete with LD, both in respect to capital and operating costs, the growth of LD has been phenomenal. As a result of wide applications, the process has also undergone improvements in various ways. The vessel size has increased from a mere 30 to 400 t with 500 t vessels being planned. The vessel shape underwent changes as shown in Figure 17.1. The trend in LD configuration is

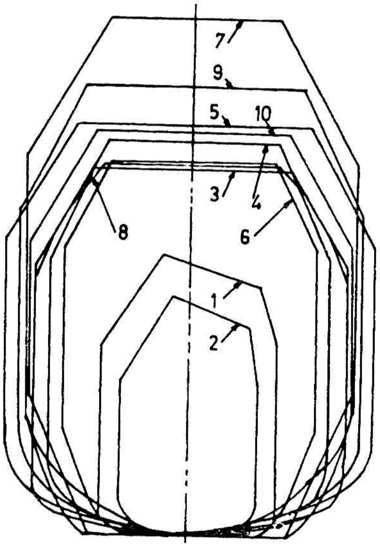


FIGURE 17.1. Various LD vessel shapes [30].

towards a larger specific volume of about one cubic meter per metric ton and a higher height-to-diameter ratio of about 1.5, as would be obvious from the table below[21]:

Plant	*Capacity t*	*Specific volume*	*Height / dia ratio*
Ravenscraig (U.K.)	130	1.0	-
Ohgishima (N.K. Japan)	250	0.92	1.54
Fukuyama (Japan)	250	0.75	1.45
Oita (Nippon Steel Japan)	300	1.26	1.53
TISCO	130	1.1	1.5

The comparable figures for the OBM process at Thy-Marcinelle for a 150 t converter are 0.91 cubic meters per metric ton as specific volume and the height-to-diameter ratio of 1.2 only.

The single-hole nozzle lances have universally been replaced by multi-hole lances (3-7 holes) with attendant cut in blowing time, increased lining life, and improved yield. An LD shop with only one operating vessel at a time can now produce 4-5 Mt of steel annually. The average lining life of few hundred heats has been increased to nearly 1000 heats, and in some exceptional cases even up to 2000 heats. This has been possible through the use of better refractories and improved operation. The operation has been automated nearly fully to the extent that carbon level of the bath and its temperature are continuously estimated and controlled to obtain correct turndown conditions.

	Company	*Vessel capacity metric tons*	*Operation date*
1.	McLouth	60	1954
2.	Rourkela	40	1960
3.	Armco	150	1963
4.	Wisconsin	120	1964
5.	Great Lakes	300	1971
6.	Bokaro	100	1973
7.	Ohgishima	250	1976
8.	Thy-Marcenille	150	1976
9.	Bokaro	300	1981
10.	TISCO	130	1983 & 1996

The process has been suitably modified to deal with Thomas iron and is known as the LDAC or OLP process. The original limitations regarding the chemistry of the product no longer exist. High carbon and alloy steels, and even stainless steels with the required cleanliness can now be successfully made in LD vessels. The thermal input has been improved to take much higher proportion of scrap in the charge and thereby compete with open-hearth even in this respect. The LD gas handling system underwent some notable changes to recover fully or partially the CO rich exit gas, in order to use it as a fuel elsewhere in the plant. Some of these developments are described in detail below.

17.1. LDAC or OLP Process

The continental countries like Germany, France, Belgium, Luxembourg, the UK, etc., produce Thomas grade iron. The LD process was, therefore, modified by the CNRM in Belgium to refine Thomas iron and was put into commercial practice at the ARBED-dulange works in Luxembourg. The process is known as the LDAC process. Fundamentally the

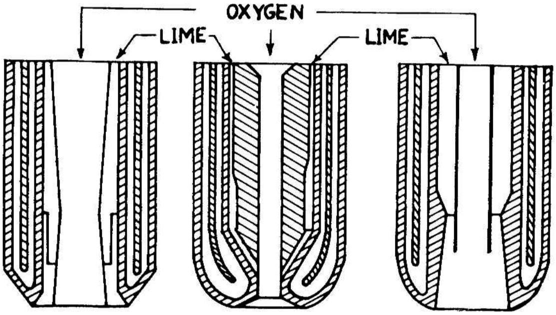


FIGURE 17.2. Oxygen lance designs to introduce lime powder along with oxygen.

dephosphorization rate in the LDAC process is increased such that the dephosphorization of high phosphorus iron is over well ahead of decarburization of the bath, as is possible in a standard LD process. This was possible by introducing a major portion of the line charge as powder through the lance along with oxygen. These lance designs are shown in Figure 17.2. Operating experience led to the necessary changes in the design and operation of the vessel.

Basically a process is called LDAC if iron containing more than 0.4% P is refined to produce at least one intermediate phosphoric slag that is removed from the vessel, and the heat is finished by further additions of lime and oxygen blowing.

There is a lot of similarity between LDAC and LD plants in design and operation. The vessel shape is however altered to a tulip shape with two tap holes at diametrically opposite positions. The vessel volume is greater by nearly 20% over that of LD to accommodate extra slag volume generated in the process.

Thomas iron may have composition like

Carbon	3.2-3.5%
Phosphorus	1.5-2.0%
Manganese	0.7-1.0%
Silicon	0.4-0.6%
Sulfur	0.05% or so

The LDAC charge contains less scrap than that in the LD process because a large excess of lime is to be dissolved to form slag to accommodate the extra percentage of phosphorus in this process. Iron oxide and dolomite are incorporated in the charge to help dissolve lumpy lime in the initial charge.

The hot vessel containing some slag from the previous heat is charged with scrap, ore, bauxite, hot metal, and one-third of the total lime required for the heat. If silicon is low some silica may also be charged to form foamy slag quickly. Blowing commences with the lance at its highest position. Powdered lime (1-2 mm size) is added through the lance at a predetermined rate after five minutes of oxygen blowing from the high position. The lance is then carefully lowered to control the foam. The blow is stopped after 15 minutes, by which time carbon is brought down to nearly 1.5-1.7 % and phosphorus to 0.2%, the bath temperature being just over 1600°C. The slag is then drained out as completely as possible. Fresh additions of coolants, fluxes, etc., are made and the vessel is blown again for 5-8 minutes. At the end of refining, sample is taken and is analyzed. The temperature is measured as usual and the heat is tapped, leaving some or all of the slag in the vessel for the next heat.

It has already been discussed in earlier chapters that dephosphorization is favored by raising the lance since it hastens foam formation. Decarburization is favored by lowering the lance. All the lime required for dephosphorization in this case cannot be charged in the vessel because it would lead to a large slag volume and increased attendant troubles. Intermediate slagging-off is a must. Addition of lime powder from the lance directly at the hot spot, after the formation of initial foam with lumpy lime, hastens the formation of foam and thereby the dephosphorization rate as well. This is the key to the success of the process. The operating experience has proved that for a given weight of lime added, the final phosphorus content in the bath will be lower in the case of lime fed through the lance than can be achieved by lumpy lime addition, as in standard LD practice. The proportion of lime, of the total fed through the lance, therefore increases as the phosphorus content of hot metal increases beyond 0.4% P below, which standard LD practice is adequate.

The intermediate slag contains:

CaO	45-50%
P_2O_5	18-20%
SiO_2	5-8%
Fe	10-11%

MnO and MgO a few percent each.

This can be sold as a fertilizer, hence spar should not be used as a flux. Excess silicon in the charge reduces the P2O5 content of the slag. The final slag may contain CaO = 50%, SiO = 2-4%, P2O5 = 10%, Fe = 20 - 25% and MnO and MgO a few percent each, which is highly basic and oxidizing. It is too useful to be thrown away, and hence is retained in the vessel for the next operation.

In this process phosphorus is reduced down to 0.02% in the metal. If extra low-phos-steel is required the process is modified to remove two intermediate slags. The first is fertilizer grade, the second to be thrown away, and the third is retained for the next heat. The tap-to-tap time is around 55-60 minutes. The yield is somewhat less than that in the LD process.

The process was earlier designed to produce dead soft steels but now up to 0.8% C steels are being produced successfully. The process became popular in spite of a little lower production rate and higher refractory consumption as compared to the LD process.

The OLP process (oxygen lime process) is quite similar in principle and practice to LDAC. It was independently developed by the IRSID in France. If the IRSID design is adopted, it is known as OLP, or else it is popularly known as the LDAC process.

17.2. High Carbon and Alloy Steel Making

In spite of being very fast in refining, the LD process has been developed to produce high carbon and alloy steels with necessary chemical and metallurgical control, as is possible in open-hearth and electric furnaces. The field still continues to expand to make more and more types of these steels by the LD process.

17.2.1. Carbon Steels

High carbon steels can be made by recarburization of dead soft steels, provided the level of phosphorus and sulfur are sufficiently low. The method of recarburization may be good enough to produce good quality low carbon steels, but it suffers from various disadvantages such as:

(*i*) The nitrogen level increases in proportion to the recarburizer additions.

(*ii*) More oxygen is unnecessarily blown to first produce soft steels.

(*iii*) Production of soft steels increases lining wear.

(*iv*) The recarburized steels are inferior in terms of cleanliness.

(*v*) For recarburization, a higher tapping temperature is necessary to dissolve the additions, which adds to several operational and economical problems.

High carbon steels are produced in the LD process by what is known as the catch carbon technique, in which the blow is adjusted in such a way that dephosphorization is complete well ahead of decarburization of the bath. At the turn-down, the bath has nearly the required carbon, and phosphorus is below the usual level. The practice is not only free of the above-mentioned troubles, but is cheaper and the product is definitely of better quality, particularly in terms of cleanliness. The practice does involve extra consumption of lime and spar to expedite dephosphorization. It consumes less scrap because it is not blown fully.

The heat has to be kept waiting, in this practice, until the analysis is obtained from the laboratory. This decreases the productivity of the shop and has a detrimental effect on lining life. The lance height is increased and soft blowing conditions are maintained to achieve faster dephosphorization and decarburization. Oxygen-borne lime feedings, as in the LDAC process, are recommended even while using standard LD iron to expedite dephosphorization and catch the bath carbon content at the correct point in producing medium and high carbon steels.

In another modification, heat is blown to dead soft level and a known amount of molten pig iron is mixed with the blown metal in the ladle to make up for carbon. It is possible only if proper quality of pig iron is available.

17.2.2. Alloy Steel Making

Some of the low alloy steels of the type such as case hardening, chromium-molybdenum, ball-bearing, dynamo-sheet grade, and so on containing a percent or so of common alloying elements like Cr, Mo, Ni, Mn, etc., are now regularly made in LD plants. Alloying additions are made in the ladle while tapping. The additions may be made in cold conditions, preheated or even in premelted form depending upon the total alloying additions since it should not have a chilling effect on the metal in the ladle. Non-oxidizable elements like Ni and Mo can be added to the vessel itself. Even reducing slag has been used in LD as is possible in an electric arc furnace.

Hot blast cupola is used to melt pig iron alloy scrap to produce hot alloyed pig iron, which is added to the ladle during tapping of the heat from the LD vessel. This method has proved to be more economical than the usual electric arc furnace practice. The main advantage of this practice lies in the production of alloy steels with lower nitrogen content than in electric or open-hearth furnace practices.

Use of vacuum degassing for these alloy steels produces a better quality product (see Section 28.1).

17.3. Stainless Steel Making

Generally stainless steel is made in an LD vessel in which blowing is carried out mainly in three stages.

1. Dephosphorization of the basic hot metal charge should be achieved first. For this the lance is initially held in the usual position to form foam. It is then raised and the oxygen flow rate is reduced to form thin, basic and highly oxidizing slag to hasten dephosphorization in preference to decarburization. The bath temperature is not allowed to rise beyond nearly 1450°C to help dephosphorization. Slag basicity is maintained at around 3. The blow is over in 8-12 min by which time phosphorus is reduced to blow 0.02% and carbon to nearly 2%. The slag is removed as completely as possible.
2. After the removal of phosphoric slag alloy scrap, non-oxidizable alloying additions and flux are added. Some amount of Fe-Si may be added to produce heat and promote slag formation. The second blow is shorter than the first. The bath is fully decarburized and its temperature is raised to about 1650°C. The second slag is also removed fully. Its basicity is 2-3.
3. High carbon ferro-chrome, flux, chromite ore, etc., are added and the vessel is blown for a very short duration to produce slag of basicity 2-2.5; Fe-Si may be added as a heat producer. Oxidation of silicon from the bath prevents over-oxidation of bath-chromium. In the presence of silicon and chromium, the bath has to be killed by Al, Ti, Ca, or Mg to obtain satisfactory deoxidation. Addition of chrome ore to slag helps to prevent over-oxidation of chromium from the bath.

Minor composition adjustments can be made in the ladle while tapping if sampling and analysis has been carried out just before tapping. It may be vacuum treated to further improve its quality and cleanliness.

Sulfur content can be reduced to 0.005% by employing a reducing slag, as in electric steel melting. An external heat source may be needed to melt this slag. The bath, however, tends to pick up carbon during this period.

It is proposed to smelt chromium-bearing iron in a blast furnace to be later blown in an LD vessel to produce stainless steels.

17.3.1. LAM Process

A partial modification of the above process has been worked out by the Jones and Laughlin Co. in the United States and is known as the LAM (LD Alloy Steel making) process. In this process, after removal of phosphoric slag, a lot of chrome ore is charged in the vessel, and is melted to form slag. Once the carbon level of the bath is lower than the required value, the metal and the slag are poured in an electric furnace where chrome silicide is used to reduce the chromium level in the slag to a very low value. Heats of 45 t have been successfully made.

17.4. High Percentage Scrap Melting Techniques

In industrialized countries, the LD process is required to take more than the usual (about 25%) scrap in the charge. Although there is not much of a provision to increase the thermal efficiency of the process, the thermal energy input has been increased in various ways to achieve this objective.

(1) Steel scrap is heated in the vessel for a few minutes prior to the hot metal charging using an oxy-fuel burner. The heating time is nearly 10 minutes and it saves in terms of blowing time. By this method 40-50% scrap has been successfully included in the charge. It is believed that this practice is better than electric arc furnace melting.

(2) Scrap can be preheated outside the vessel just before charging but the handling problems are yet to be solved.

(3) Charging of lime at around 1000°C directly from the kiln has increased scrap proportion by 2-3%. Hot lime also hastens early slag formation and decreases spar consumption.

(4) Calcium carbide (78% carbide and 22% lime) has been used in place of lime as a flux. The heat evolved per metric ton of this material is sufficient to melt 7 t of scrap. Scrap proportions up to 49% have thus been successfully used in commercial vessels.

(5) At times, coke can be used as a part of the charge to increase input thermal energy. The LD process has also been modified to use an entirely cold charge. The charge is melted by an oxy-fuel burner. Alternatively coke in the charge is burned by oxygen from the lance to melt the charge. Such processes have yet to become economical competitors to other processes.

Extra scrap charge in an LD vessel should also be viewed from the point of view of its quality. Dirty, rusty, and greasy scrap may impair quality of the final product.

The process has been modified to increase scrap consumption by burning part of the CO inside the LD vessel with the help of an additional oxygen lance. The vessel needs to be rotated, with its axis vertical. The process is known as the *Rotovert* process and can increase scrap consumption by 18%. It also helps alloy additions that are dissolved rapidly in the bath in a rotating vessel, allowing extremely low sulfur levels to be achieved.

17.5. OG Process

In a normal LD process, the exhaust gases from the converter contain 75-80% CO, which is entirely burned in the hood by allowing air to infiltrate in. The heat of CO to CO2 reaction practically goes to waste, if not utilized for steam rising in waste heat boilers. A new process now known as the OG process (oxygen converter gas recovery process) has been developed in Japan to collect and store the gas as is done in a blast furnace, for its subsequent use as a fuel. The overall plant economy has been improved with a marginal increase in the capital outlay. The scheme as developed and installed by Nippon Steel at their Kimitsu works is shown in Figure 17.4. The gap between the hood and the vessel nose is closed by a skirt to prevent any infiltration of air. The gases are cooled and cleaned by methods described earlier and are stored in a gas holder made of steel.

The carbon monoxide-rich gases from the vessel during the LD process refining generally amount to 60-110 Nm^3 per metric ton of steel refined. The calorific value of such a gas may range between 1800 to

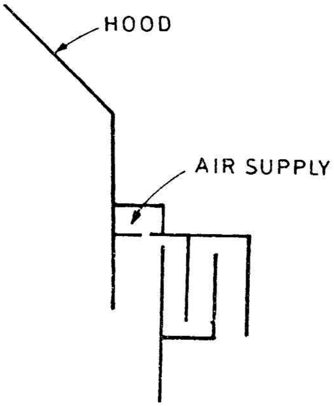


FIGURE 17.3. Suggested air curtain seal.

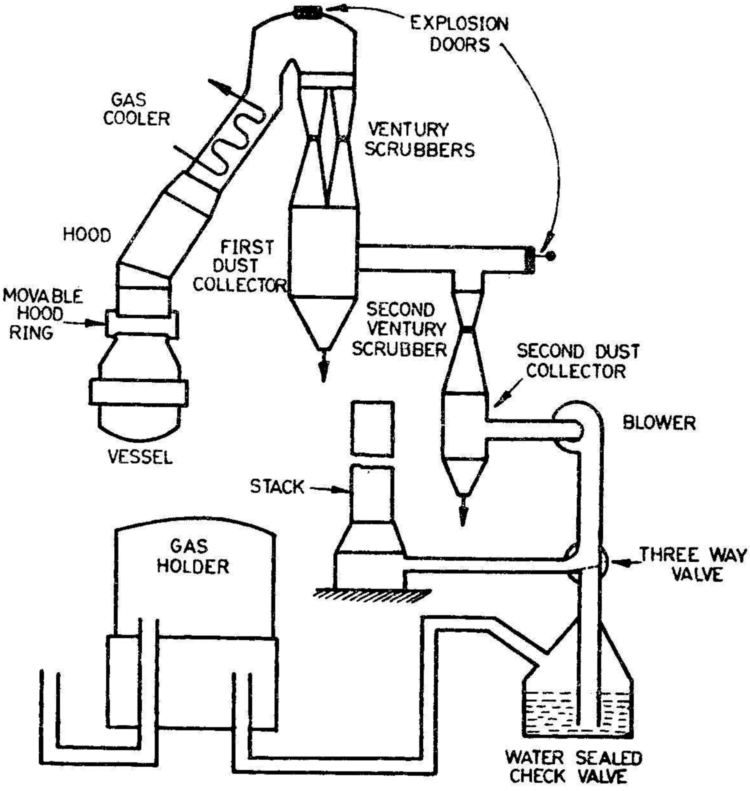


FIGURE 17.4. Hood and gas cleaning arrangement in OG process.

2000 kcal per Nm^3. From a 100 t LD vessel therefore nearly 2×10^7 kcal thermal energy is generally recovered through such a gas recovery system. This is a huge amount of energy and can be economically used elsewhere in the steel plant. This has been universally accepted and is being adopted on top and bottom, and hybrid processes in increasing number the world over.

In order to obtain clearly the gas only and minimize the air infiltration into the hood, several seals have been developed. One such seal is shown in Figure 17.3.

17.6. LD-CL Process

The chief drawback of the LD process is the stratification of the metal bath towards the end of refining finally resulting in off-grade heats and segregation. This is due to the inherent shortcomings in carrying out the process with fixed position of lancing from the top. An important development for improving LD steel making is the use of the circling lance, developed and perfected by the Nippon Kokan of Japan. In this, the top end of the lance is anchored, but during the blow the lower tip moves through a circle at around 0.1-5.0 rpm with an eccentricity of up to 40% of the distance from the furnace center to the furnace wall. The first commercial application was on a 250 t LD converter at the Ohgishima plant[24].

The test results showed that the iron yield was increased by 0.5% compared to the fixed lance, because of a lower loss of iron in slag. The loss of manganese in slag was decreased and thereby the manganese recovery was improved marginally. The iron loss in slag in blowing 0.1% C steel was reduced by about 2-5%.

17.7. Sub-Lance LD Process

A sub-lance is used to introduce a device in the bath inside the vessel to measure bath temperature and carbon in-situ and obtain a steel sample for further analysis. It can also measure the level of the bath before blowing.

Use of a sub-lance in LD enables the operator to carry out all the measurements while the vessel is being blown in the vertical position, and obviates the necessity of interruption, as in the manual method, which may mean loss of as much as 5-10% in tap-to-tap time.

The use of a sub-lance also permits dynamic control of the blowing operations. The major advantages claimed by using a sub-lance are a significant increase in the hit-rate of end point temperature and carbon content, reduction in the number of heats reblown or cooled, and consequent improvement in productivity, yield, and refractory consumption. These advantages have made the sub-lance an essential item of equipment for achieving high efficiency of the LD process.

A number of LD shops have already installed this system, and it is expected that this technique will be increasingly adopted[20].

17.8. Freon Turbine

This is yet another improvement in energy saving/energy recovery from the converter gas via the cooling water, which is used to cool the offtake gas. Electrical energy is produced in a freon turbine generator using the heat recovered from the LD process cooling water. A full-scale installation commenced operation in 1979 on 250 t LD converters at the Kashima Steel Works in Japan[30]. The output claimed is of the order of 3MW.

17.9. Automation and Process Control

The LD process is so fast that a loss of even a minute may mean a loss of 2 to 3% of the production. A saving of seconds is even well worth trying. A saving in heat time can be attempted by way of optimizing the material handling system to avoid delays, and by way of controlling the refining to arrive at the correct turn-down conditions in a minimum duration of the blow. The material handling system is, therefore, made more than adequate and, more and more automatically controlled. Operation control systems are also being developed to effectively control the refining for obtaining minimum deviation in turn-down conditions.

The static control system using data logging, material and thermal balance calculations and so on, as is universally adopted in almost all LD shops, is found to be inadequate to control a dynamic process. The quality of the steel product in an LD vessel depends on the actual refining operation, i.e., on the process dynamics. The static control becomes more difficult as the vessel size increases. The frequency of getting correct turn-down conditions increases with dynamic control.

Three different types of dynamic control system have been developed viz. the stop method by Nippon Steel, Japan; the route adjustment method by Jones and Laughlin, USA; and the reaction control method by CNRM, Belgium. All of these require assessment of the carbon content of the bath and its temperature during the blow. Sensors have been developed for the same. The carbon content of the bath can be estimated from continuous analysis of waste gases for its CO content. The temperature can be assumed from the dynamic heat balance, and the temperature and flow rate of the waste gases. In yet another design the carbon content is assumed from its liquidus temperature measured by a consumable sensor dipped in the bath. A consumable thermocouple measures the temperature of the sample during its freezing. A sub-lance is used to dip such a sensor in the bath.

In the stop method the blow is stopped at the most desirable point without adjustment during the blow. In the route adjustment method the carbon content and temperature are adjusted during the last stages of the blow to obtain the desired end-conditions. In the reaction control method the refining is controlled right through the entire blowing period by continuous measurements and adjustments.

The information obtained by the sensing elements is fed to the feedback computer, which interprets and transmits the signals that are compared with the reference points. It makes the automatic control system act in time for corrective action. This has been possible by the development of mathematical simulation of the process encompassing all the possible variations in the process operation. The information obtained from dynamic control can be utilized for optimizing the process. The dynamic control, however, is yet to be perfected for its universal applications as a process control tool at a reasonable investment and running cost.

A considerable sum is being spent in the collection of process data, development of sensor elements, and installation of more and more automatic controlling devices on the furnaces.

The reduced need for reblowing, brought about by improved BOF end-point control practices has not only increased production efficiency but has also improved steel yield and extended refractory life, thereby considerably reducing the production costs. Effective dynamic control has resulted in lower dissolved oxygen level at the end of blowing, and which has enhanced steel quality. More accurate predictability of tap-to-tap time has also improved synchronization of BOF with continuous caster operation.

17.10. Concluding Remarks

The LD process, along with all its modifications, that is the BOF process as a whole, has now been developed to the extent of accepting a very wide variety of charges, and producing therefrom a wide range of steels including high alloy steels. The problem of fumes has also been solved fairly satisfactorily and economically.

The process however suffers from a few disadvantages. It needs tall shops to accommodate lance maneuvering equipment and gas cleaning systems. Even after more than four decades, the dynamic nature of the process is a matter of opinion rather than accepted fact. The process always maintains a sort of imbalance between the slag and the metal phases because of the formation of metal-gas-slag emulsion and stratification of the metal phase, which makes the process control a difficult task.

The chief drawback of the LD process that has now been identified is lack of homogenization in the bath with respect to its carbon content and temperature arising out of inadequate mixing, in spite of the supersonic jet striking the bath surface. Any attempt to penetrate the jet in the bath, by way of either lowering the lance or increasing the pressure beyond a certain critical value for better homogenization, results in splashing and that it does not markedly change the extent of mixing in the bath. The stratification of the bath with respect to the carbon content and temperature is almost unavoidable in a standard LD practice. It manifests in the form of sloping, high oxygen potentials in the bath, and lower yields, etc., and tends to damage the bottom.

Since its advent in the early 1950s, the LD process very rapidly became popular because of its faster productivity and better quality as compared to the basic open-hearth which had dominated the steel industry for almost 100 years. The LD process dominated the decades of the 1960s and 1970s without being challenged. By 1980 nearly 550 million metric tons of steel the world over were being made yearly by the LD process. The emergence of oxygen bottom blowing, OBM and the combined blowing processes, however, started replacing the classic LD process of only top blowing since the 1980s. These will be described in later chapters.

17.11. US Oxygen Furnaces 2012-2016 (Source USGS)

Salient Statistics—United States:	**2012**	**2013**	**2014**	**2015**	**2016**[e]
Pig iron production[2]	30.1	30.3	29.4	25.4	23
Raw steel production	88.7	86.9	88.2	78.8	80
Basic oxygen furnaces, percent	40.9	39.4	37.4	37	33
Electric arc furnaces, percent	59.1	60.6	62.6	63	67
Continuously cast steel, percent	98.6	98.8	98.5	99	99

CHAPTER 18

RISE AND FALL OF STEEL MAKING TECHNOLOGIES-IV ROTATING BOF PROCESSES

In This Chapter

- Steel Making by the Kaldo Process
- Steel Making by Rotor Process

Like the LD process of steel making, two other commercially important processes were concurrently, but independently, developed to refine molten iron by gaseous oxygen. The vessel lining, the oxygen lance, etc., used in these processes are nearly similar to those used in the LD process. However, unlike the LD vessel, the vessels in these processes are rotated during refining. A major portion of the CO evolved during refining is burned to CO_2 inside the vessel in these processes, and hence the rotation of the vessel becomes essential to safeguard the lining from overheating and to effect some heat transfer from gas to metal via the lining. The rotation also helps in mixing up the slag, the gas and the metal phases to hasten refining. These processes are known as the Kaldo and the Rotor processes of steel making and are described below.

18.1. Steel Making by the Kaldo Process

The process was developed at the Domnarfvet Works of Stora Kopparbergs Bergslags A.B., Sweden, by a team led by Professor B. Kalling and hence the name. Early experiments were tried on a 3 t rotary furnace and then on a 15 t pilot plant. A commercial plant of 30 t capacity vessels was installed at Domnarfvet in 1954 and it started production in 1956. A 125 t vessel was installed in France in 1963, and a similar one in Sweden in 1961. In 1967 there were a total of ten plants in countries like France, Sweden, the UK, USA, and Japan. The largest vessel has a 135 t capacity. Although the process was originally designed to refine Thomas grade iron it can refine any quality of basic iron.

18.1.1. Kaldo Plant

In the Kaldo process, refining is carried out in a vessel that is similar in shape and about the same size as that of the concentric LD vessel with a solid bottom. It is lined much in the same way as the LD vessel. The vessel is placed in a cradle and is rotated therein, around its long axis, at any desired speed up to a maximum of about 30 rpm during refining. The cradle is mounted on trunnions so that the vessel can be tilted in various positions to permit charging, tapping, slagging, etc. The scheme is shown in Figure 18.1. In the blowing position, the vessel is held at an angle of 16-20° to the horizontal. The waste gases are carried through a water-cooled hood that swings in to fit over the mouth of the vessel during blowing. The

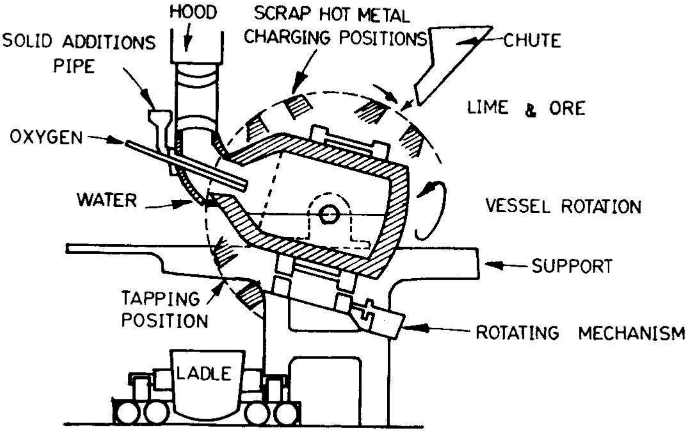


FIGURE 18.1. Kaldo vessel plant.

oxygen lance is inserted through the hood into the vessel at an angle to the surface of the bath. A water-cooled feeder (also known as a feeder lance) is also provided, besides the charging chute at the top, to make solid additions without interrupting the blow.

In order to ensure continuous production, at least two vessels are provided, and one out of the two works at any time. In some plants only one cradle is provided so that the vessel has to be lifted out of the cradle and placed elsewhere for refining. In others, both the vessels are provided with their own cradles, eliminating the task of lifting such a heavy piece of equipment every now and then.

The facilities required for scrap and flux charging, hot metal pouring, tapping, slagging, etc., are in a way similar to the LD process and hence are not described in detail again.

The largest vessel shell is 8.5 m in length and 5.5 m in diameter. Fins are welded to the shell for effective cooling. The volume of the vessel after lining is around 0.45-0.55 m^3/t capacity as against 0.75 m^3/t of LD vessel. The specific volume increases with increasing phosphorus content of iron to accommodate the extra volume of slag that will be formed.

Oxygen is fed through a water-cooled lance as used in an LD process. It usually has a single nozzle but multi-nozzle lances have also been used. The lance is capable of oscillation in the vertical plane through an angle of 20-37° to the horizontal. It usually oscillates about 15-20 times per minute. It can also be held in any fixed inclined position during blowing. The rate of oxygen input needs to be adjusted during refining.

A water-cooled feeder is provided to charge sized solid materials during the blow. These are blown into the vessel at the rate of 2 t/min by compressed air.

Tarred dolomite is cheapest but the life of 50-60 heats is quite inadequate for the other vessel to be relined, hence magnesite lining is preferred.

A portion of heat released by the burning of CO to CO_2 inside the vessel is transferred to the metal via the lining by rotation of the vessel. The lance is oscillated to sweep the entire slag surface to hasten heat transfer because, as in open-hearth, the slag acts as an insulator in between.

Although it is designed to refine any iron, it was adopted more for the Thomas iron. It is also designed to take up any proportion of metal and scrap like the open-hearth.

18.1.2. Heat

The hot vessel, containing some slag from the previous heat, is rotated into vertical position and it receives lime, ore, and other fluxing material. Scrap is charged after making the vessel nearly horizontal. Scrap is literally poured in the vessel. Hot metal is charged immediately after the scrap. The vessel is tilted to blowing angle of 16-20° to the horizontal. The hood is swung in and held firmly against the nose of the vessel. The lance is inserted through the hood at a proper angle. The rate of oxygen supply and the speed of rotation of the vessel are varied during blowing. Blowing commences at an oxygen flow rate nearly 50-60% of the maximum allowable value. The vessel is also rotated slowly. Both are increased progressively and they attain their respective maximum values in about 5-7 minutes.

Initial conditions are such that all the silicon, most of the manganese, and some iron are oxidized. The formation of SiO_2, MnO, and FeO take lime in solution and form a thin, basic and oxidizing slag. The slag retained from the previous heat is helpful in this respect.

If a high proportion of scrap is used as coolant, nearly 50% is charged in the beginning, and the remaining at the end of about 10 minutes of blowing. Since the scrap charge is more, all of it cannot be charged in the beginning for it may cause severe chilling of the bath. The refractory wear also increases because the scrap pieces get dragged during rotation of the vessel.

At the end of 18-20 minutes of blowing all the solids are completely molten. The oxygen flow rate is then decreased by nearly 50% and the speed of rotation is reduced to about 10-12 rpm. The lance angle is also decreased. This is a dephosphorization period. The blow is stopped after nearly 25 minutes of blowing and the phosphoric slag is removed as completely as possible. The slag at this stage should be slightly foamy and thin so that metal shots do not remain entrapped and get lost along with the slag removal. In a two-slag operation almost 80% of the total oxygen is blown before the removal of the first slag. Bath temperature is measured at the end of the first slag removal.

Fresh lime and ore, for correcting temperature of the bath, are added and blow is continued at a vessel speed of 6-8 rpm and low oxygen flow rate. In a two-slag practice, the second blow continues for a predetermined time, at the end of which, bath analysis and temperature are determined and the heat is tapped. If a very low phosphorus level is to be achieved, one more intermediate slag is removed and the blow is continued. This slag may be recycled in the next heat.

The third blow is very short. Fresh charges of lime and ore are made again before blowing. The heat is finished by withdrawing the lance and the hood, adding fresh lime and rotating the vessel at its maximum speed for just one minute. It clarifies the slag and the metal layers and thickens the slag to an almost solid state, which renders formation of a slag dam unnecessary for tapping.

The final bath analysis and temperature should be obtained as required by adjusting the last stage of the blow. The steel is tapped by piercing the solid slag layer. Finishings may be added to the ladle. The nearly solid slag is left behind in the vessel for the next heat. This slag does not react violently with the hot metal. The nose should be cleaned and repaired before commencing the next heat. The sequence of elimination of the impurities is shown in Figure 18.2.

If the silicon content of hot metal is high, the same vessel can be used to desiliconize the metal. Oxygen is blown at its maximum rate and the vessel is rotated at its top speed for desiliconization. A very low basicity slag is made and is removed. The rest of the operation is carried out as described above. This relatively acid slag should not be mixed with basic phosphoric slag.

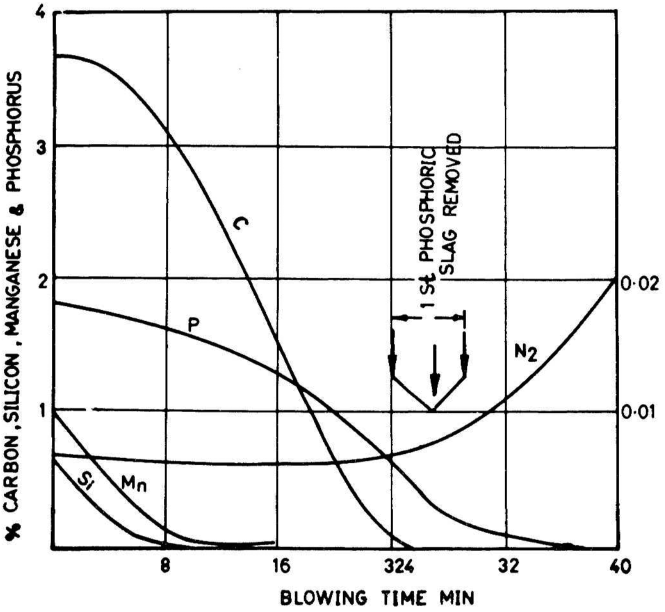


FIGURE 18.2. Sequence of elimination of impurities in a Kaldo process (schematic).

Ore additions. If the bath is to be cooled by iron ore alone, it should be added in batches to prevent chilling of the bath. The first lot of ore is charged after the formation of the first slag, i.e., after 5-10 minutes of blowing. The gas off-take temperature is used as an indication to charge the next quantum of coolant. Ore should not be added to a cold bath or else sporadic eruptions may occur resulting in metal ejections. Although it is designed to take an equivalent amount of ore as coolant, in general maximum use of scrap is made to cool the bath.

18.1.3. Process Control

The process is controlled by varying the speed of rotation of the vessel, the flow rate and pressure of oxygen, and the lance angle. Rotating the vessel at a very slow speed is good enough to cool down the lining heated by the combustion of CO inside the vessel. High speed of rotation is made use of as a major tool to control refining. As the vessel rotates, the liquid metal is drawn up along the wall, and it slides back along the wall if the speed of rotation is low. At high speeds (30-40 rpm) the metal that is drawn up along the wall cascades back and a shower of metal droplets falls through the gaseous atmosphere inside the vessel. The droplets are, therefore, oxidized directly by the gas phase. It also leads to the formation of a tremendous amount of slag-metal interface at which the partitioning of P, Si, Mn, and S takes place.

The jet velocity is lower than in LD and it is inclined to the bath surface at a shallow angle. Neither of these factors leads to the formation of emulsion to any significant extent as in the LD process. The high speed of rotation of the vessel, however, more than compensates in this respect.

With such a large available slag-metal interfacial area, the oxidation of Si and Mn do not pose any problem in a basic process. These are oxidized in the early part of the blow itself. The formation of thin, basic, and oxidizing slag takes place within a short time, due to the presence of slag from the previous heat and also due to the heavy bath oxidation in the beginning, because of the low speed of rotation. It takes nearly 10-12 minutes to form such a slag and the rate of dephosphorization rapidly builds up thereafter.

The decarburization rate is initially slow, but once the metal cascades (after 5-7 minutes of blow) and metal drops are exposed to oxidizing gases (oxygen and carbon dioxide), it builds up very rapidly. The rates of decarburization and dephosphorization are adjusted to finally obtain the correct turn-down conditions.

Unlike the LD, the dephosphorization need not be completed well ahead of the collapse of emulsion; dephosphorization and decarburization can be carried out much more independently in Kaldo than in the LD process. Much of the carbon is eliminated well ahead of phosphorus removal.

The rate of decarburization is increased by:

(*i*) increasing the speed of rotation which causes more metal droplets and thereby more gas-metal interfacial area to form,

(*ii*) increasing the velocity of oxygen flow,

(*iii*) lowering the lance further down, and

(*iv*) making the lance steeper.

The rate of dephosphorization is increased by:

(*i*) decreasing the speed of rotation of the vessel, which builds up (FeO content of the slag),

(*ii*) decreasing the velocity of oxygen flow,

(*iii*) raising the lance further up, and

(*iv*) making the lance angle shallower to build up (FeO) content of the slag and to burn more CO to CO_2 to thin down the slag.

The amount of CO to CO_2 reaction inside the vessel can be increased by a shallow angle of the lance. A shallow angle directs oxygen more towards the vessel bottom and is thus available for burning CO; it also thereby sweeps the entire slag surface more effectively and hastens heat transfer. Heat transfer from the gases to the metal bath is hastened by rapid rotation of the vessel and formation of thin slag. If silicon content is high and more lime is added, heavy sponging of the slag occurs. It can be minimized by increasing the speed of rotation of the vessel and by decreasing the oxygen flow.

A suitable combination of the above factors can alter the relative rates of decarburization and dephosphorization in the desired direction. At any moment all of these factors may not be available for adjustment, but the adjustment can be made with whatever factors are available for adjustments at the time. These factors are adjusted, manually and/or automatically, continuously during the blowing period. The process is controlled by preplanning the blow in terms of the charge materials and the end-product. The adjustments required are so many, that as far as possible automatic control is affected as in the LD process.

18.1.4. Operating Results

The Kaldo process is more versatile than LD to produce a wide range of products using hot metal of almost any composition. The slower rate of refining and the availability of several factors to control the process as compared to LD make it more amenable to obtain steels with low P, S, C, and N for extra deep-drawing purposes. It can also produce carbon and alloy steel as in the LD process. It produces fertilizer-grade slag that can be sold as a byproduct.

The approximate compositions of slags are

	1st slag	*2nd slag*	*3rd slag*
% CaO	50 or more	50 or more	40-50
% P_2O_5	18 or more	10-12	6-7
% SiO_2	8	6	4
% Fe	10-12	18-20	25-30

The tap-to-tap time is in the range of 44-60 minutes.

The output varies with practice, i.e., on the number of slags made. The yield varies with the proportion of iron ore used as coolant. In general the values are better than those in the LD practice.

18.1.5. Concluding Remarks

In spite of being a very scientifically designed and developed process and hence a promising process at the beginning, it could not be widely adopted for the simple reason that the refractory consumption per metric ton of steel produced is still very large in Kaldo vessels. The lining life is nearly 60-100 heats as against 200-300 heats for similar material in an LD vessel.

Such a low lining life drastically reduces the furnace availability and decreases the production rate. The Kaldo process is a typical example emphasizing the importance of refractory consumption in the economics of steel making processes. The process has been modified to overcome its major drawback to some extent and is described below.

18.1.6. LD-Kaldo Process

The vessel in this process can be blown in a stationary vertical position as in the LD or LDAC process and can be rotated and blown in an inclined position in a Kaldo process. It is, therefore, named the LD-Kaldo process. The vessel design is intermediate between the two, i.e., the volume is 0.75 m^3/t as against 1 m^3/t in LDAC and 0.5 m^3 in Kaldo. The process is designed to refine Thomas grade iron. The vessel is blown in a vertical stationary position in the first part. After the removal of the phosphoric fertilizer-grade slag it is rotated in an inclined position, oxygen is blown at low pressure, and the heat is finished as in the Kaldo process. The details of the practice vary with respect to the final product requirements. The tap-to-tap time is between 55-70 minutes. The process can better utilize the retained slag and can control the composition and temperature of the bath more precisely as in the Kaldo process. The yield is better than that in the LD process and lining life is better than in the Kaldo process. Carbon and alloy steels can be produced more easily as in the Kaldo process. In short it retains the advantage of the Kaldo process and at the same time improves its lining life over the standard Kaldo practice. It is particularly suited for producing a variety of steels otherwise produced by LD and Kaldo independently.

A plant with 35 t vessels was first commissioned in Belgium in 1965, and a second plant with 70 t vessels was completed in 1973.

18.2. Steel Making by Rotor Process

The Rotor process was developed at Oberhausen in Germany in 1952. A rotary melting furnace was used for early trials wherein the burner was replaced by the oxygen lance. The original idea was just to pre-refine the blast furnace metal to subsequently refine it in an open-hearth. The trials, however, indicated that it could by itself refine the hot metal plus scrap change to finished steel. A 60 t rotary furnace was, therefore, designed and installed on the blast furnace casting bed. A few more plants with 100-200 t capacity vessel are now in operation in the UK, South Africa, and Germany. The process is known as the Rotor process of steel making.

A long cylindrical vessel (length ≅ 4 times the dia) is used to provide enough space for burning most of the CO evolved during refining to CO_2, and thereby utilize the heat for steel making. The primary oxygen lance or refining lance is dipped in the slag-metal system, whereas a secondary lance is inserted well above the slag surface to burn the evolved CO to CO_2. The turbulence caused by the primary submerged lance and the sweeping of gases over almost the entire slag surface in the vessel causes a good proportion of heat released in CO to CO_2 reaction to be used in steel melting. As in Kaldo about 80% or more of the evolved CO is burned to CO_2 inside the vessel. The rotation of the vessel is essential to safeguard the lining from overheating and assist in heat transfer.

18.2.1. Plant Design

For 100-120 t capacity the vessel is about 15 m long and 4-5 m in diameter with nearly 1-1.2 m openings at both ends. The vessel is rotated around its own axis in horizontal plane at a fixed speed of 0.2-4.0 rpm. The vessel is mounted on a turntable of about 8 m diameter and which can be rotated in a horizontal plane. The arrangements around the table are such that the lance and gas off-take are located at diametrically opposite points as shown in Figure 18.3. Similarly a burner and another off-take are located in case scrap is to be melted or preheated. The table has tapping and deslagging positions. The vessel can be raised in a vertical plane up to 90°. For tapping, an inclination of a few degrees is all right but for changing the vessel it is made erect. A jack car always takes the old vessel for relining and an already relined vessel is replaced to maintain continuity in steel production. The lining is similar to LD and Kaldo vessels. Ground charging machines are used to charge solid materials. The lances are water-cooled. The primary lance has one nozzle and the secondary may have one or more nozzles.

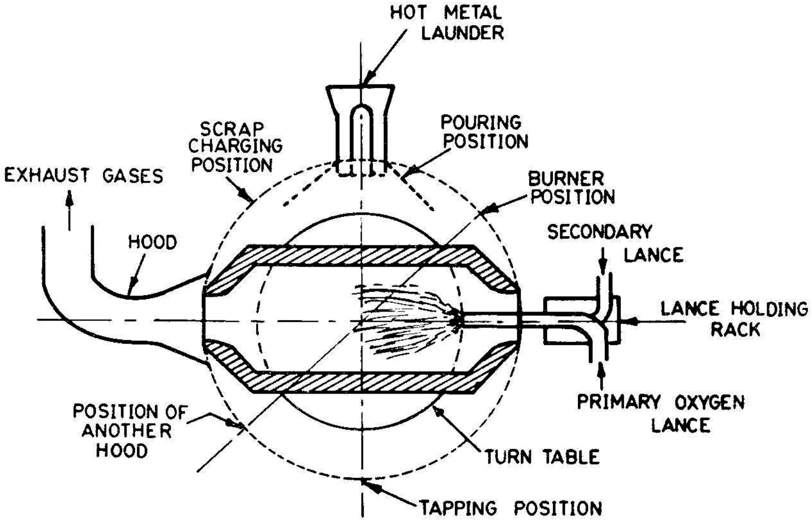


FIGURE 18.3. The rotor plant.

18.2.2. Operation

Some or all of the final slag from the previous heat is retained for the next heat. The heat commences with hot repairs, followed by charging some lime. The vessel is rotated to spread it all over the lining. The remainder of lime, scrap, ore, and hot metal are then charged. If the scrap proportion is more, it is charged from both ends. The vessel is moved to the blowing position. The lances are inserted and blowing is started. Simultaneously the vessel is also rotated. The primary oxygen flow is nearly constant throughout the blow at about 70-85 m^3/min. The secondary lance is put on a little after the primary is put on and it blows air, enriched air, or pure oxygen per the design. It is also equipped to feed lime powder if required. The vessel is blown for half the time from each end to even out lining wear over the entire length of the vessel.

The first part of the blow lasts for nearly 20-24 min., 10-12 min. from each end, by which time the first phosphoric fertilizer-grade slag is ready. The vessel is then rotated without blowing to clarify slag and metal. The slag is removed and a new one is formed by fresh additions of lime directly or through the secondary lance. A little sand may be added to hasten slag formation. The second blow may last for up to a minute from each end. The sampling, analysis, temperature measurement, tapping, and adjustments are carried out as in LD and Kaldo practices.

18.2.3. Control

If the LD and Kaldo process control factors are properly understood it should be easy to appreciate the Rotor process control. Since the process could not become popular, automatic control is not developed to the same extent as those of LD and Kaldo processes. The control is still carried out more manually, and the experience of the melter is a big asset. The noise of the jet after slag formation is used as a guide. Decarburization is hastened by dipping the lance further down in the bath, and dephosphorization is hastened if it is withdrawn and (FeO) is allowed to build up in the slag. The vessel speed does not influence refining at the low speeds employed in this process. Refining is controlled only by adjusting the oxygen pressures in both lances. Blowing from both ends in the latter part of the blow helps to homogenize the bath composition.

18.2.4. Operating Results

The process is designed for Thomas iron and does not pose any problem producing dead soft or even carbon steels. The silicon content of iron should be low to produce fertilizer-grade slag. If it is more, then metal may be desiliconized in the same vessel as in the Kaldo process prior to the usual refining.

Typical operational times are:

Charging	20-23 min.
Total blowing	35-40 min.
Deslagging, sampling, etc.	18-20 min.
Tapping	7-8 min.
Total	80-91 min.

If the time required for hot repairs is taken into account, a tap-to-tap time to nearly 2.0-2.5 hours may be obtained. Primary oxygen is blown at 6-12 atmospheric pressure and is directed at an angle of 27-45° to the bath surface. As much as 50% of the total time requirement of the process may be fed through the secondary lance. The slags are similar in composition to LDAC or Kaldo slags. It can take up to 40-45% scrap or equivalent ore or a combination of bath as coolants.

The amount of total refractory lining required in a Rotor vessel is 3-4 times the nominal capacity of the vessel, of which nearly one-third is working lining. The lining life of good quality magnesite in this process has been as much as 400-500 heats without fettling.

18.2.5. Concluding Remarks

As in Kaldo, the refractory consumption in the Rotor process also happens to be fairly high due to the burning of CO inside the vessel. This problem has been solved to some extent by using compressed air in place of oxygen to burn CO. In some cases the CO is not at all burned inside the vessel by withdrawing the secondary lance. The plants that were constructed in earlier years continue to work satisfactorily using the above modifications.

The process however is unable to compete with LD in terms of capital and operational costs. Hence no new installation is likely in near future.

The beneficial effect of vessel rotation could, however, be successfully utilized in the production of costly alloy steels, which is appreciated. It may give a fresh lease on life to this process in commercial practice.

CHAPTER 19

OXYGEN BOTTOM BLOWING PROCESSES (OBM)

In This Chapter

- Historical
- OBM Process
- Operation
- Metallurgical Aspects
- Modern Developments
- Concluding Remarks
- Future Prospects of OBM
- Applicability Under Indian Conditions

19.1. Historical

As the deleterious influence of high nitrogen contents in steel on its cold workability and weldability was recognized, during World War II efforts were made to decrease the nitrogen content of steel, made particularly by basic Bessemer or Thomas processes, by enriching the blast with oxygen. The vessel bottom life drastically decreased as the oxygen content of the blast increased to 40%, beyond which point it became uneconomical to increase. The nitrogen level, in this way, came down to 0.001% from the usual 0.015% while blowing air. The proportion of scrap that could be charged increased from 8-10% to about 20% due to the oxygen enrichment.

Refining of pig iron with pure oxygen could, therefore, be carried out only by introducing it from the top as in the case of LD, or from sides as in the case of Kaldo or Rotor and their modifications. Out of these, the LD process using a water-cooled oxygen lance, introduced vertically from top, became popular and soon dominated the steel industry. It had to be modified in the form of LDAC/OLP to meet the requirements of Western European countries where Thomas grade iron was to be refined. Although this modification did offer a solution, efforts did not stop the development of a process of blowing oxygen from the bottom of the vessel in the classical Bessemer style. The problem was solved in a rather ingenious way.

In the bottom blowing processes, in principle, oxygen is introduced from the bottom through tuyere holes with a peripheral shield of protective (heat sink) fluid. The tuyere is, therefore, made of two concentric tubes, oxygen being blown through the inner tube and the protective fluid through the annular space between the tubes. The process was developed by the Maximillianshutte Iron and Steel Company in West Germany in 1967 and was named the OBM-oxygen bottom blown process. It used propane or other gaseous hydrocarbon as a protective fluid. A process similar in design using steam or fuel oil as a protective medium was developed in France in 1969 and was named the LWS process. US Steel developed their own version of a similar process using gaseous protective fluid and named it the Q-BOP (quiet or quick or springing up oxygen bottom blowing process). In this context the OBM is the general term used for all such designs and processes.

19.2. OBM Process

The OBM vessel is essentially a Bessemer-like converter fitted with a special bottom as shown in Figure 19.1. The tuyeres are inserted from the bottom in such a way that the oxygen would be surrounded by a protective hydrocarbon gas like propane. On entry, propane cracks down in an endothermic reaction and takes up some of the heat generated by the entry of oxygen. The relative feed rates of these two fluids are adjusted to obtain optimum temperatures at the tuyere tip and thereby ensure its reasonable life as well as speed of refining. The deposition of carbon, which is a product of cracking, also helps to protect the bottom from heat generated due to the refining reactions at the tips of tuyeres.

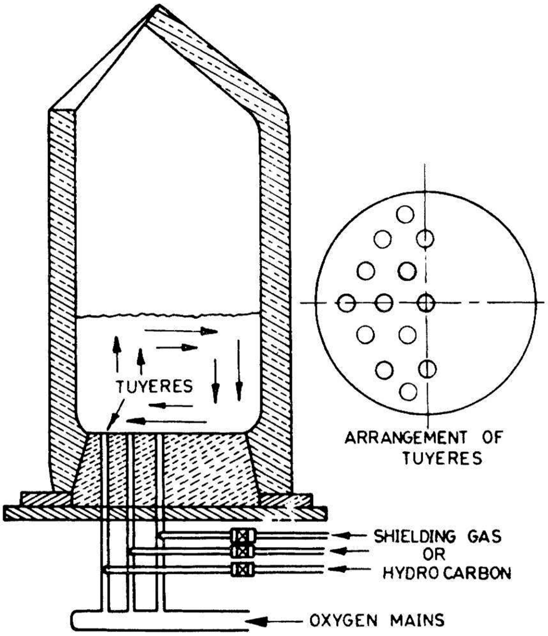


FIGURE 19.1. Oxygen bottom blown converter (OBM process). One of the possible arrangements of tuyeres in an OBM converter is also shown. The arrows in the bath indicate the direction of circulation of the bath.

The other product of cracking is hydrogen. It finds its way into the metal. The amount of hydrogen thus dissolved increases particularly towards the end of the blow when flow rate of hydrocarbon is increased relative to that of oxygen. A nitrogen rinse is given at the end of refining to desorb dissolved hydrogen. But even so, these steels end up with high hydrogen contents as compared with Thomas steels. The distinct advantage of this situation is that nearly 50% of the hydrogen burns with dissolved oxygen and thereby generates heat.

In order to promote turbulence in the bath and thereby ensure good slag-metal contact, the tuyeres are arranged only on half the converter bottom as shown in Figure 19.1. Experience dictated that provision of a few bigger tuyeres is better than a large number of fine tuyeres. Maintenance problems are minimized without losing in terms of metallurgical requirements of turbulence. By this arrangement, it is ensured that the direction of metal circulation is upwards in the tuyere half of the vessel, and downwards in the other half. This arrangement is also helpful in minimizing the damage to tuyeres while charging scrap, since it can now be charged on that part where there are no tuyeres. It automatically increases the vessel capacity for the same inner volume, as compared to the Thomas vessel, since hot metal can be filled up to the almost-half bottom area in an inclined position. The existing Thomas converters, when converted to OBM, in fact increased their capacity by anything up to 40-50%.

The tuyere arrangement is such, and coupled with the provision of hydrocarbon and oxygen for usual blowing, it can be used for preheating the scrap. The scrap proportion in the charge is thus rendered flexible to run the process with equal ease.

The tuyeres are generally made of stainless steel pipes embedded in the magnesite bottom. Longitudinal ribs in the annular part maintain the dimensional accuracy until the end of its useful life. This is essential for consistent operation of the converter during its entire campaign. During blowing the pressure of the protective gas is about 20-50% below that of the oxygen.

19.3. Operation

The operation is described by illustrating the process at the Sulzbach-Rosenberg plant of Maximillianshutte Iron and Steel Company. The shop previously had 6 × 25 metric ton Thomas converters and produced 0.7 million metric tons of steel per annum. These were converted to 6 × 30 metric ton OBM vessels and produced 1.0 million metric tons of steel per annum. The bottom life of 50 heats in Thomas increased to 200 heats in OBM.

The operation of the vessel is more like the Thomas process since it has been developed for Thomas grade iron (1.8% P). After charging scrap and hot metal, blowing is started. Lime is added immediately afterwards from an overhead chute. The blow is divided into three distinct parts. The first part lasts for 16-17 minutes, at the end of which a high phosphoric slag, 18-22% P_2O_5 and 12-15% FeO is raked off. The carbon and phosphorus are analyzed by taking a sample. The analysis is generally 0.3% C and 0.08% P at this stage. Fresh lime is added and blowing continued for about a minute when the carbon comes down to 0.1% and phosphorus to 0.025%. The final blowing is of nitrogen without protective gas and thereby the bath averages finally 0.0004% H_2 and 0.005% N_2.

The relevant data are given below:

Vessel capacity	30 metric ton
Charge	20.5 metric ton of hot metal and 12 metric ton scrap
Oxygen pressure	10 atmos
Oxygen consumption	200 N-m^3/min or 60-65 N-m^3/metric ton
Propane pressure	6 atmos
Propane consumption	0.5 N-m^3/min or 2 N-m^3/metric ton
Lime consumption	125 kg/metric ton
Tap-to-tap time	40 min
Yield	90-92% depending upon scrap rate.

The process is capable of producing all varieties of plain carbon steel with equal ease, using all varieties of hot metals.

19.4. Metallurgical Aspects

In the classical air blown Bessemer process the decarburization rates of the order of 15-20 wt% per hour are obtainable. At any gas metal interface, the nitrogen may still be obstructing the transport to some extent in Bessemer but in oxygen bottom blown processes even this barrier to transport, in reaction kinetics, is not present; hence the refining reactions are extremely fast in OBM processes. The stirring caused by the blowing action therefore results in achieving near equilibrium conditions with respect to the gas-metal and metal-slag interfaces. The system as a whole, inside the vessel, operates at near equilibrium. On the contrary, in top blowing, oxygen is supplied to the metal not directly as in OBM, but through a slag layer thereby overoxidizing the slag beyond the equilibrium state. Even the use of a supersonic oxygen jet does not result in required stirring of the metal bath in top blowing, thereby resulting in temperature and concentration gradients within the bath causing periodic slopping and ejections. No such phenomenon is observed in top blowing. Depending upon the top lance position some oxygen is always available for CO combustion above the bath, which contributes some heat to the process, in turn allows 3-6% additional scrap in the charge. This is practically absent because of almost 98% oxygen being reacted with metal in OBM;

hence that much scrap rate is lower in the OBM. If scrap is cheaper, the top blowing can offer some cost advantage in this respect. The iron losses in top blowing are nearly 5% more than those in OBM. Very low carbon steels are achievable in top blowing only at the expense of extra iron loss in slag. But this is readily achievable in OBM. This also leads to a situation wherein higher carbon levels can be obtained by catch carbon techniques, easier in LD than in OBM, at low P contents. The stirring intensity, which is estimated to be nearly ten times more in OBM than in LD, gives better partition of phosphorus and sulfur, higher manganese and lower oxygen at turndown, resulting in better ferro-alloy recovery. The sequence of elimination of impurities as observed in a 230 t capacity OBM vessel at Chiba, Japan is shown in Figure 19.2.

Because of the introduction of hydrocarbons, the OBM steels have higher hydrogen contents; even if an argon rinse is adopted it is still on the higher side. The purging requirements add to the cost of steel making. A brief summary of the characteristics of the top and the bottom blown processes is given in Table 19.1.

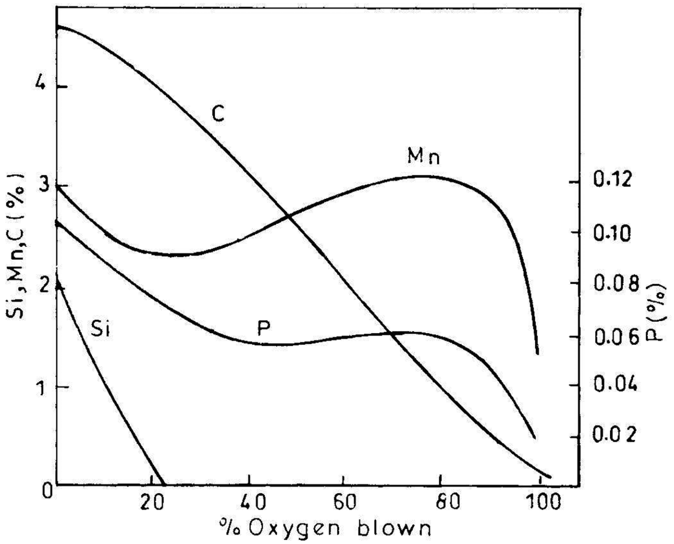


FIGURE 19.2. Sequence of elimination of impurities as observed in a 230 t OBM vessel at Chiba, Japan [31].

Table 19.1 [32]

Parameter	*LD*	*OBM*
Fe in slag at 0.04%C	17-22	10-15
Fe lost in dust kg/t of steel	10	2
Oxygen blowing rate Nm^3/t of steel	3-3.5	up to 5.5
Sp. O_2 consumption Nm^3/t	48-60	45-55
Oxygen in bath at 0.04%C	0.08	0.04
Min. %C in bath at turndown	0.04-0.05	0.01
Typical product C × O in the bath (difference higher for low %C)	30	22
Al recovery, %	standard	+ 10
Hydrogen at 0.04%C, ppm	2-3	4-5
Nitrogen at the same level, ppm	20-40	15-30
Ratio of oxygen potential of slag to that of metal	8-10	1
Mn% at 0.04%C	0.1-0.15	0.25-0.30
Lime size used, 1000 cm^2/cm^3	100-200	800-1000
P partition coefficient at 0.04%C	70-80	100-120
S partition coefficient at 0.04% C	4.5-6.0	7-8
CO% burnt above bath	5	2-3
p_{CO} at 0.05%C, atmos.	1.0	0.6
Scrap% in charge at 1.5%Si	32-33	27-28
Slopping	always	virtually nil
Yield of liquid steel, %	standard	+ 0.7-1.5%

The LD process has been widely used for making low and medium carbon steels using nearly 55 Nm^3 of oxygen per metric ton of steel at a blowing rate of 3-3.5 Nm^3/min/t and 20-25% scrap; whereas, the OBM is ideally suited for even low carbon steels (0.01-0.02%) at a blowing rate of oxygen nearly 4-4.5 Nm^3/min/t and scrap rate 4% lower than in LD. The scrap rate, besides other factors, primarily depends on silicon contents in iron. With the rise of silicon from 0.5 to 1.8, the scrap may go up correspondingly from around 20 to 35%.

The fume problem is serious in LD, but it is problematic in OBM during inclined positions of the vessel. It is easier to feed lime with oxygen in top blowing than in OBM. The thermal balance at the tuyere tip in OBM is very critical and has to be carefully adjusted, by the flows of oxygen, hydrocarbon, and the lime. The maintenance of the bottom tuyeres against any mechanical damage requires close watch. On the whole, it adds to the cost of OBM.

A description of the mechanism of OBM tuyere in refining is not readily available. One such attempt has depicted it as shown in Figure 19.3. As the hydrocarbon comes in contact with the molten metal, it gives rise to a cracked mushroom growth and then perhaps accumulation of carbon at the tuyere tip. This is perhaps followed by intense combustion of this carbon with oxygen. The hydrogen from cracking may dissolve in the metal, which later combines with oxygen giving heat. The way all this damages the tuyer tips is still a matter of guess only, but requires study to devise a means of improving the tuyere life.

It is expected that the refractory performance of the main lining of the OBM vessel should be comparable with that obtainable from any

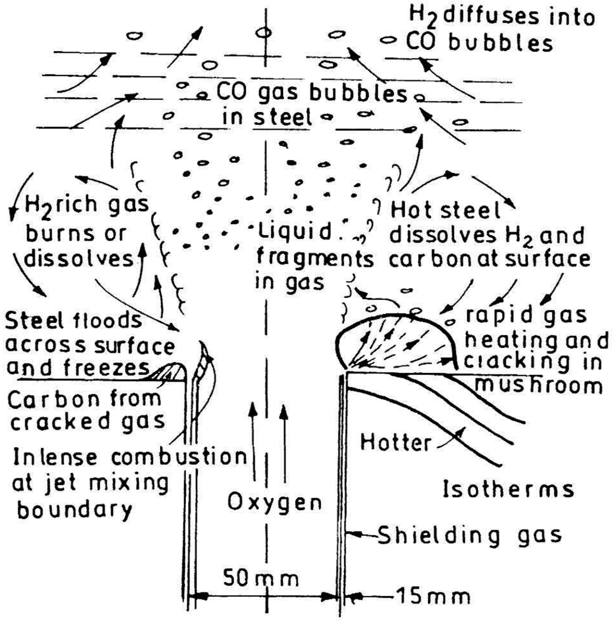


FIGURE 19.3. Tentative mechanism[33] of tuyere protection. The right half of this scheme shows what happens in the beginning on instantaneous exposure of hydrocarbon to molten iron and the left hand side that thereafter.

conventional vessel. The unique bottom design in OBM is the heart of the process and can also become a limiting factor in vessel operation unless the life on the whole matches with that of other lining.

19.5. Modern Developments

The OBM also underwent several modifications in the direction in which the BOF processes were modified in the last decade or two. The possibility of using argon in place of nitrogen for the final rinse did improve upon the nitrogen contents in the final steel. It similarly improved upon the hydrogen content as well.

The OG gas recovery system was designed and adopted for OBM at the CHIBA, Japan. It achieved a world record of 265×10^3 kcal/t in 1980 owing to the uniform nature of decarburization reaction in this process. The CO content of the gas recovered is higher than in LD, and the calorific value increased to 2700 kcal/N-m^3.

To improve upon the gas recovery in the end, and to effect endothermic reaction at the tuyere tip to improve upon the lining life, injection of limestone as partial replacement of lime was adopted and proved very effective. The limestone got calcined at the expense of extra heat available at the tuyere tip, and the resulting carbon dioxide reacted with the bath carbon to decarburize the same, thereby finally enriching the gas phase with carbon monoxide. The overall consumption of lime was also found to be somewhat lower than when only lumpy lime was used.

The process has been fully automated and controlled by the dynamic computer control, which is easier in this case than in the LD process. This has resulted in developing a quick and direct tapping technique in which it has been possible to tap the vessel without waiting for sampling and analysis, thereby decreasing the heat time.

It has also been used as a means of dephosphorizing hot metal with a short blow of 2-4 minutes, using lime, ore and spar additions along with oxygen.

19.6. Concluding Remarks

The OBM process is not only efficient in refining, but the results are consistent with respect to chemistry of the refined steel and its temperature.

It no doubt suffers from disadvantages. The construction of tuyeres and its maintenance is a big headache. Dead soft steels cannot be made by this process since the nitrogen content is high. Similarly the hydrogen content is also on the higher side. The refractory consumption is also rather on the high side.

The process can accommodate a higher percentage of scrap by resorting to its preheating in the vessel using the tuyeres as burners. Fume formation is much less than in LD, hence the gas cleaning requirements are not as extensive as in the case of the LD process.

The shop height is almost equal to that of an open-hearth and hence can replace the open-hearth using the same sheds. Replacement of open-hearth by LD requires taller shops. For a new installation, therefore, the cost is nearly 30% lower than that for the LD shop.

The understanding of metallurgical aspects of bottom blowing opened up a completely new vista in refining, that is, that conditions close to equilibrium could be obtained in slag-metal systems during refining without sacrificing efficiency of refining. This has now been fully exploited in developing combined blowing processes wherein both top and bottom blowing are adopted to obtain advantages of top and bottom blowing both, in one single process. These are described in the following chapter.

The process has also been considered as a possible simple solution in adopting argon bubbling along with desulfurizing agent injections and use of synthetic slags for obtaining better quality steel, which otherwise is possible through secondary steel making alone.

Practically all the leftover Thomas shops have changed over to this process. The new installation has reached the size of 250 t capacity at a number of places.

19.7. Future Prospects of OBM

There are some important aspects of OBM that need to be exploited further. These are:

A substantial amount of inert gas, 2 m^3, and a large amount of flux, 20 kg/t, can be injected through the bottom blown converters per metric ton of steel per minute. The temperature losses of the bath for such treatments are relatively much less. Some important achievements of ladle furnace secondary refining using argon injection and flux treatment can thus be accomplished in the bottom blowing converter in the future.

None of the secondary refining techniques permits simultaneous desulfurization and dephosphorization. In the bottom blown converter, desulfurization under oxidizing conditions can be achieved by obtaining highly dispersed slag with a high lime content by the injection of lime powder with argon. In the after blow period of about a minute nearly 90% dephosphorization and 50% desulfurization have thus been achieved. For example, by injecting 20 kg lime/t of steel, the sulfur was reduced from 0.032 to 0.018% and phosphorus from 0.010 to 0.001% in about a minute of blowing.

Final obtainable carbon content in OBM is of the order of 0.02% at the end of blowing. By purging with 2 m^3 inert gas for about one minute it is reduced to 0.01%; and by purging for three minutes, carbon content as low as 0.005 has been attained. Such low carbon contents are achieved in vacuum treatment plants only with a long treatment time. By the same purging, the hydrogen content is simultaneously reduced down to 1.5 ppm. The oxygen content of the bath is also reasonably low.

The possibilities of such achievements through OBM, even after its suitable modification, need to be probed further to obtain steels with better specifications.

19.8. Applicability Under Indian Conditions

Specific trials have been carried out on hot metal with typical composition under Indian conditions (high silicon and phosphorus 0.3%) at the IRSID pilot plant in France using the LWS bottom blown design. The metal temperature was 1330-1350°C. Flux mixture with 70% CaO and 14% CaF_2 was injected from the bottom. Trials were taken with and without intermediate slagging. The results without intermediate deslagging were superior to those with intermediate deslagging. The phosphorus content of around 0.03% for carbon level of 0.3% was achieved. Sulfur could be readily reduced from above 0.04 to 0.02%. Carbon content could be arrested at a higher level along with required P and S removal. Extra low carbon steels could also be produced up to 0.06% C at 0.02% P. A short metallurgical stirring at the end decreased the phosphorus further down to 0.04 to 0.017%.

It thus looked very promising for adoption in Europe and India and might have been preferred, but for the fact that around the same time the bottom agitated processes offered a simpler alternative; hence the adoption of OBM was shelved. The final reliable status of hybrid processes in the 1980s proved the demise of the OBM process. Wherever it has already been installed it would continue to operate until its establishment costs are amortized, but it should be accepted that the last pure OBM vessel has already been built.

CHAPTER 20

MODERN BOF (HYBRID) PROCESSES

In This Chapter

- Introduction
- Hybrid Process Spectrum and Their Characteristics
- Bottom Gas Purging Design
- Metallurgical Superiority of Hybrid Blowing
- Future of Hybrid Processes
- Some Hybrid Plant Details[34]
- Hybrid Process at Tata Steel[24]
- Concluding Remarks

20.1. Introduction

In the absence of any better alternative, LD was the only choice for new steel plants and also for replacement of old steel making shops in the 1950s, 1960s, and 1970s. During the period before and after the Second World War, the fundamentals of steel making began to be appreciated by the steel makers. The diffusion-controlled kinetics of refining reactions and factors affecting this soon became apparent. The LD process was developed more from the point of view of thermodynamic advantages of using oxygen for refining. The study of the physics of the interaction of oxygen jet with molten metal, however, revealed the severe limitations of this process in achieving the desired metallurgical results. In spite of impinging the molten bath with a supersonic oxygen jet from the top, the kinetic energy of the jet does not get converted into the desired effect in stirring the bath. This gives rise to concentration and temperature gradients within the bath and the attendant problems of getting the correct chemistry and temperature at the turndown. Often it requires reblows and a subsequent fall in productivity. Production of very low carbon steels poses some difficulties.

In a way as a reaction to LD and modification of Bessemer, the OBM process was developed, and at one time it appeared that it would overtake the LD in its popularity for steel making. Intense investigations in OBM revealed many pitfalls from which it suffers, but it established the fact that bottom blowing results in near perfect stirring required for steel making, and because of which near equilibrium conditions prevailed in OBM vessels during blowing.

Attempts to increase the stirring in top blowing by increasing the oxygen pressure or lowering the lance towards the bath did not prove very effective; in fact it proved counterproductive beyond a certain limit because of resultant excess slopping and ejections. These problems become more serious with increasing silicon contents in iron.

Interestingly, the study of even mild bottom stirring resulted in good mixing and increased mass transport in two-phase simulated systems. This finally led to the development of what are now known as bath agitated processes, or BAP. This is brought about by introducing a small amount of nitrogen or argon gas either through a porous plug situated at the bottom or by a sub-lance introduced from the top. Similarly, instead of blowing oxygen only from the top or bottom alone it was introduced from both ends, thereby developing the hybrid processes of steel making to obtain advantages of both the top and the bottom processes of steel making in one, i.e., two-in-one.

All such processes wherein the gas phase is introduced from both top and bottom of the vessel are known as hybrid processes of steel making. Many call these combined blowing processes. Some major findings in steel making studies, in the 1970s primarily, gave impetus to the development of such processes. These were tried on the existing LD/LDAC/OLP/OBM/LWS and such vessels after modifying them suitably. Consequently a large number of such processes are available in the market. They have a variety of names depending upon the type of gas blown, whether from top or bottom, how it is blown, and the name of the steel company that developed it. The name given to such processes can be debated, but the term hybrid processes seems logical. However, the term combined blowing may be given when oxygen is blown both from top and bottom, and bath agitated processes if inert gas stirring is carried out from the bottom only, with top oxygen blowing.

1. In bath-agitated processes, bath stirring is caused by a neutral gas introduced below the metal bath either through porous refractory plugs or tuyeres fitted in the bottom refractories, or through auxiliary lances or pipes introduced from the top.
2. In combined blowing processes, oxygen is introduced from both top and bottom, bottom oxygen being protected by hydrocarbon shield, for refining, causing the necessary stirring for refining. The proportion of oxygen blown from each end varies from process to process.
3. Hybrid process is a term applied to any such mixed blowing process. Nitrogen or argon may be chosen judiciously as the inert stirring media separately or together. For experimental purposes the neutral stirring medium was introduced in the beginning from top through auxiliary pipes; it is now universally introduced from the bottom through porous refractory plugs or tuyeres.

The characteristics observed in these are:

1. A small amount of inert gas, about 3% of the volume of oxygen blown from the top, introduced from the bottom, agitates the bath so effectively that slopping is almost eliminated.
2. However, for obtaining a near equilibrium state of the system inside the vessel, a substantial amount of gas has to be introduced from the bottom.
3. If 20-30% of the total oxygen if blown from the bottom, can cause adequate stirring for the system to achieve near equilibrium conditions. The increase beyond 30% therefore contributes negligible addition of benefits.
4. However at 30% oxygen blowing from the bottom leads to formation of very dry slag and possibility of its ejection during refining unless it is also accompanied by lime.

5. The more the oxygen fraction blown from bottom, the less is the post combustion of CO gas, and consequently less scrap consumption in the charge under identical conditions of processing.
6. Blowing of inert gas from the bottom has a chilling effect on the bath and hence should be minimal. On the contrary the more the gas is blown, the more is the stirring effect and resultant better metallurgical results. An optimum choice, therefore, has to be made judiciously.
7. As compared to top blowing, the hybrid blowing eliminates the temperature and concentration gradients and effects improved blowing control, less slopping, and higher blowing rates. It also reduces overoxidation and improves the yield. It leads the process to near equilibrium with resultant effective dephosphorization and desulfurization and ability to make very low carbon steels.
8. As compared to bottom blowing, the bath agitation gives the advantages of bottom blowing in a top blown process, to the extent of gas used for purging.
9. Most of these processes were put into practice in the period from 1978-1982, changing the existing vessels of sizes from small ones to 340 metric ton capacity, and almost all proved successful. These initial trials led to the development of the means of introducing inert gas through the bottom.
10. The extent of gas purging from the bottom of the vessel or combination blowing, in consequence, results in varied stirring effects or varied mixing times — a measure of the time taken for homogenization of the bath. All these hybrid processes, with reference to mixing times, e.g., in between the LD in its classical form on the one extreme, with relatively highest mixing time, and OBM on the other, with minimum mixing time as shown in Figure 20.1. There is a whole range of processes that have been successfully developed at various major steel plants the world over in the hybrid category. The variations in these can be seen in the form of the following parameters:
    - (*a*) What is blown from the bottom, inert gas or oxygen?
    - (*b*) How much inert gas is blown from the bottom?
    - (*c*) At what stage of the blow is the inert gas blown, all through the blow, at the end of the blow, after the blow ends, and so on?

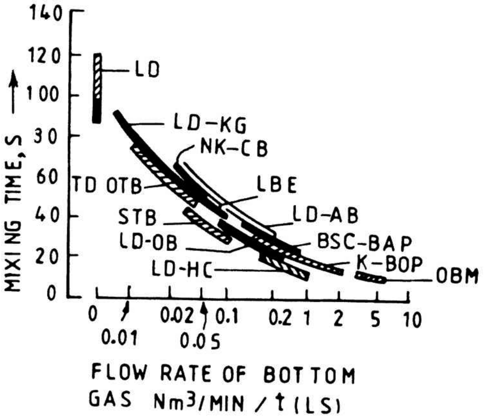


FIGURE 20.1. Effect of basal gas flow rate on time for complete bath mixing[32].

(*d*) What inert gas is blown, argon, nitrogen or their combination?

(*e*) How is the inert gas blown, permeable plug, tuyere, etc.?

(*f*) What oxidizing media is blown from bottom, oxygen or air?

(*g*) If oxygen is blown from the bottom as well then how much of the total oxygen is blown from the bottom?

(*h*) How is the bottom oxygen shrouded?

(*i*) Whether the hydrocarbons are used for heating of scrap? And such factors in detail.

20.2. Hybrid Process Spectrum and Their Characteristics

Because of the variations adopted as stated above, the hybrid processes are varied and can be classed as shown below:

Process Class	*Representative Process*
1. Classical LD with only top blowing and lumpy charge from top	LD
2. LDAC/OLP process with top oxygen and lime powder from top	LDAC/OLP
3. LD with additional bottom gas purging	LBE/LD-AB/LD-KG

4.	Oxygen blowing from top and bottom	BSC-BAP/LD-OB
5.	Oxygen blowing from top and bottom with lime powder from top	LD-HC/STB-P
6.	Oxygen blowing from top and bottom and lime powder from bottom	K-BOP
7.	Oxygen and lime powder from bottom only	OBM-P
8.	Oxygen and protective hydrocarbon (classical OBM)	OBM
9.	Oxygen bottom blown plus oil/gas injection for preheating	KMS-KS/OBM-S
10.	Oxygen and inert gas from top only	LD-GTL/AOB/AOD
11.	Oxygen from top only but with jet pulse	LD-PJ
12.	Oxygen from top only with circular motion of the lance	LD-CL

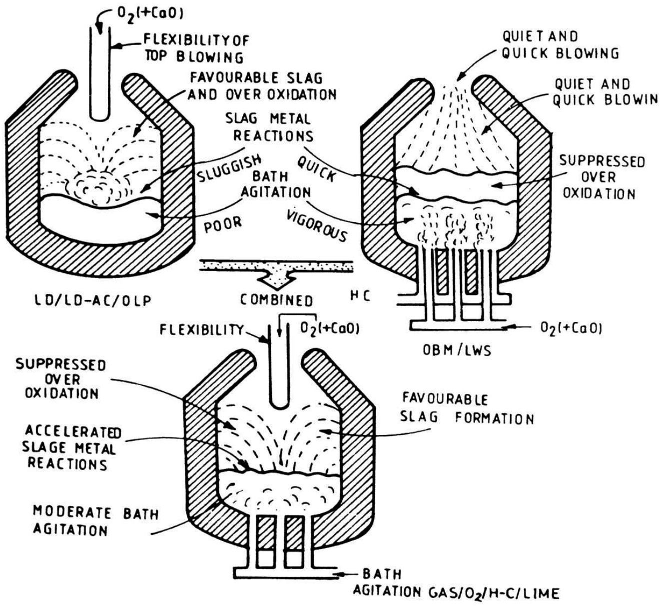
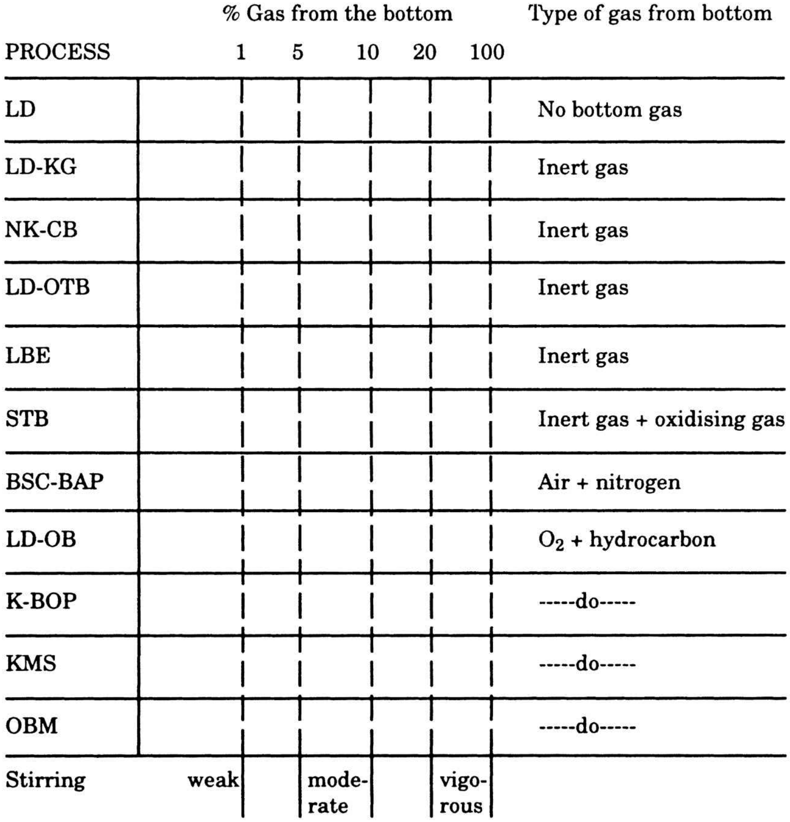


FIGURE 20.2. The scheme of top and bottom hybrid blowing process.

Out of all these processes, it is easier to convert classical LD into a bath-agitated process typically represented by the LBE design. This is most popular because of the following reasons:

1. Existing LD finds it necessary to improve upon its drawbacks without much alterations and capital expenditure.
2. Availability of porous plugs and ease with which it can be fitted in an LD vessel.
3. A general reluctance to adopt complicated bottom oxygen blowing with shrouded hydrocarbons.

The amount of top and bottom blowings and their types are shown in Figure 20.3 at a glance [32].



PROCESS	% Gas from the bottom 1	5	10	20	100	Type of gas from bottom
LD						No bottom gas
LD-KG						Inert gas
NK-CB						Inert gas
LD-OTB						Inert gas
LBE						Inert gas
STB						Inert gas + oxidising gas
BSC-BAP						Air + nitrogen
LD-OB						O_2 + hydrocarbon
K-BOP						-----do-----
KMS						-----do-----
OBM						-----do-----
Stirring	weak		moderate		vigorous	

FIGURE 20.3. The variety of hybrid processes along with the amount of basal gas injected [32].

20.3. Bottom Gas Purging Design

Because it was the easiest to fit and operate, the bottom gas purging came into operation early. A variety of devices have been developed to introduce inert gas from bottom in an otherwise classical LD vessel; these are shown in Figure 20.4. These devices are simple and rugged enough to last for almost the life of the rest of the wall lining of the vessel during refining. They allow varying of the stirring times and its extent by controlling the gas flow. Even in emergency if it fails to operate, the vessel can still be operated in the form of classic LD style to continue steel production. The supporting pressure required to purge gas through them is usually in the range of 2-3 bars only. As a rule the mass flow permeable element is maintained at around 4 m^3/min to inject 1.7-2.0 m^3 of inert gas per metric ton of liquid steel. Even if

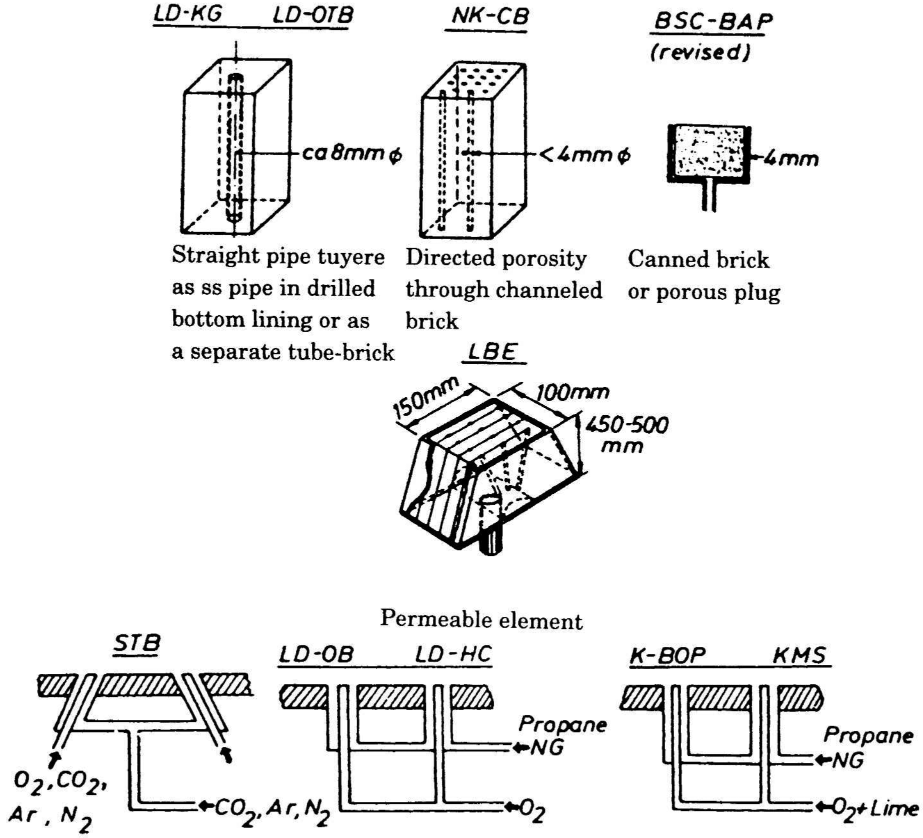


FIGURE 20.4. Schematic representation of methods used for bottom injection in hybrid processes [32]

one-third of the elements are damaged during the campaign, the operation is still not adversely affected. If the bottom flow rate requirement is high, then it is better to adopt oxygen tuyeres rather than these permeable elements. Similarly if lime powder is to be injected from the bottom, then the permeable elements are of no use and oxygen tuyeres are the only alternative.

20.3.1. Hybrid Blowing Design Considerations

In hybrid processes, the top and the bottom blowing need to be adjusted so as to obtain maximum stirring without any adverse effect on the refining reactions and yield. The top and bottom blowings are designed to obtain optimum conditions of overall blowing. The factors that affect it are:

(*i*) The inclination of the nozzle to the vertical axis in a multi-hole lance; number of nozzles as well

(*ii*) Usual bath diameter inside the vessel

(*iii*) Usual height(s) of the lance in blowing position

(*iv*) Oxygen pressure used for blowing

(*v*) Position, size, and number of purging gas inlets at the bottom (of oxygen if blown from bottom)

(*vi*) Pressure of bottom gas used for blowing

In fact, points (*i*), (*ii*), (*iii*), and (*iv*) decide the jet depression area during blowing, i.e., the area actively associated with oxygen-metal interaction. Factors. (*v*) and (*vi*) decide the gas-metal plume size formed as a result of bottom purging. The plume must interact with the jet depression area to kinetically expedite the reactions by matching each other. This match has to be established through trial and error to decide the final matching parameters of the above factors. This should assure optimum operating conditions of the vessel, and from there, the results.

20.4. Metallurgical Superiority of Hybrid Blowing

The processes have been developed to obtain the combined advantages of both LD and OBM to the extent possible. Therefore the metallurgical performance of a hybrid process has to be evaluated in relation to these two extremes, namely, the LD and the OBM. The parameters on which this can be done are:

(*i*) Iron content of the slag as a function of carbon content of bath

(*ii*) Oxidation levels in slag and metal (*iii*) Manganese content of the bath at the turndown

(*iv*) Desulfurization efficiency in terms of partition coefficient

(*v*) Dephosphorization efficiency in terms of partition coefficient

(*vi*) Hydrogen and nitrogen contents of the bath at turndown

(*vii*) Yield of liquid steel

It should be kept in mind that the performance of each process at a particular shop is different from a similar performance at another. But the general performance data can be chosen for the sake of comparison and evaluation. Broadly, three hybrid processes have been selected as typical examples for comparison with the two extreme ones, such as LD and OBM. For the sake of brevity they are referred to as follows:

(*a*) Classic LD operation as	LD
(*b*) Top oxygen blown plus bottom inert gas stirred	LBE
(*c*) Top and bottom oxygen but no lime powder	LD-OB
(*d*) Top and bottom oxygen with bottom lime powder.	K-BOP
(*e*) Classic oxygen bottom blown	OBM

The performance of these three hybrid processes with reference to the performance of LD and OBM are depicted in Figures 20.5 to 20.9.

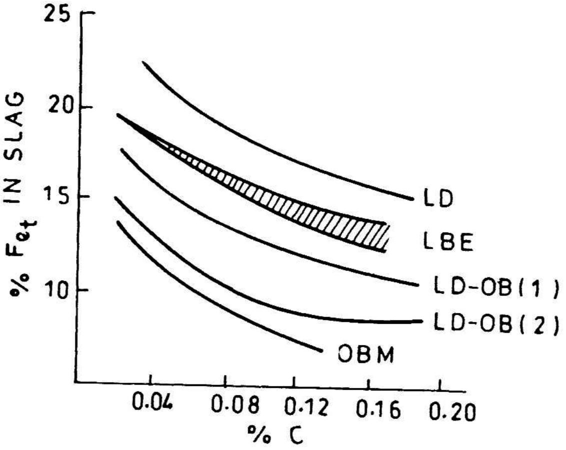


FIGURE 20.5. Variation of iron content of the slag as a function of carbon content of the bath. The amount of gas blown from the bottom is more in LD-OB (2) than in LD-OB (1), hence the relative position above OBM line. The shaded area for LBE again depends on the actual amount of gas blown in [32].

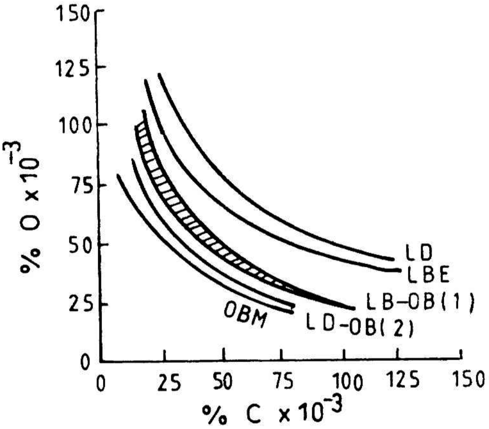


FIGURE 20.6. Variation of carbon and oxygen in the bath. As agitation increases, the oxygen decreases for the same carbon content [32].

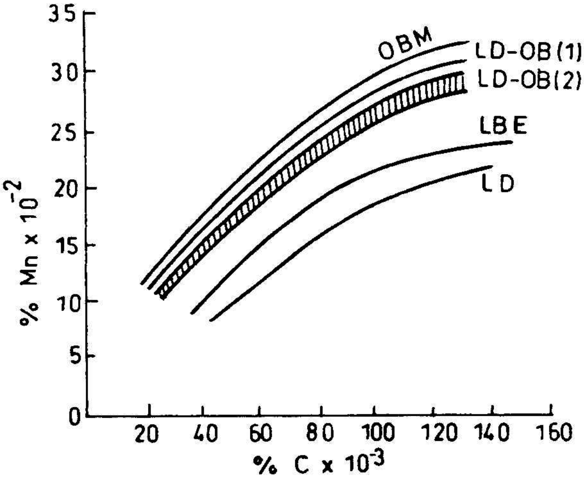


FIGURE 20.7. Relation between C and Mn at the end of the blow [32].

All these figures are reproduced from a good review of such hybrid processes. It is clear that for the same carbon content of the bath, both the iron content of the slag and the oxygen content of the metal (both are indicative of oxidation potential of the system) decrease as the stirring increases from LBE to LD-OB to K-BOP. It is apparent that the hybrid processes perform in between the LD on one side and OBM on the other. Similarly the manganese content of the bath at the end of the blow increases from LBE to K-BOP. The generally less over-oxidized slag and metal tend to help desulfurization as bath agitation increases, but the real effect can only be seen if lime is blown from the bottom. The LBE type processes do not indicate any appreciable desulfurization because mere stirring does not help it much. If sulfur is to be removed, then it is better to adopt the K-BOP type process.

Dephosphorization is a more complex reaction and is affected by many factors that vary in these processes, e.g., top lance height, purging gas flow rate, lime injection from bottom, and so on. The inert gas purging rate is also varied during the blow with respect to the lance height to achieve the desired dephosphorization. In one set of combinations, vigorous stirring in the beginning is followed by low stirring when decarburization is at its peak, and then again vigorous stirring at the end when the temperature favors phosphorus removal. Alternatively, low stirring in the beginning and more vigorous towards the end is to be followed by a mere rinse after the blow is over to remove phosphorus.

The more the use of hydrocarbons for shrouding the bottom oxygen jets, the higher is the hydrogen content of the bath. Argon stirring does help in reducing it because of the purging effect, wherein the dissolved hydrogen diffuses in the rising argon bubbles.

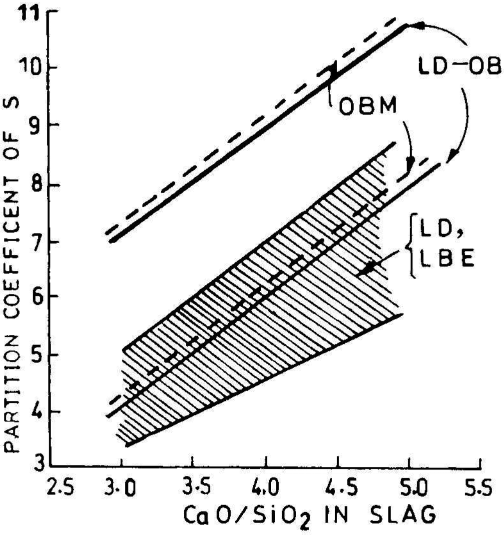


FIGURE 20.8. Sulfur partition as a function of basicity [32].

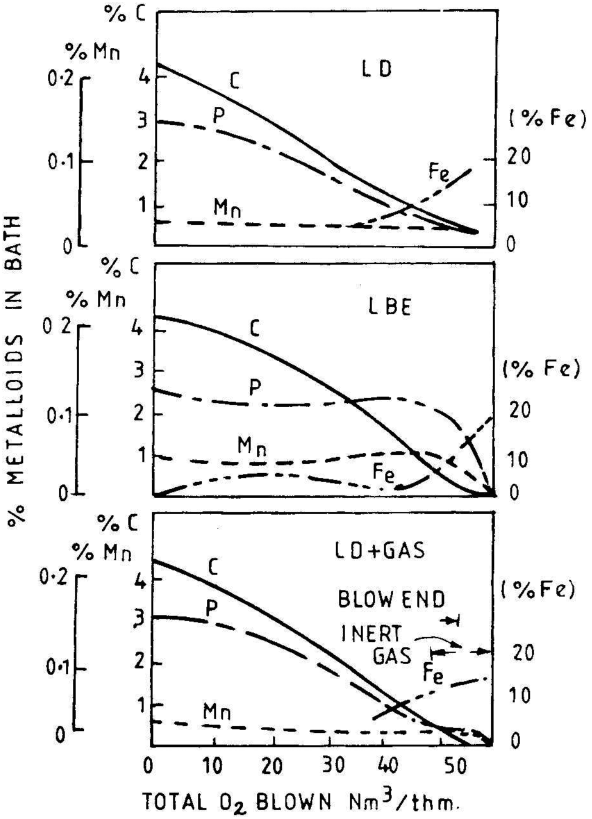


FIGURE 20.9. Effect of bottom gas flow on refining. It indicates the strong influence of stirring on P removal [32].

The nitrogen content of the steel bath in almost all hybrid processes is practically the same except when nitrogen is used as a purging media and the argon rinse is not given in the end.

As one moves from top blowing to any hybrid blowing there is a net decrease in slopping and oxidation potential of the slag and the bath, hence the yield increases by 0.3 to 1.0% depending upon the operational details.

The refining data obtained in such hybrid processes is summarized in Table 20.1. It is obvious that the performance of LBE type processes is closer to LD, and that of K-BOP type processes is closer to OBM, with the LD-OB lying in between.

Table 20.1 [32]

Typical Hybrid systems and their metallurgical parameters along with those of LD and OBM

Parameter	*LD*	*LBE*	*LD-OB*	*K-BOP*	*OBM*
Top blown oxygen flow rates Nm^3/min/t	3.0-3.5	3.3-3.5	2-4	1-3	-
Bottom blown gas flowrates Nm^3/min/t	-	0.01-0.10	0.1-0.5	1-1.5	up to 5.5
Fet % at 0.05%C	20	18	14-15	13	10
0% at 0.05%C	0.06	0.05	0.04	0.04	0.03
Mn -do-	0.14	0.18	0.21	0.22	0.30
N ppm -do-	20-40	20-25(A.F.)	18-22	15-30	15-30
H -do-	2	2	2-4	5	5
Minimum %C at blowend	0.05	0.02	0.02	0.01	0.01
(P)/P at 0.05% C	70	80 (S.B.)	80-90	110	120
(S)/S at 0.05% C	5.6	5.8	6.5	7.6	7.8
P_{CO} at 0.05% C, atm.	1.0	0.95	0.80	0.75	0.60
%CO_2 in gases	12	12	8-9	6	5
% Scrap at 1.5% Si iron	33	31-32	30	29-30	28
slopping	yes	minor	nil	nil	none
Yield %	standard	+ 0.3	+ 0.4	+ 0.5	+ 0.7
Mixing time, sec.	100	50-75	30-35	10-30	10

Note: A.F. means after flushing, S.B. means for softer blowing, neg means negligibly small, standard means that with which others are compared.

20.5. Future of Hybrid Processes

It can be said with certainty that the last OBM and classic LD vessels have definitely been built and no one is going to install them in their classic form hereafter. One of the other hybrid processes will be the only choice for new Greenfield installations. The choice may depend upon the cost and the products to be made. Any bottom oxygen blowing facility is going to cost more than one without it. A facility of mere argon blowing from the bottom may be found as the least costly. Plants particularly making thin flats, wherein carbon has to be lowered to a very low value, may prefer the K-BOP type process; for those making beams, channels, heavy plates, etc., the LBE type may be preferred for its lower cost.

For brownfield installation, particularly when open-hearth shops have to be replaced as a part of modernization, the OBM or its modified form that can be accommodated in the low-height-open-hearth sheds may be found suitable. If height is no problem, then the choice range is greater.

The existing LD shops can be readily converted into the LBE type process using argon purging as an additional facility at relatively less cost and easy-to-operate modification. This conversion has been implemented all over the world in the last ten years or so, also because of widely adopted classic LD plants all over the world in the decades of the 1960s through the 1980s.

The decarburization techniques using oxygen-argon injection are quite common, and as stainless steel production is rapidly increasing, this may become more popular.

20.6. Some Hybrid Plant Details [34]

20.6.1. LBE Process

The LBE process was developed jointly by ARBED and IRSID using permeable refractory block set in the bottom of the LD converter for blowing argon. The name LBE originates from the Lance Bubbling Equilibrium system. Soon the life of this plug could be brought at par with that of the main lining i.e., 500 heat then, thereby solving the problems of intermediate shutdown and repairs faced earlier.

At the Usinor-Denain, magnesia or chrome-magnesite plugs of 450-500 mm thickness with about 100 × 150 mm^2 area exposed to metal bath is used on a 65 metric ton converter. The flow rate can be varied up to a maximum of 5 N-m^3/minute per element. This was considered adequate for obtaining desired metallurgical results.

The campaign was later extended to a 150 metric ton LDAC converter at Esch-Belval Works where a special multi-hole double-flow oxygen lance was later designed and used to increase scrap rate without affecting production rate and product quality. The CO generated during refining was burned by this additional stream of oxygen. While maintaining a proper slag regime, the heat liberated by the combustion of CO was nearly completely transferred to the melt, and in this way up to 27% CO could thus be burned inside the vessel during refining. The top oxygen flow is milder by keeping the lance at a higher position. The top lance supplies oxygen to the gas and slag, which maintains equilibrium with the bath because of bottom purging by a neutral gas. If purging is not resorted to, heavy slopping occurred due to the use of the post combustion nozzle, which allowed 6 N-m^3 of oxygen per minute per metric ton when total blowing time could be as short as 10 min/heat.

The hot metal composition that was refined was

	Carbon		3.90%
	Manganese		0.28%
	Silicon		0.64%
	Phosphorus		1.65%
and	temperature	was	1328°C

and the results obtained on nearly 2500 heats are summarized in the following table [35]:

	LD-AC	*LBE*	
		Without post combustion	*With post combustion*
Hot metal kg/metric ton	786	698	664
Pig iron kg/metric ton	54	13	-
Scrap kg/metric ton	271	390	440
Iron ore kg/metric ton	6	4	1
Iron yield %	95.1	96.5	96.5

The scrap proportion increased from 27.1% in old LD-AC to 39% by bottom purging, and 44% when post combustion was adopted. Similarly, the mean iron content of the first slag decreased from 8 to 5% with virtually no slopping and as a consequence the yield improved by about 1.4%.

The converter specific volume decreased to 0.6 m^3/ metric tons. Since the top lance height is raised by 0.5 meter, the nozzle life increased, skull formation decreased, and lime dissolution is complete without flux addition.

The bath is purged even after the oxygen blow is over so that further desulfurization and decarburization are obtained. For Thomas hot metal the final results of phosphorus 0.011%, sulfur around 0.005%, and product of carbon and oxygen as 0.0018 with carbon as low as 0.008%, have been achieved.

Steels could also be produced by the catch-carbon technique without any adverse effect on desulfurization and dephosporization. Ferro-alloy consumption reduced by 1 kg/metric ton because of lower oxygen potential of the bath. The lime consumption also was decreased because of improved partitioning of phosphorus between slag and metal.

20.6.2. LD-KG Process

In order to improve bath agitation in LD, the Kawasaki Steel Corporation developed this process of inert gas purging. The flow rate of gas was very low, 0.02 N-m^3/t/min. Encouraged by the results of preliminary trials they adopted this technology to three out of six, 180 t LD converters at its Mizushima Works, and three out of five 150 t LD converters at its Chiba Works. The new technology has resulted in a decrease in oxygen consumption by about 1.7 m^3/t and increased the yield by about 0.5 to 0.7%. Except manganese, other distributions such as of P and S were altered favorably.

20.6.3. LD-AB Process

Nippon Steel, at its Yawata Works, converted a 70 metric ton LD vessel to the bath agitated process. Argon was bubbled at 0.013 N-m^3 per min per t. The results are similar to that of OBM if carbon content is higher than 0.06%. For lower carbon contents the results are more similar to LD. Similar results are observed for P and Mn. The critical carbon contents under which the OBM results are not obtainable can be decreased by increasing the purging gas flow rate, but blowing was not smooth if the flow rate exceeded 0.25 to 0.5 N-m^3 per minute per metric ton.

Similar trials carried out at Kobe Steel indicated that an increase in flow rate steadily improved dephosphorization for low carbon steels, but an optimum was observed at 0.05 Mn^3 per min per metric ton for high carbon steels.

20.6.4. LD-OB Process

Nippon Steel converted 179 and 320 t converters at Yawata and a 340 t converter of Oita to the LD-OB process. Nearly 0.3-0.8 N-m^3 per minute per t of oxygen was blown from bottom tuyeres with protective hydrocarbon gas. Superior phosphorus distribution is obtainable in this process. It shows that oxygen blown from both the bottom and the top is more beneficial than mere argon stirring in obtaining effective dephosphorization.

20.6.5. STB Process

Sumitomo Metal Industries developed a process for blowing oxygen from both top and bottom. Ten double tuyeres were provided on a 250 t LD vessel at the Kashima Steel Works. CO_2 and argon was provided as protective gases. For very low carbon steels argon was used instead of CO_2. It was observed that after a critical amount of oxygen was blown from the bottom, the problem of spitting became serious. This limiting flow rate of oxygen from the bottom was 0.1 to 0.2 N-m^3 per min per metric ton when flow rate from the top was 2.3 N-m^3 per min per metric ton.

Theoretical calculations indicate that for their design mixing time for obtaining good results is 70 sec as against 100-125 sec in LD and 10 sec in OBM[34].

20.6.6. LD-HC Process

The original LD-AC, using lime-powder with oxygen blown from the top, was converted to a combined blowing process at Hainaut-Sambre in collaboration with C.R.M. The vessel was provided with 4-6 bottom tuyeres using CO_2 or natural gas as protective media. The proportion of oxygen blown from the bottom varied from 5 to 43% of the total. It was observed that total blowing rate, which was 2.6 N-m^3 per min. per t in original 40 t LD vessel, increased to 4.0 N-m^3 per min. per t in this LD-HC process [34]. Lime was fed from the top lance along with oxygen. The carbon-oxygen relationship at turndown achieves near equilibrium. Two slags were made with two blows, since high phosphorus iron was refined. The second blow was small and was carried out in three different ways:

1. True combined blowing, oxygen from both top and bottom.
2. Pure bottom blowing, no oxygen from the top.
3. Pure top blowing with nitrogen stirring from the bottom.

After the final blow, bottom stirring with nitrogen was also practiced. It was observed that for a given P_2O_5 content of the slag, the best results were obtained after nitrogen stirring. It was also noticed that the final blow as a pure bottom blow resulted in achieving closer equilibrium. The closeness to equilibrium with respect to phosphorus distribution in all these cases is shown in the table below:

Technique adopted	*Phosphorus deviation from equilibrium*
1. Blowing oxygen from top and nitrogen from bottom	1.67
2. Combined blowing, oxygen from top and bottom	1.46
3. Blowing oxygen from the bottom	1.33
4. Nitrogen stirring through the bottom after final blow	1.00
5. Pure top blowing (LD) for comparison	1.75
6. Pure bottom blowing (OBM) for comparison	1.10

It was, therefore, concluded that finishing the heat with pure bottom blowing is desirable.

The hydrogen content of the final steel increased as the proportion of oxygen blown from the bottom increased. It would be advantageous to restrict the bottom blowing of oxygen to obtain lower hydrogen contents.

The effect of variation of percentage oxygen blown from the bottom on the scrap rate was also studied. Maximum scrap could be added in the charge at round 20% oxygen blown from the bottom. The scrap rate was thus 6% higher than that in pure top blowing [36].

As the proportion of oxygen blown from the bottom increased from 3 to 11%, the iron content of the slag dropped from 25% in LD process to 13% for bath carbon content of 0.06%. The drop is more apparent for low carbon steels. The final short afterblow of inert gas decreased the oxygen to 0.035% for 0.025% C.

20.6.7. K-BOP Process

The Kawasaki Steel Corporation converted one of its 250 t BOP vessels into combined blowing with much higher flow rates from the bottom. Oxygen flow rate of 1.5 N-m^3 per minute per metric ton was employed and lime powder was fed along with oxygen from the bottom as in OBM. The high oxygen flow rate from the bottom was required to feed lime flux from the bottom. This combination did not result in spitting even though high flow rate was used from the bottom. The results on the whole were quite encouraging.

20.6.8. KMS/OBM-S Process [37]

This is more a modification of the OMB process to incorporate particularly a high scrap proportion in the charge, since by itself the OBM process has a limited scrap melting capacity. The modified versions in the form of KMS or OBM-S include the following actions[27]:

1. Preheating of scrap using bottom tuyeres as burners.
2. Modifying the vessel by incorporating top-side tuyeres in the vessel cone for oxygen supply to burn CO to CO_2 during refining.
3. Injection of carbon into the bath during refining to generate additional heat.

The first alternative can increase scrap rate by 10 to 20% depending upon the degree of preheat employed. Similarly the third alternative can increase the scrap input to values as high as 60-80% of the total metallic charge. These, however, have nothing to do with combined blowing. The second alternative only falls in the category of combined blowing and hence is being described here.

The OBM vessel is converted to the KMS/OBM-S design by providing tubes to feed oxygen and hydrocarbon both through the conical nose of the converter, in addition to the provision for bottom blowing, as shown in Figure 20.10. The process gases are piped to the furnace shell through multiple rotary joints and passages in the trunion pins and the trunion rings. Oxygen and oxygen-flux mixtures are kept usually on the idle side only, while hydrocarbons are piped through the drive side. The heat is carried out as follows:

After the previous heat is tapped, the tuyeres are protected from overheating by low flow of air or nitrogen.

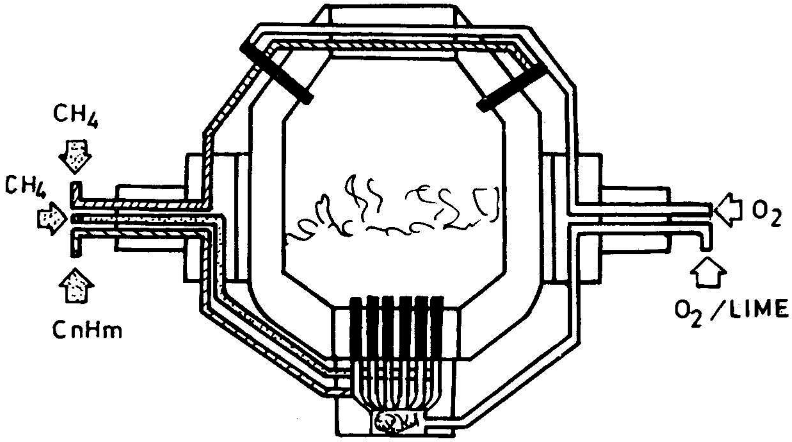


FIGURE 20.10. Schematic of KMS furnace [37].

Scrap is charged and the furnace is turned up.

Preheating of scrap commences with oil and oxygen from the bottom tuyeres.

Preheating is predetermined as per requirements and once it is over the oxy-fuel supply is put off, and the vessel is turned down and hot metal is charged. During the vessel turndown condition nitrogen is blown through the bottom tuyeres.

The blow is started by switching from nitrogen to oxygen from both the bottom and the top tuyeres. Lime is fed as decided previously.

The blow is stopped by switching over to nitrogen again after a predetermined blowing period.

Sampling, analysis, additions, etc., are carried out as usual.

Anything between 3-8 minutes of scrap preheating is usually adopted and this does not in any way alter the metallurgical advantages accruable from OBM process.

The process essentially utilizes the heat of CO gas available during refining by burning it inside the vessel. Merely through this modification the scrap rate may go up by about 6%, and with preheating it can go up additionally by about 15-18%. The applicability of this process can be assessed in terms of this advantage with reference to local conditions.

Very strictly speaking this process does not fall in the category of combined blowing, because blowing from both ends is not strictly used for refining purposes in this process. It may rather better be referred to as a modified OBM process only.

20.7. Hybrid Process at Tata Steel [24]

Tata Steel installed their first LD steel making shop in early 1982, and the second one in 1993, thereby replacing most of their open-hearths, except perhaps two.

As usual, their LD blowing strategy was the result of consideration of many variables such as Si, Mn, and P content of the molten iron, the desulfurization aimed at during refining, the final P content at turndown, final Mn and C at turndown, and so on. With nearly 1.6-2.0% Si and more than 1.0% Mn in the hot metal, as conditioned by the compulsions of blast furnace operation in those days, early fluid slag formation within 2-3 min of blowing was aimed at. This helped to form the slag-gas-metal emulsion inside the LD vessel early to dephosphorize the bath well in advance of carbon removal. In this scheme, in the last 25% of the blowing, only carbon was being eliminated and thereby effective control was exercised in obtaining the desired carbon at the turndown, including the medium and the high carbon by the -catch carbon- technique.

It is certain that dephosphorization has to be completed well before all carbon is removed, otherwise it would not be possible to remove it when the stirring of slag and metal is at its low intensity towards the end. This was possible because of high silicon irons, which on oxidation right in the beginning would ensure slag formation by its combination with lime and heat formation to raise the temperature. The use of a lance with fewer nozzles in a way helped because it had to be kept relatively at a higher height thereby making it a soft blow. A softer blow helps dephosphorization-contrary to decarburization. Use of flux, like dolomite, further facilitated early thin slag formation and was expected to erode the lining less because of slag being saturated with MgO. All these were beneficial for dephosphorization, no doubt.

It however resulted in a very adverse effect on the lining life. The thin slag led to excessive formation of emulsion, which at times even overflowed the vessel and caused ejections and loss in yield. But more than that, the slag was splashed on the vessel walls,and flowed down over the lining, thereby causing lining dissolution and consequent low lining life of the order of 160-180 heats for tarred dolomite lining. The basic reason of minimizing the lining dissolution by slag, by saturating it with dolomite flux addition, was thus totally negated by the adoption of this practice.

This entire strategy was altered by altering several parameters of the process thus:

(*i*) Dolomite charge-flux was entirely eliminated.

(*ii*) The hot metal silicon content was brought down to 0.7-0.9% and manganese to 0.1-0.2% because of alterations in blast furnace practice and adoption of external desulfurization.

(*iii*) Adoption of six- or more hole lance to increase the jet spread on the bath surface, thereby increasing the slag oxidation potential required for dephosphorization.

(*iv*) Introduction of bath agitation by argon injection from bottom permeable plugs, thereby achieving better slag and metal mixing and more close to equilibrium conditions. The vessel lining along with argon plug is shown in Figure 20.11.

All of these together achieved the required dephosphorization in time, i.e., ahead of decarburization. The dephosphorizing index achieved was better than that in LD in spite of the slag being very viscous in the earlier part. Phosphorus was, in fact, effectively removed only in the later part of the blow along with carbon. The earlier insistence of LD operators to obtain thin, oxidizing and basic slag right in the beginning, as a prerequisite for proper LD blow, was thus proved wrong. In fact this thin slag was the chief reason for low lining life in LD vessels. The viscous early slag, in fact, relatively acid as well, proved far beneficial for improving lining life. The Tata Steel has already obtained more than 1200 heats of lining life from tarred-dolomite lining, quite unheard of anywhere in the world. It also improved the yield by decreasing the slopping to almost nil. The viscous slag did not splash on the walls and did not erode it. The earlier theory of thin slag attacking the lining more was also thus proved correct. The wider spread of six-hole lance, oxidized the slag well and stabilized the phosphorus in slag in spite of it being less basic. The basicity of the slag continuously increased during the blow because of continued dissolution of lime with the progress of blowing, and hence dephosphorization continued and finally when adequate basicity was achieved and, the slag was thin and oxidizing

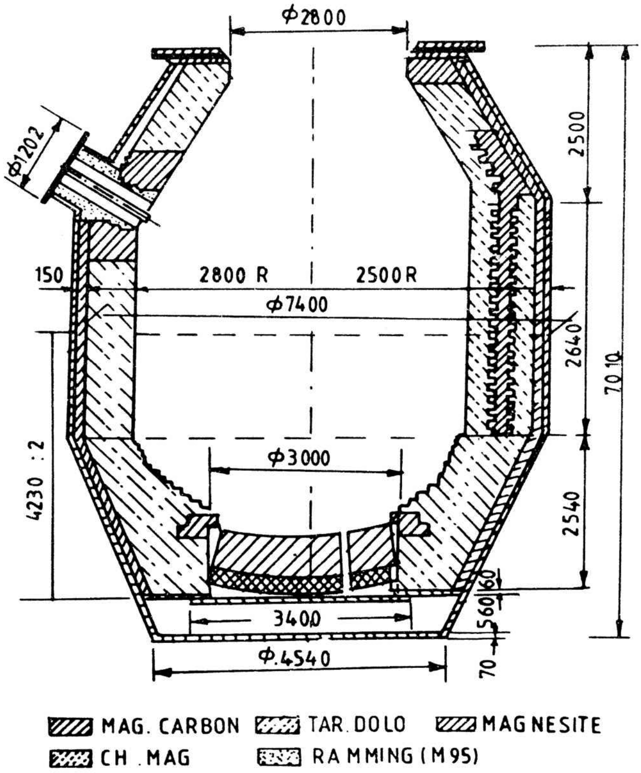


FIGURE 20.11. Various types of bricks used in different regions of the BOF at LD Shop 1 of Tata Steel [24].

in the later part of the blow that dephosphorization was completed as per expectation. The evolution of slag and bath composition during the blow in such a heat is shown in Figure 20.12 and Figure 20.13. The bottom stirring by argon helped the reaction kinetics of dephosphorization. This was shown earlier in Figure 20.9.

The drop in silicon in hot metal not only reduces the heat generated in the initial few minutes of the blow but also does not help early slag formation readily. The effect of lack of early slag formation is offset by the bottom stirring effect and over-oxidation of the slag because of the wider spread of jet as a result of more holes in the lance.

On the whole the slag basicity improved from nearly 3 in standard LD practice to 3.5 in the modified practice and the dephosphorization index

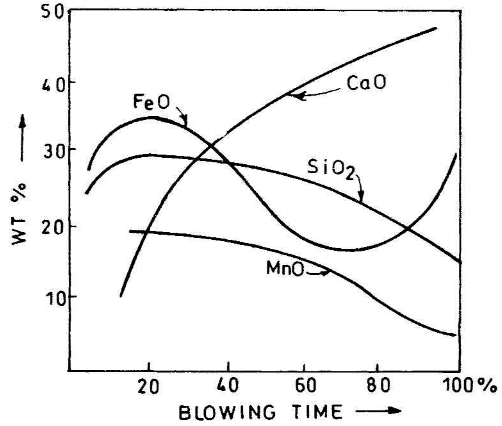


FIGURE 20.12. Evolution of slag composition with blowing time [24].

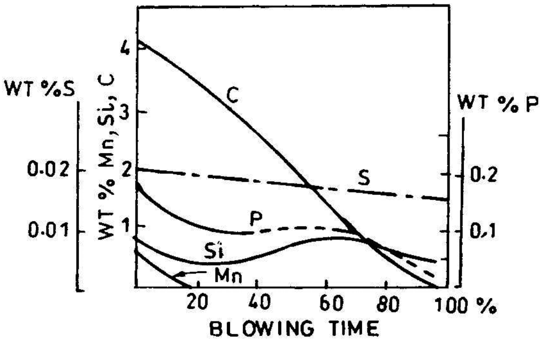


FIGURE 20.13. Evolution of bath composition with blowing time [24].

improved from 60-65 to 90-100 in the modified practice. The yield increased by about 0.5% and more importantly the percentage heats straight tapped without reblow increased to 65%.

All the above changes were not made at one time. The argon stirring was introduced in 1990. The six-hole lance replaced the earlier three-hole lance in 1993. The hot metal silicon fell slowly from 1.5% in 1985-86 to 0.75% in 1996 because of the changes in blast furnace practice and adoption of external desulfurization to deal with the sulfur problem. Consequently, the turndown phosphorus continuously fell from 0.03% in 1984 to 0.012% in 1996. The manganese ore burden in the blast furnace decreased to almost nil and thus the hot metal manganese content fell from 0.5% in

1984 to 0.1% in 1996. The elimination of manganese from blast furnace burden and lower operating temperatures of blast furnaces, coupled with external desulfurization, led to obtain 0.01% sulfur at turndown. The frequent earlier slopping came to almost nil in 1996. The lining life increased from around 275 heats in 1984 to over 1200 heats in 1997.

The blow profiles of the hybrid process using a six-hole lance for hot metal charge above 0.8% Si and that below 0.8% Si are shown in Figure 20.14 to illustrate the way the blowings are carried out and how the lance height is adjusted during the blow. This operation has been found to be much smoother when compared with that using a three-hole lance earlier.

20.8. Concluding Remarks

The hybrid steel making processes are essentially the modified versions of either the LD or the OBM processes in which the flexibility of independent control of decarburization and dephosphorization by merely raising or lowering the lance at constant oxygen pressure is not at all sacrificed. Nevertheless the definite advantage of bottom blowing for adequate mixing of the bath is also taken care of. These processes are therefore a happy marriage of both the top and bottom blowing processes. Consequently the slag-metal

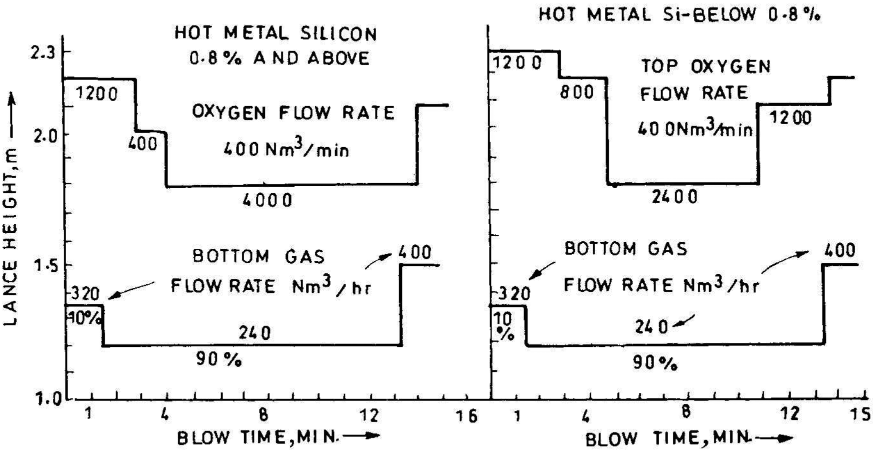


FIGURE 20.14. Blow profiles of hybrid vessel at Tata Steel using six-hole lance. This is much smoother when compared to using a three-hole lance [24].

and the gas-metal reactions of refining are closer to equilibrium. The net result is decreased oxygen content of the slag and the bath (as reflected by the iron content of the slag in these processes), decreased ferro-alloy consumption, increased yield and practically slopping-free operation. Both the desulfurization and dephosphorization are more effective.

The proportion of scrap in the charge that can be accommodated in a steel making process, if economical in a given situation, might make the hybrid process choice certain because of the extra scrap melting possibilities in such hybrid systems like the KMS/OBM-S. The necessity for any additional scrap melting facility, as was necessary in earlier SMS designs and adopted in the form of an open-hearth or an electric furnace, is no longer valid with the advent of these special hybrid systems.

In the last couple of decades, experience in running these varieties of hybrid systems has led to the accumulation of vast data that can now form a basis for designing these systems per the local needs. The changeover from the existing LD or the OBM or LDAC and such vessels can be carried out with enough perfection and satisfaction of the users. Within a few years after their inception, more than one hundred vessels were in regular operation; from then on everyone was slowly converting their existing vessels to one of these hybrid designs. The Tata Steel has converted both of their LD shops to hybrid blowing using the LBE system and their performance has been extremely satisfactory.

The Tata Steel has retained two of their open-hearths of 200 t capacity in SMS 3 in spite of installation of the LD shop by dismantling the remaining ones in SMS 3. This takes care of extra scrap available in the circulation inside the plant since even the modified LD practice would not be able to consume all this home scrap. But these will be phased out by 1997 totally.

In 1990, the SAIL converted the classic LD into a hybrid operation of LBE design by converting a 130 t vessel at the Bokaro Steel Plant. This vessel had a detachable bottom, and hence the change could be readily brought about. It then also converted the 50 t vessels of the Rourkela Steel Plant. The performance has been satisfactory.

The age-old steel industry was revolutionized for the first time by the introduction of Bessemer process over one and one-half centuries ago. The advent of oxygen steel making in the 1950s revolutionized it again from the point of view of thermodynamic aspects of refining. The hybrid blowing technology has revolutionized it yet again for the third time from the point of view of thermodynamics and kinetics of refining, since now it would lead to attain virtually the equilibrium conditions in steel making without sacrificing the pace of production and quality of the product, along with flexibility of operation with respect to proportion of scrap and hot metal in the charge.

CHAPTER 21

ELECTRIC FURNACE PROCESS VIS-A-VIS BOF STEEL MAKING

In This Chapter

- Introduction
- Characteristics of Sponge Iron
- Charge Proportion
- Advantages of Hot Metal vs. DRI as Charge
- Process of Refining
- Energy Considerations
- Oxygen Lancing and Process Dynamics
- DRI Melting

21.1. Introduction

In general, flat steel products have long been the domain of traditional integrated steel plants adopting the classical B.F./BOF technological route. This has now been challenged successfully by the DRI-EAF technological route since the last couple of decades. The electric furnace, both arc and induction, are no longer associated with only alloy or special steel making and that, too, on a relatively small scale. The evolution of competitive alternative routes of iron production, particularly the solid state reduction processes, even on a large-scale commercial level, are basically responsible for evolution of this alternative route. This new route has several advantages as well over the electric steel making from all scrap charge. It is equally as free of tramp elements as the B.F. route is. The development of a variety of DRI technologies to produce DRI, HBI, etc. from different raw materials as per local conditions, in recent times and their excellent energy efficiencies are equally responsible for the success of this alternative route of steel production. The alternative route is very flexible with respect to the quality of raw materials. It is equally flexible in adopting any secondary steel making technologies for forward integration for quality steel production in competition with any such alternatives. In the absence of an in-house source of iron because of some serious breakdown or any such thing, the B.F.-BOF route comes to a standstill, whereas the electric furnace DRI route can continue steel production on purchased DRI from external source or an on cent-percent scrap charge without any problems.

A large number of steel plants using one or the other DRI-making technology, along with electric arc or induction furnace for steel making, have been successfully set up and operated in the last few years in India, particularly since the economy was deshackled from the socialistic monopoly in 1991. The typical examples are Usha Alloys Plant at Adityapur in Jharkhand, Ispat Industries Plant at Dolvi in Alibagh District of Maharashtra, Essar Steel Plant near Surat in Gujrath, and so on.

These plants usually have annual capacities in the range of 0.5 to 3.0 M metric tons. A new variety of integrated steel plants using coal-based DRI and an induction furnace have also proved equally economical, at least, under Indian conditions, to meet local long product demand of angles, channels, constructional bars, et cetera, of ordinary grades of steel. They are successful because of their small scale of production in the range of up to even only one hundred thousand tons annually. This popularity of the DRI route is essentially due to the paucity of scrap everywhere, its prohibitive costs, and its poor quality.

The popularity of such alternative strategies of steel production using electric furnaces has increased many fold because of its acceptance of liquid pig iron or solid pigs as charge material almost in any proportion for producing any steel. This flexibility surpasses any other, including the traditional route, of steel making par excellence. The electric furnace thus has evolved into an effective primary process of steel making to be coupled with any secondary processing as per requirements.

In the present scenario, therefore, the electric arc furnace is very well adopted, besides using DRI of any form, with traditional B.F. for making a variety of steel, using any secondary steel making technology along with it. Herein lies the importance of why it should be studied in greater detail.

Continuous efforts are being made to improve upon the specific electrical energy consumption per metric ton of steel and shortening of the tap-to-tap time to improve upon its productivity. For this the Foamy slag practice, oxygen blowing from top and side, adoption of Jet-Box or Coherent-Jet (Co-Jet for short), coke-breeze burner, hot heel, and such modifications of steel making practices have become necessary. Similarly, design alterations such as use of water-cooled panels, UHP transformers, current conducting electrode arms, eccentric bottom tapping, level 1 to 3 automation, et cetera, have been incorporated for better performance of the furnace. Continuous on-line exhaust gas analysis has been developed to improve automatic control. The current UHP facility provides nearly 1.5 MW/ton of steel charge to expedite melting. All these have been able to reduce the tap-to-tap time to nearly 35 minutes and electrical energy consumption to less than 350 KW/ton and graphite electrode consumption to less than 1.2 kg per metric ton of steel made.

The adoption of secondary steel making technology has shifted much of the fine refining part from the electric furnace to the LF and/or vacuum metallurgy devices leaving only melting, heating, and major macro refining to the electric furnace.

These electric furnace–based integrated steel plants now can produce deep drawing (DD), extra deep drawing (EDD), special deep drawing (SDD), special wire rod, special bar, Interstitial Free (IF), electrode wire grade, steel for petroleum pipes (API), stainless of all varieties, and such high-quality steel with ease and at competitive economies.

Because of all these reasons, the applicability of electric furnace steel making continuously increased and is continuously increasing its percentage share in the total steel production all over the world. The net effect of all these has been that the share of electric process steel making in the total world steel production steadily increased from a mere 0.2% in 1910 to 8% in 1950, 20% in 1980, and 33% in 1995, and it continued and still continues upwards. It presently contributes nearly 34% of the total world steel production and is expected to increase over 50% by the year 2020. This has already been shown in Figure 2.1 earlier.

21.2. Characteristics of Sponge Iron

Sponge iron or DRI or HBI or any such material is quite different in its chemical and physical properties, and as a result its performance in the steel making furnace is quite different from that of scrap. These peculiar characteristics have already been dealt with earlier in Section 14.5.

The DRI is specifically low in residual elements such as Cu, Sn, As, Zn, Cd, etc. It means that it can be blended with scrap containing these impurities suitably to reduce their extra-presence in the charge and thereby finally in the steel. It is otherwise a headache in reusing any such scrap for steel making because of the stringent specifications of such tramp elements in the final specifications. This problem is more serious in advanced countries, and there these residuals are strictly under surveillance.

Carbon content of DRI can be controlled to suit specific requirements of any practice, although it costs more to get carbon in DRI. When it is to be melted under oxygen blowing conditions, it is necessary to have extra carbon in DRI. This is at times as high as 2.0%.

The DRI quality with respect to the metallic content of iron and degree of metallization entirely depends on the original iron ore quality used for making DRI and the process parameters. Generally, 92% metallization is considered good and is achieved if gaseous hydrocarbon is used for its production. In the coal-based DRI production, this comes down to 86%–88% only. The balance of the iron oxide is left in the DRI as iron oxide and it may be equivalent to nearly 0.6% to 1.2% oxygen in the form of unreduced iron oxide. The variation of metallic iron content of DRI as a function of original iron content of the ore at fixed degree of metallization is shown in Figure 21.1.

The carbon in the DRI reacts with this oxygen and maintains reducing conditions around the DRI particle or else the DRI oxidation can be a serious issue during melting in an arc furnace under oxidizing conditions. The melting and refining of DRI in an electric furnace goes on side-by-side as a continuous process. It is therefore advantageous to have these two in certain balanced proportion in the DRI to help the steel making process.

1. The phosphorus content of the DRI should preferably be below 0.05% and sulfur below 0.03%. These help improve the steel making efficiency.
2. The DRI can be used as a substitute of scrap up to 15%–20% of the total charge. However, 100% DRI charge is also a distinct possibility

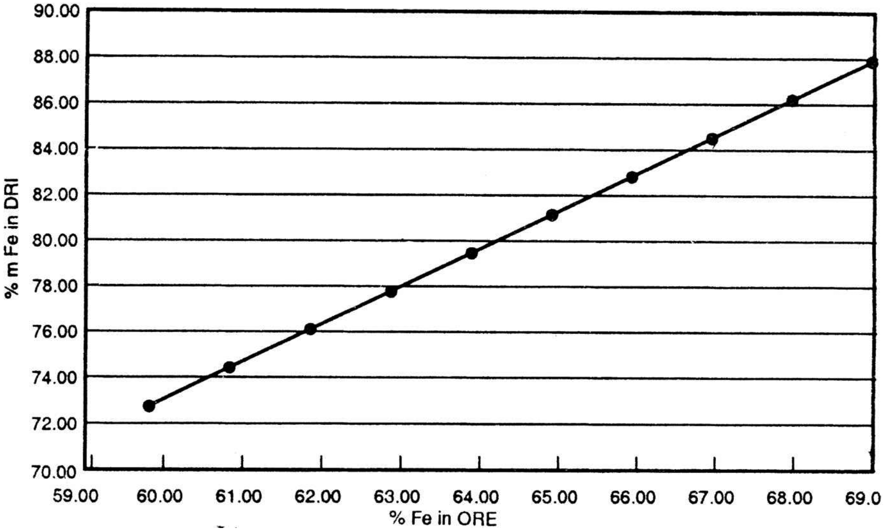


FIGURE 21.1. % Fe in iron ore vs. maximum attainable %Fe in DRI at 92% metallization.

in high-powered arc furnaces. This, however, needs facilities for its continuous charging to avoid excessive oxidation by furnace gases.

3. The volume of slag produced in the furnace using only DRI or DRI and scrap as charge shall depend on the amount of gangue introduced in the furnace via DRI. This volume will increase with increasing proportion of gangue in the DRI.
4. In a continuously DRI-charged furnace, the rate of charging and electric power supply and /or chemical fuel supply put together should match each other for effective melting and finishing of the heat. The recommended rate is 27–30 kg/min/MVA power input, and the material is fed at an angle of 35–50 degrees to the horizontal bath surface.
5. Charging of hot DRI has been made possible to improve the thermal efficiency. DRI up to 650°C temperature has been successfully charged in commercial practices.

21.3. Charge Proportion

Modern electric furnaces can take any proportion of scrap, DRI, and hot metal as the charge, depending upon the availability, need, and compulsions of the particular practice. The role of the electric furnace cannot be surpassed by any other steel making process. If the proportion of DRI in the charge is more than 20%–30% or if the design of the furnace setup is such that the DRI is always added continuously, in either case it is added through the hopper continuously. In order to make the DRI addition effective and prevent any excessive oxidation of DRI, it is added below the bath level, and for this 20%–30% of the charge from the previous heat is maintained in the furnace as hot heel. The DRI is continuously added as and when and in whatever amount is required.

It took a few decades for the electric furnace to come to the center stage of steel making in this way. The process had to undergo the ups and downs of historical developments, as did many other such technologies. But it survived all this and finally established itself as a competitor technology for steel making, particularly for conversion of even low-phos hot metal.

21.4. Advantages of Hot Metal vs. DRI as Charge

It must be clearly understood that the use of hot metal and refining by oxygen gas make the steel making process a autogenous one, wherein no external fuel is required to carry out the steel making operation (Section 4.4.4), except for mechanical operations involved therein. In fact, extra heat is available for melting some proportion of solid charge along with it. Since the oxygen refining of hot metal is very fast, the productivity of such shops is very high. When DRI is used as a source of iron in place of hot metal, to derive its other advantages and since thermal energy is required to melt it, the productivity of the shop comparatively falls drastically. The production of DRI is cheaper as compared with hot metal production using a blast furnace because it uses cheaper fuels for its reduction. A judicious combination of the two as a charge material, in EAF, needs to be evolved to take advantage of the use, both while attaining maximum productivity. This is precisely the underlying principle of the process of EAF steel making, in its modern modified form, in which the advantages of both these routes are sought at the optimum.

The proportion of hot metal and DRI in the charge and its influence on power consumption, tap-to-tap time (productivity), and oxygen consumption should be established in a given plant to work out the best proportion. These are shown in Figure 21.2 and Figure 21.3.

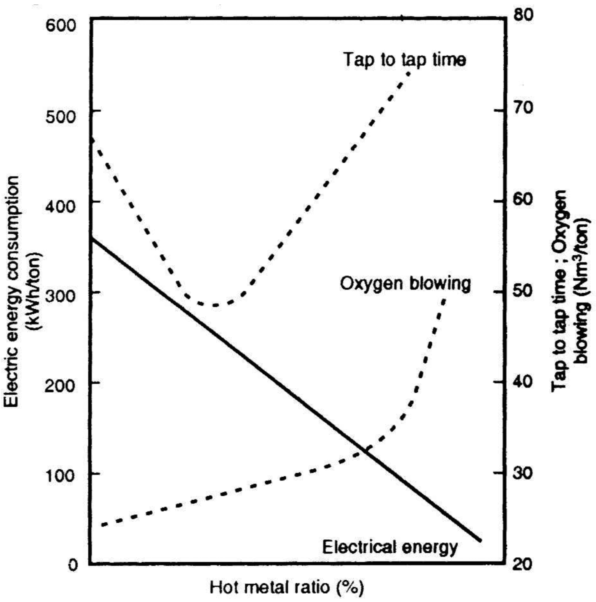


FIGURE 21.2. Shows the effect of hot metal proportion in the charge on power, oxygen consumption, and on tap-to-tap time.

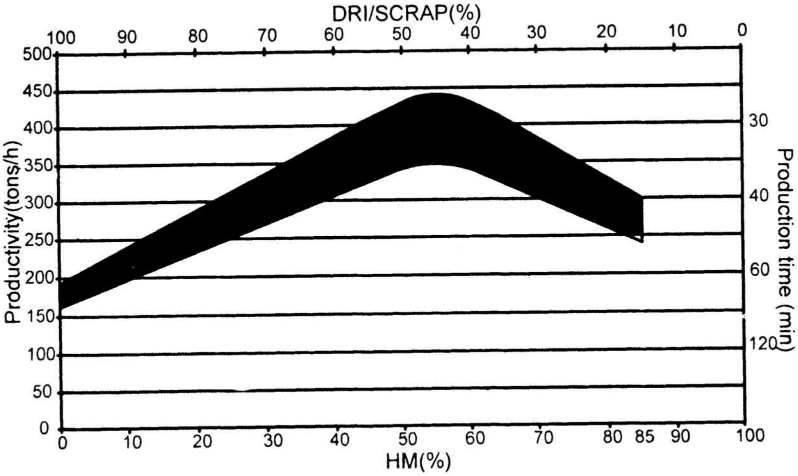


FIGURE 21.3. Shows the variation of hot metal proportion on productivity of the shop.

21.5. Process of Refining

The electric furnace is best suited when only DRI is to be converted to steel. The choice is between arc and induction furnaces. The process of conversion of DRI to steel is more a melting process to eliminate the gangue contained in the DRI, and that refining plays a small role therein. Both of these furnaces are otherwise ideal for melting the DRI because they do not have any oxidizing atmosphere of their own. In India induction furnaces are widely used for small-scale production of steel to meet local needs, whereas arc furnaces are universally used for large-scale production.

21.5.1. Current EAF Practices

For large-scale steel production in terms of millions of metric tons per annum in the form of an integrated steel plant blast furnace, DRI plant, preferably gas-based reduction plant like Midrex-Process, are coupled with EAF to first produce iron and then steel. The EAF is further coupled with a ladle furnace and possibly a vacuum treatment plant to produce quality steels for both flat and long products with equal ease.

The EAF process in this case is a combination of BOF process and conventional scrap/DRI melting and refining process. In short it is a hybrid process. The overall efficiency of this process lies in carrying out the two processes at their optimum. It means the exothermic heat of refining of molten iron by oxygen even in this case is fully utilized for melting the solid DRI and/or scrap included in the charge on economic grounds.

It means that the hot metal refining will be carried out in the EAF, treating it as a BOF with top oxygen blowing (more like LD or any of its modifications in the form of a hybrid process). From the thermal-balance point of view, the vessel size has to be a certain minimum so as to make it autogenous. Along with the size of the EAF, the hot metal proportion in the charge must also be large enough to make it autogenous, to more than compensate the heat losses, during refining, by the exothermic heat of refining. This is the reason why this process, in the absence of adequate hot metal supply, is reduced to a mere traditional EAF operation. This is the reason why the BF(Corax)/DRI/EAF combination route has to employ large-capacity EAFs.

The EAF sizes used in these cases are generally in the range of at least 80–100 t or so. A minimum of two such furnaces are used together, and in combination, perhaps coupled with ladle furnaces, to produce steel. These are invariably coupled with continuous casters to achieve optimum yield.

21.5.2. Induction Furnace Process

In view of the above problems in melting DRI in an EAF, attempts to melt DRI in an induction furnace proved highly successful. An induction furnace is an almost ideal furnace to melt DRI since it does not have its own oxidizing or reducing atmosphere. It is a melting process wherein what goes in comes out with respect to the chemistry, as not much refining is possible. It has a perfectly neutral atmosphere except that the melt is in contact with open atmosphere and to that extent some oxidation of iron may take place. But this is much less than the oxygen level already present in DRI in the form of leftover oxide of iron.

An induction furnace has the following advantages:

1. High temperature can be generated quickly to the extent the crucible lining can permit.
2. Loss of heat and electricity is very low.
3. The temperature and melting rate can be adjusted at will.
4. Continuous stirring through the *fountain* effect homogenizes composition and temperature and helps non-metallics to float to produce relatively better clean steel.
5. Gas pick up, such as that of hydrogen and nitrogen, is low.
6. There is minimum melting loss of alloying elements in the charge during melting.
7. There is better recovery of alloying and deoxidizer additions.
8. There are low noise and emission levels as compared to EAF.
9. It can be readily coupled with a suitable secondary refining unit to produce quality steels as well.
10. Being relatively smaller in sizes as compared to the EAF, these can be located to make general purpose steel at the market site itself, thereby avoiding unnecessary transport of finished steel to market sites.

11. Small-scale production of liquid steel is best suited to foundries.
12. These can be used in the form of cheaper acid-lined furnaces (the basic lining is relatively very costly).
13. The inability to melt slag (insulating) is advantageous since it remains viscous and is easily removable cleanly from the steel melt.

Melting of DRI can be safely carried out in acid-lined induction furnaces since the gangue in DRI is totally acidic in nature (i.e., silica and alumina mainly), and generally it is 2:1 in proportion under Indian conditions. The slag thus produced is nearly saturated with respect to silica. It may contain SiO_2, Al_2O_3, and some FeO as main constituents. The silica and alumina together make only very dry and refractory slag but the presence of iron oxide may make it somewhat thin but sticky.

The lining is made by placing a both-ends-open cylindrical welded steel shell inside the furnace and then by ramming silica sand in the annular space left between the shell and the primary induction coil. The steel shell is not removed. It melts with the first charge and bonds the silica grains in the outer layer, thereby giving it sufficient strength to withstand the rigors of charging and melting. This is the cheapest form of lining and much cheaper than the basic lining, which must be used for EAF using scrap-HM-DRI as the charge.

This process is simple in operation as compared to the EAF. The charge generally consists of steel scrap approximately 20%, cast iron about 15%–20%, with the balance being DRI. After the previous heat is tapped, some heavy steel scrap is charged and melted to prepare a small pool of steel. Then cast iron is charged to make a metal pool containing some carbon and silicon. This is necessary to take care of the oxygen present in the DRI; otherwise, it is very prone to attack the lining. The iron oxide combines with silica to form $2FeO\text{-}SiO_2$, or fayalite, which is a low-melting liquid. It therefore results in eroding the lining. The carbon and silicon in the initial melt, before addition of DRI, thus help to reduce this iron oxide and thereby reduce its attack on silica lining. Once these two are melted, the DRI charging can begin. The rate of charging has to be adjusted with respect to the power available for heating and melting. The DRI charging can be in the range of 27–30 kg/min/MVA power input. It is better to feed the material at an angle of approximately 30–40 degrees to the horizontal. Balance steel scrap can be charged in between judiciously. The charging and melting thus goes on simultaneously to melt the entire charge. Whatever refining is possible takes place during the melting, and no specific refining is generally carried out with the addition of any oxidizing agent.

Since no refining of sulfur or phosphorus is possible, the charge must have their total contents below the product-specification level. Petroleum coke is used to reduce the iron oxide present in the DRI as much as possible. Use of ordinary coal is generally prohibited because of its high sulfur content. If it is used, it is bound to increase the sulfur content of the final molten steel. Slag is removed manually during the melting period and finally by tilting the furnace as well at the end of the heat.

After the slag is nearly completely removed, the heat is deoxidized by Fe-Si and Fe-Mn or Si-Mn and aluminum to make up the final chemistry. Bags full of these materials are thrown in the furnace just before the tapping to make up the additions. It is advantageous to deoxidize the bath in the furnace itself because the inductive stirring action of the furnace helps to clean up the melt.

This steel making route is adopted for small-scale production of the order of up to 3000 t/day or a production rate of 0.1 Mt/annum. This is the cheapest mode of making the ordinary steel required by the construction industry in the form of rounds and twisted bars, small angles, channels, strips, etc. They can thus meet the local needs very efficiently and very economically. A million-ton or above-level plants cannot compete with these process economies readily in spite of their high productivity.

21.6. Energy Considerations

In spite of adoption of the electric arc furnace on an ever-increasing scale, the debate did not stop as to which one of the two energy types, electrical or chemical energy, is economical for steel making in a given situation. This debate still continues, and it swings every now and then on the relative economics in a given situation. The energy efficiency for carrying out the various functions in steel making has therefore been analyzed theoretically and is shown in Figure 21.4.

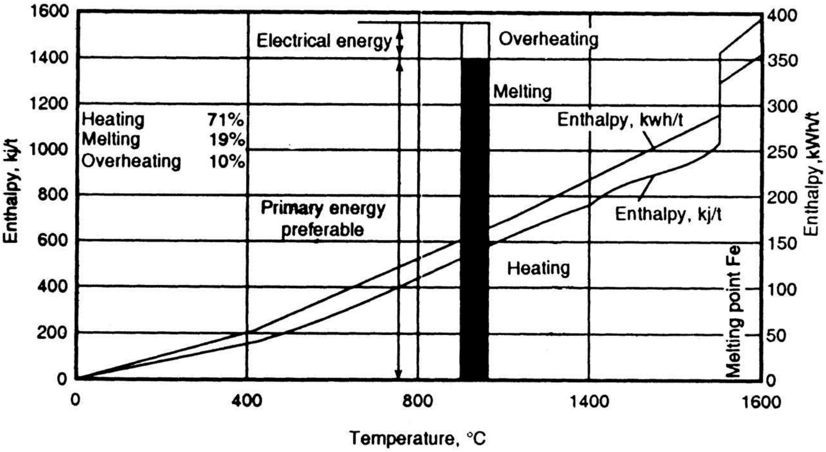


FIGURE 21.4. Energy requirements in electric arc furnace process [38].

From the point of view of energy consumption using solid charge in an electric arc furnace, the energy is required for three major functions as [38]:

1. Heating the solid scrap to its melting point
2. Melting the solid scrap
3. Superheating the bath

From Figure 21.4 it is clear that nearly 71% of the total energy input is required for just heating the scrap (first step), that is, when the charge is still in solid state. Of the total energy input, 19% is consumed in melting of the solid charge and only 10% is required for superheating the liquid steel to the refining temperature. Since a large surface area in the form of solid charge is available, the heating efficiency of chemical fuels (fossil fuels) in heating the solid scrap is much higher than by the electric arc. On the contrary, heating of molten bath by arc is far more efficient than by chemical fuels because of lack of adequate surface for efficient chemical heat transfer. In the traditional open hearth the heating of the molten bath was continued by the chemical fuels even after melting the solid charge, and it made it far more inefficient in terms of chemical fuel efficiency. This fact contributed immensely in making it obsolete. The specific surface area of the charge therefore decides which mode of supply of energy would be more efficient. The present electric arc furnaces therefore make use of fossil fuel energy to the maximum for melting the solid charge and, once the charge is molten, they switch over to electric arc heating since it is far more efficient at that stage to finish the process. The development of jet-box or the more efficient co-jet to introduce fossil fuel energy for melting solid charge in arc furnaces is the net result of this understanding alone.

The exothermic heat of refining in BOF processes is used to melt a certain amount of solid charge along with the refining to make them energy efficient. This fact is fully utilized in modern electric arc furnaces while refining the hot metal charge part before charging and melting, particularly the DRI.

The modern emphasis is in reducing the overall greenhouse gases in the environment in any processing. Much of the electrical energy is generated initially by burning coal in thermal power stations. Hence the direct use of the fossil fuel in an electric arc furnace needs to be weighed vis-a-vis that used indirectly by producing electric power first. Generally, the thermal power stations run on 35%–42% efficiency. The efficiency further gets reduced because of an additional 10%–12% of transmission losses in transporting electric power to the end user. This power is reconverted to thermal energy for steel making through arc, and it has its own efficiency of conversion.

Therefore, it is advantageous to use directly the fossil fuel for steel making to the extent that it is more efficient. It means the heating of the scrap and its melting, as far as possible, should be carried out using fossil fuels, and once the charge is molten, subsequent heating of the bath be carried out by electric arc alone. This is the trend now in designing and operating the modern electric arc furnace and its practices for steel making. A large number of alternative designs of electric furnaces are being developed the world over in this direction. Similarly, the thermal energy of refining available in the form of carbon monoxide, that is generated during refining, is utilized to its fullest extent in preheating the solid charge in steel making or as a fuel in thermal power plants operated as adjuncts to the steel plant.

21.6.1 Charge Preheating/Melting

The energy requirements for electric steel making must be decreased to improve its economy. The efforts in this respect have been directed mainly in using the enthalpy contained in the exhaust gases evolved, during refining for preheating the solid charge, such as DRI or scrap. Alternatively, the enthalpy of exhaust gases is being used for generation of thermal power in the plant as a by-product. Similarly, fossil fuel energy is being used, to the extent possible, for heating the solid charge to its melting point. Some alternative processes have been developed in which the charge is melted in a separate furnace, using fossil fuel and/or exhaust gases, and the liquid steel flows into the EAF for further refining. Such technologies are now commercially available. They are normally known by their designers.

A number of technologies are available wherein the solid charge is preheated nearly to 5°C–600°C before being put into the EAF. It has not been found economical to preheat the charge beyond this temperature. Exhaust gases from the EAF can be used for this purpose.

21.7. Oxygen Lancing and Process Dynamics

The use of oxygen for refining in open hearth and electric arc furnaces is not new. It was, however, very limited in the beginning. The rate of blowing was limited due to the danger of splashing of slag on the roof and the side walls. This limited the refining rates that could be achieved even after resorting to oxygen refining.. Once the EAF shell was constructed out of water-cooled panels and its height was increased to some extent, it paved the way for even top lancing of oxygen as is done in the LD process. Yet the rate of top blowing of oxygen has its own limitations. The top blowing necessarily results in slag-metal foam or emulsion formation during refining. The EAF is limited in providing the free-board required to accommodate this foam as is available in the LD vessel. In EAF the door is at a very low level, and that limits the freeboard availability. This results in profuse slag-flush during refining. It may be advantageous if high phosphorus iron is to be refined. But when phosphorus is to be removed only marginally, such a flush is not required at all. In fact, the slag is very rich in iron oxide content when in foam form, and if such slag flushes out, it results in great loss of iron and hence the yield is adversely affected.

The obsession of increasing the production rate, by increasing the refining rates, to be achieved by increasing the blowing rate and also by lowering the lance-height, ultimately results in enormous foaming and consequently rapid slag flush and attendant entrainment of liquid metal, causing greater loss of metallic yield. Judicious use of top oxygen blowing and side oxygen blowing is a must to achieve higher yields. The fact remains that flat EAF hearths cannot be used for refining with the same vigor by top lancing as in the standard LD practice.

Instead of getting the best of LD and EAF practices in this process, as a combination, it results in getting the worst of both. The typical example of this is the Con-Arc process, which has been adopted in Ispat Industries Ltd at their Dovi plant. There are only two such plants in the world, the other being at Saldana in South Africa. The slag that flows out from EAF, in this process, is very rich, sometimes reaching as high as 40%–45% Fe, as against the normally accepted value of 15%–20% for any basic steel making operation. This can be somewhat reduced, but by maintaining a very high level of basicity such as 3.5 as against the usual level of 2.0–2.5. Normally 12%–15% of the charge weight is generated as slag volume during a basic steel making operation. This excess iron oxide content of the slag amounts to a loss of more than 3%–5% of the yield and which simply eats away a great deal of profit of the operation, no matter how efficiently it is run.

As discussed earlier, either jet-box or now more popularly co-jets are installed on the EAF to enhance the productivity by increasing the thermal energy supply to the charge. These jets make the entire top blowing period more dynamic. During the DRI melting, under arcing conditions, the co-jets are operated at their maximum ability. It contributes more dynamism to the already boiling, such as steel-melt, due to the bottom purging and carbon reaction. The practice of bottom purging, which is otherwise a must for slag and metal mixing, contributes its own dynamism to the refining operation. All together these factors make the whole process very dynamic. The use of foamy slag practice, often aided by addition of coal/coke breeze, increases the dynamism such that the slag flows out of the furnace door at a very fast rate. In a 200-t furnace this flow may be as high as 1.5 to 2.0 t of slag per minute. Consequently, this high flushing rate entrains a great deal of metal along with it. In fact, the liquid metal flows out of the furnace as a pipe-like stream along with the slag flush. All this is due to the very dynamic nature of the process developed because of all the above-mentioned reasons. It reduces the metal yield correspondingly by another 2%–6%. These are the typical problems that plague the Con-Arc practice.

This is a typical illustration to emphasize the necessity to understand, not only the thermodynamics and kinetics of metallurgical operations, but to appreciate the dynamic nature of fluid flow in steel-making techniques and their limitations before copying them from one process to a new area of application.

21.8. DRI Melting

The DRI melting should strictly be made in reducing conditions to minimize the iron loss on oxidation. If this is not possible then it should at least be melted in neutral conditions. Melting of DRI while arcing is going on and when even low-pressure oxygen spray is supplied for usual refining through co-jets makes the whole atmosphere quite oxidizing inside the furnace. The DRI is a victim of all this. This is the reason why the slag from such a furnace is very rich in iron oxide, which is all lost along with slag. It increases the slag volume and thereby contributes to higher metal losses even by mechanical entrapment. Ideally DRI should melt with no oxidation of iron and in fact the oxygen contained in DRI should react with carbon to give more units of iron as a whole. It is therefore recommended to melt DRI after preparing a pool of liquid iron so that it can be charged into the melt to minimize furnace oxidation. In this respect, HBI is better because its being denser makes it sink in the melt, whereas the DRI floats and facilitates iron oxidation unnecessarily.

The DRI variables such as % metallization, % carbon, and % gangue all have an effect on energy consumption during steel making. If there is sufficient carbon to balance the FeO in the DRI, the total iron is recoverable. For this, nearly 1% C content is required to balance 6% of FeO. This can improve iron yield. Generally, 100–200 kWh/t extra power is consumed in melting DRI as compared to melting of scrap. Because silica is invariably the gangue associated with DRI, a basicity of 2.5–3.0 is recommended for slag to melt DRI efficiently. This will lead to relatively lower iron oxide loss in slag.

21.8.1. Mechanism of DRI Melting

DRI is generally added in the furnace when there is a pool of molten metal already present. This is prepared either by melting steel scrap first or by addition of hot metal as the charge. The melting process on a micro level can be described as follows:

1. DRI particle gets heated during its free fall from hopper till it reaches the bath.
2. It may be heated during this fall and also oxidized partially.
3. The DRI particle has a chilling effect on the bath at the contact.

4. It slowly gets heated with time.
5. In effect mass transfer and heating is going on at the Fe(DRI)/Fe(melt), FeO(DRI)/Fe(melt), Gangue(DRI)/Fe(melt), Fe(DRI)/slag, and Gangue(DRI)/slag possible interfaces on micro level.
6. The rate of each one of these interactions decides the rate of dissolution of DRI in the melt.
7. On macro level, DRI as a whole takes heat from the melt and eventually itself dissolves in the melt.
8. All the above micro interactions are time-dependent.
9. In an actual operation some other processes also occur along with those outlined above.
10. It is a mass transport controlled process and it requires heat for its completion, and therefore the rate of feed has to match with rate of energy input.
11. If the rate is faster than this match, DRI particles may float in slag as like *icebergs* and if the rate is short then energy input efficiency is bound to be low, adversely affecting economics.

The process becomes more complicated if slag is allowed to flush out simultaneously. This mechanism is somewhat similar to that of cored wire dissolution in liquid steel.

Therefore, the feed rate must follow:

Power input rate = Thermal energy requirements for heating and melting DRI up to bath temperature

Hence, correct feed rate has to be established in tune with the total electrical and chemical energy input rate in any furnace. Production pressure often hastens feed rate, disregarding this basic fundamental fact and end up in loss of yield. It is equally necessary that most of the oxygen contained in the DRI must also get reduced by carbon or silicon at the rate at which melting is carried out. Alternatively, iron loss in flushing slag is bound to be high. The DRI particle size also can have a decelerating effect on its dissolution rate. An optimum size of DRI is thus recommended in a given practice to obtain optimum efficiency of melting.

CHAPTER 22

SECONDARY STEEL MAKING PROCESSES

In This Chapter

22.1. Introduction

For most applications, the surface quality, internal quality, micro-cleanliness quality, and mechanical properties of steel produced by conventional means, such as open hearth, LD, BOF, OBM, and the like, bulk steel making processes were once quite satisfactory. Hence these processes met most of the requirements of steel making. However, with time the quality requirements continuously increased and became more and more stringent in terms of cleanliness, tighter grain size, narrow hardenability range, etc.

Initially, the steel makers attempted to produce steel of such requirements by modifying or sometimes by prolonging the furnace refining practice. Many times, such methods failed to achieve consistently the desired results or at times these methods of refining were not practical or even if they proved feasible, it was at the cost of loss in production and under-utilizing the costly equipment that was provided.

These requirements and compulsions led to the development of a series of new and different supplementary steel processing techniques as per the requirements. The Perrin process is the earliest of such techniques to appear on the scene as early as in 1933, specifically for improving cleanliness of ball bearing steel, after the heat was made in the usual electric arc furnace. Similarly, the low-hydrogen specifications, particularly in some high-alloy steel of special applications led to the development of vacuum degassing of liquid steel, after it was made in a usual bulk steel making process. It was introduced during World War II. Several such processes have since been developed and put into commercial practice all over the world.

As these developments were taking place, the whole process of steel making was subject to continuous review from the fundamental standpoint of economy of production without sacrificing required quality of the product. This approach finally led to the development of the concept of *secondary steel making* for quality production. As a result, the bulk steel making processes were restricted to produce, by themselves, only the ordinary general purpose variety of constructional steel. All the rest had to be processed through one or more of the secondary steel making route as well, to achieve the desired end quality.

Although the Perrin process, introduced way back in 1933, contains the principal components of secondary steel making, the terms *secondary steel making* or *secondary steel refining* are of recent origin. On this basis now, the major bulk steel making processes such as open hearth, LD, OBM, LDAC, EAF, etc., are all classified as *primary steel making* processes wherein the major portion of the total refining, including melting, if any, is carried out. The final refining and finishing are carried out in any one or more of the secondary steel making processes.

In principle, this modification is nothing but a duplex practice of steel making, as is understood, in the conventional sense of steel making. In that the major part of lengthy refining, alloying, deoxidation, that is the lengthy and the slower part of refining, including the deep desulfurization, are all shifted to the secondary zone, leaving the faster part of refining, including melting, if any, to the primary unit of steel making. The secondary processes are not by themselves capable of carrying out melting and/or major refining using the usual charge mix. The secondary steel making vessels along with their accessories are relatively cheaper, and by installing one or two of these as an adjunct to the bulk process unit, the primary unit can be more fully utilized for melting and primary refining. This combination aims at carrying out the bulk refining in the primary unit, while the slow tail-end of refining and finishing is carried out leisurely in one or more secondary units.

The fundamental understanding of the secondary refining techniques has had its influence in modifying the primary steel making process as well. The stirring techniques, once adopted as primarily a secondary treatment, and because of its low cost, has now become an inalienable part of primary steel making furnace to achieve better mixing of slag and steel bath. The LD and the EAF are both now fitted with bottom purging to improve upon the mixing of slag and steel melt in the primary unit. It helps to achieve better equilibrium conditions all through the bath.

The history of secondary steel making went through *three distinct stages* of development. In the first phase, simple ladle metallurgy was developed. This was an extension of the progenitor Perrin Process. The improvements aimed at were:

1. Improve deoxidation control
2. Remove inclusions by gentle bath stirring
3. Desulfurize steel by synthetic slag and injection metallurgy
4. Modify inclusions primarily by calcium

In the next development stage, ladle furnaces were developed to improve productivity and the quality of steel. Ladle furnace was used to achieve:

1. Reheating of steel and control of teeming temperature
2. Buffer heats for sequential casting
3. Large alloying to produce high-alloy steels
4. Homogenization of heat with respect to chemistry and temperature produce ultra-clean steels with an extended and gentle gas stirring
5. Desulfurize and in some cases even dephosphorize the steel with synthetic slag, employing even, one after the other, two different slags

In the third stage of development, mere vacuum degassing was developed into full-scale vacuum treatment of steel, which has resulted in its very wide-ranging applications. As a result, an increasing proportion of steel is being vacuum treated. The net result of the development of these secondary treatments is that, now a steel plant, worthy of the name, is designed to include, along with the primary steel making process, as in the past, necessarily a secondary processing line such as an LF furnace and a vacuum treatment plant of some sort. This design is shown in the form of a layout in Figure 22.1.

Today, secondary steel making is basically resorted to achieve one or more of the following requirements:

1. Improvement in physical quality of the product in terms of surface quality and internal homogeneity
2. Closer and more homogeneous chemistry
3. Lower levels of impurities/tramp elements
4. Improvement in overall production rate
5. Decrease in energy consumption, if any
6. Use of relatively cheaper grade or alternative raw materials
7. Use of alternative source of energy
8. Higher recovery of alloying elements

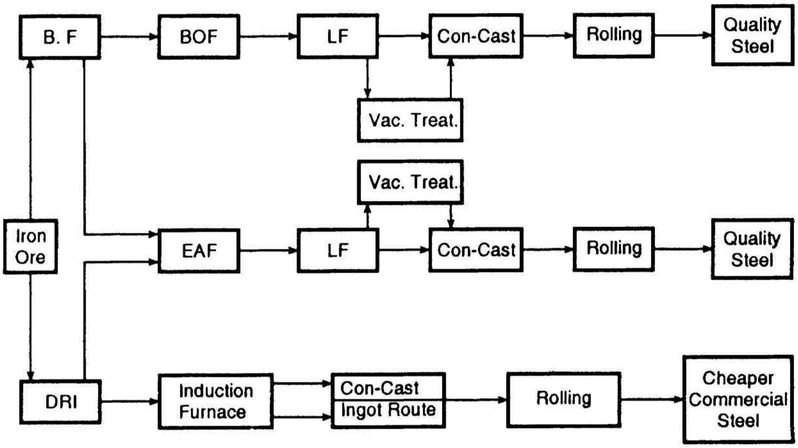


FIGURE 22.1. A modern integrated steel plant, describing the essential units provided therein.

9. Effective temperature control

10. Deeper decarburization

11. Desulfurization for extra low-end sulfur

12. Dephosphorization for extra-low P.

13. Degassing

14. Deoxidation

15. Modification of morphology/chemistry of inclusions

16. Improvement of cleanliness

17. Control of solidification structure

18. Micro-alloying

The concept of secondary steel making has resulted in a very fundamental change in technology of steel making. It has also become universal in its applications. They tailor to each specific requirement of product-range. Lately the secondary steel making itself is increasingly being subdivided into two broad divisions. The first is *gross secondary refining* and the second is *fine secondary refining* to get more exactly the required product. The following table indicates the progress in secondary steel making over the last several decades in terms of impurity level.

22.2. Process Varieties

The varieties of secondary steel making processes that have proved to be of commercial value can broadly be categorized as under:

1. Stirring treatments

2. Synthetic slag refining with stirring

3. Vacuum treatments

4. Decarburization techniques

5. Injection metallurgy

6. Plunging techniques

7. Post-solidification treatments

8. Tundish metallurgy

Table 22.1.

Indicates the decreasing level of impurities in steel specifications[39]

Contents	*Year*			
	1960	*1980*	*2000*	*Future**
C	250	150	20	10
P	300	150	50100	30
S	300	30	10	10
N	150	70	30	20
O (total)	30	30	10	10
H	6	6	1	1
Total	1036	436	121171	81

*Future indicates the achievable levels.

Each one of these processes may be capable of achieving more than one goal of refining and finishing and therefore, several designs are available in the market for the buyer to choose from, for one's own precise requirements.

Many of these processes can be carried out in the steel transfer ladle itself or in a simple ladle-like furnace known as *ladle furnace* (LF for short). Many, therefore, refer to secondary steel making as *ladle-metallurgy*, which is also a popular term in the literature.

The secondary steel making vessels along with their accessories are relatively cheap, and by installing one or more of these, the primary unit can be more fully utilized for melting and primary bulk refining, thereby raising considerably the overall production rate, since now the slow tail-end refining and finishing is carried out leisurely in a secondary unit. In other words, part of the steel making activities go parallel and hence on the whole the total time required for refining is reduced to some extent.

22.3. Stirring Techniques

Stirring a steel heat in the transfer ladle itself is the most simple kind of secondary treatment of steel. It was essentially introduced to float deoxidation products, in the form of non-metallic inclusions, and to obtain homogeneous chemistry of the bath. It has now been employed almost universally, to help mix slag and metal, in the primary steel making furnace itself, to achieve better equilibrium conditions of refining. It now aims at:

1. Homogenization of temperature and chemistry all through the steel melt
2. Minor alloying for close control of chemistry of the steel bath
3. Cleanliness improvement on the whole

Stirring of the steel bath is achieved by bubbling inert gas such as nitrogen or argon in the steel bath held in the transfer ladle as shown in Figure 22.2 There are five different processes for achieving this objective, viz., from the top through immersed pipe, from the side tuyere, from the bottom through a single centrally located porous plug, from the bottom using a single eccentric plug, and from the bottom through multiple plugs as shown in Figure 22.2 The efficiency of stirring is measured in terms

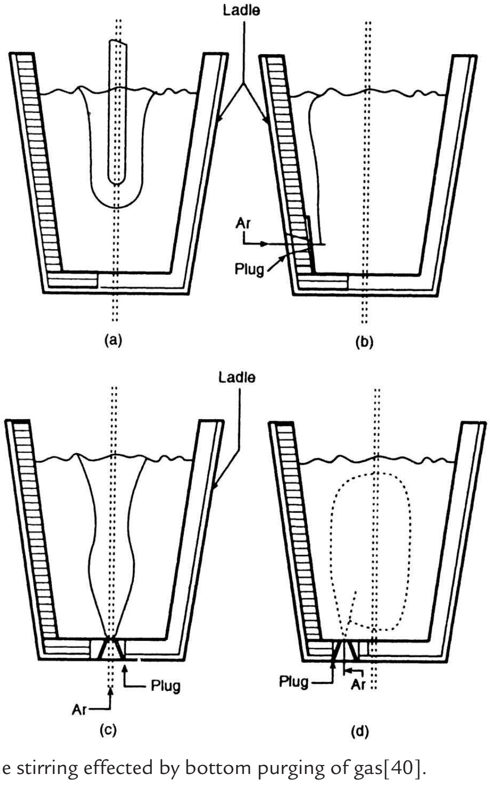


FIGURE 22.2. Shows the stirring effected by bottom purging of gas[40].

of energy associated with bubble formation and its size. Homogenization of bath temperature and composition is achieved because of the buoyant energy of injected gas, which is calculated as:

$$E = 14.23\,[VT/M]^{*} \log\,[(1 + H)/1.48\,P_0]$$

Where E = stirring power

V = gas flow rate, N m^3 /min

T = bath temperature in K

M = bath weight, ton

H = depth of gas injection, m

P = gas pressure at the bath surface, atm

The mixing time as defined by 95% homogenization is given by:

In sec = $116^{*}\ E^{1/3}\ {}^{*}\ D^{5/3}\ {}^{*}\ H^{1}$ (where D is ladle diameter, m)

The rising bubbles tend to lift up the non-metallic inclusions, due to its surface tension effect. These lifted-up particles are supposed to be absorbed in the slag that is invariably present at the top of the bath. This is the main purpose for which it was invented. As the bubbles rise in the bath, their sizes increase due to the reduction of ferro-static head, and this helps to lift up the inclusions along with the bubbles. There is a limiting flow rate at which this will happen at the optimum. Any excessive blowing rate can lead to mixing of the top slag with metal such that the inclusions may in fact get increased. The rising gas, on coming in contact with the top slag layer, opens up the slag layer, frees the metal layer of slag, depending upon the gas flow rate and volume. It gives rise to the formation of what is often called the *eye* in the slag, as shown in Figure 22.3. Formation of a small eye such as this at the slag layer is typically the right condition of blowing for a given capacity of the bath and its design. Gas is blown at 3.5–4.5 bar pressure and at the rate of 100–500 Ndm^3/min.

Since gas is bubbled in a cold condition, it has a cooling effect on a steel bath temperature and hence it must be used sparingly to achieve the most beneficial effect. The temperature drop of the steel bath can be roughly calculated from the equation below:

$$\begin{aligned}&(\text{Mass of bath}) \times (\text{Sp. Heat}) = (\text{Mass of Gas}) \times (\text{Sp. Heat})\\ &\qquad \times (\text{Temp Drop}) \qquad\qquad\qquad \times (\text{Bath temp})\end{aligned}$$

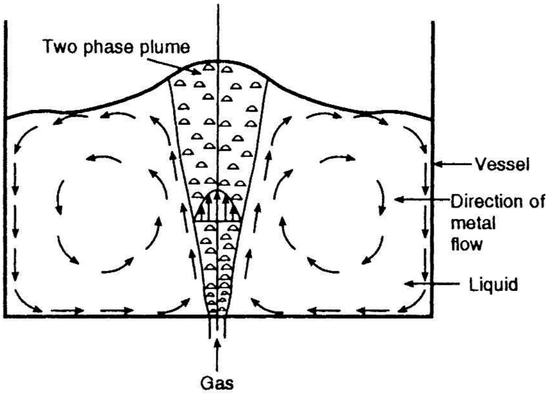


FIGURE 22.3. Schematic representation of the situation in a gas-stirred tadle.

This is assuming that the initial gas temperature is practically zero and that the gas attains bath temperature at the exit. As a rule of thumb it may be assumed that the argon purging may result in a drop of nearly 1.5°C/min in the bath temperature.

Attempts have been made to correlate the amount of gas bubbled, in terms of its energy of bubbling, with the extent of cleanliness achieved in a given practice. This mathematical quantification is available in the literature. The energy associated with bubbling is chiefly responsible for the lifting of non-metallic inclusions to bring them into contact with the slag layer at the top.

Gas is often bubbled through porous refractory plugs made of magnesia or alumina and fitted at the bottom of the ladle. Several designs are available, and some of them have already been shown in Figure 20.4. The porosity of the plug must be maintained at around 30%–35 %. The pores need to be fine enough so that metal does not enter them because of the surface tension effect. Similarly, they tend to get clogged by the slag towards the end of teeming. A good plug would readily give a life of several hundred heats. The amount of gas so purged is quite small. It is usually a few m^3 per hour. This design is useful even for fitting on to the primary steel making furnace for achieving better slag-metal mixing and homogenization of temperature and chemistry of the bath. The LD process has thus been modified into bath-agitated processes.

The gas can be introduced alternatively through a consumable steel pipe inserted from the top. Similarly, it can also be introduced by the side as well, as shown in the figure. Its capacity for homogenization is much limited and depends upon the extent to which the pipe can be immersed below the surface of the steel bath and the amount of gas blown.

The capacity of the stirring arrangement to improve cleanliness of steel primarily depends upon the ability of this arrangement to assimilate the inclusions in the top slag layer.

22.3.1. Cleanliness Improvement

The inclusions floating in the bath will have a chance of being eliminated only if they are lifted by the rising gas-metal plume and if they meet the slag at the top surface. Another necessary condition for their removal is the energy criterion. The energy of the inclusion-slag couple must be less than that of the inclusion-metal system for its effective removal. The interfacial tensions at the gas-metal, slag-metal, and gas-slag are relevant in this regard. The mechanism of inclusion removal is as follows:

(*a*) An open eye is formed when a critical gas flow rate is achieved.

(*b*) A layer of spheres of slag droplets covered with thin metal film is formed around the open eye (called *sphere-bed*) [41].

(*c*) Small inclusions are brought up to this eye by the bubbles.

(*d*) The sphere-bed acts as a filter, which captures some of the inclusions, which is possible only if energy considerations are favorable.

(*e*) Some inclusions may take more than one lift before they are caught by the slag.

(*f*) The system is far from still bath conditions when Stoke's law holds.

(*g*) Inclusions do collide, coagulate, and coalesce and thereby increase in size, which helps them float more readily.

22.4. Synthetic Slag Refining with Purging

It involves the use of specially prepared pre-melted slags to act as either a *sink* for non-metallic inclusions or to help refining of some specific impurities such as *S*, *P*, etc. An unstirred slag-metal system is most ineffective in carrying out any interaction since the apparent area of contact is too small. The effectiveness is improved by stirring the two. Originally it was used for effective desulfurization. However, now synthetic slags are used, even for producing extra-low-phos steel using the same technique in principle. The synthetic slags for desulfurization are almost free of FeO and MnO, whereas those for dephosphorization are highly oxidizing and highly basic in nature. The quality of synthetic slag has been explained earlier in Section 3.2.1.

22.4.1. Perrin Process

This was the process that was conceived to effect better mixing of slag and metal, when artificial stirring by inert gas purging as is done today was beyond imagination. A novel method was conceived to achieve the vigorous mixing of slag and metal to effect removal of non-metallic inclusions from refined steel before casting. This was the main requirement of ball bearing steel to better achieve the required properties. This came to be known as the Perrin process. It is the first process wherein the refined and finished steel from EAF was further subjected to refinement by enforcing the liquid steel to interact violently, with a synthetic, non-oxidizing molten slag, to effect further improvement in the cleanliness of steel. The required stirring of the slag-metal system, in the days when inert gas bubbling was unknown, was achieved by pouring liquid refined and otherwise finished steel in a ladle containing molten synthetic slag. The slag-metal mixing created by the turbulence of the falling steel stream in a slag pool, even in a short time, was so great that efficient desulfurization, deoxidation, and scavenging of non-metallics occurred very efficiently to produce low sulfur and better clean steels. In a later modification both the synthetic slag and the liquid refined steel were poured simultaneously in a transfer ladle at predetermined rates to achieve still better results.

The process was patented in France in 1933 and was adopted at the Mahindra Ugine Steel Co. at Khopoli in Mharashtra. In this process, steel is melted, refined, and blocked as usual in an arc furnace. The oxidizing slag is then removed. Ferro-alloy additions are made to adjust the chemistry, and the heat is tapped. The teeming ladle in this case contains a pre-melted slag of nominal composition 60% $Ca0$ and 40% $A1_2O_3$. The weight of the slag is nearly 1.5%–3.0% of the weight of the liquid steel.

The process is adopted at this plant to produce low-alloy constructional steels, engineering and special steels, ball bearing steels and the like, with much better cleanliness and lower sulfur levels. It eliminates the need of reducing the period in an arc furnace and is especially suited for raw materials containing high sulfur percentages.

In the Russian Perrin process liquid steel from a basic open hearth was poured in a ladle containing synthetic slag. A large slag bulk with higher uniformity of product quality than that obtainable in an arc furnace, was claimed to be obtainable by this process.

This process has now been replaced by the bottom inert gas purging technique. Synthetic slag of the same type is used along with it in order to achieve the desired desulfurization. Injection metallurgy techniques such as calcium wire feeding are also used, in addition, to improve cleanliness.

In a *sealed argon bubbling*, known as the SAB process, a synthetic slag is put on the surface of the bath in the transfer ladle. An immersion box is used to dip just below the slag layer. It is located vertically above where the porous plug is fitted at the bottom. This prevents the steel from getting exposed to the atmosphere due to the rising bubbles at the eye.

A similar process using a capped ladle is known as *capped argon bubbling* or CAB, process. Both of these processes were developed by the NSC, Japan. They claim that these are the simplest and very cheap secondary refining processes.

Most of these early developments have now been superseded by a standard *Ladle Furnace Technology* (LF for short), wherein a bottom purging facility is provided and arc heating is possible to make up the temperature loss. This is operated by charging synthetic slag, melting it by arc, and simultaneously passing argon from the bottom plug. One or more plugs is provided as per requirements. This has proved to be very efficient in achieving the required desulfurization and better cleanliness and teeming at correct temperature to produce sound, continuously cast product.

22.5. Vacuum Treatments

This will be described in a separate chapter (Chapter 28) in the form of vacuum treatment of liquid steel after considering the problem of gases dissolved in a refined steel bath.

22.6. Decarburization Techniques

Extra low-carbon soft steel can be produced by effective decarburization in a primary steel making furnace itself. Low-alloy low-carbon steel can be produced by ladle additions of low-carbon ferro-alloys. The problem, however, is serious if low-carbon, high-alloy steel is to be produced. The manufacture of stainless steel is a typical example in point wherein decarburization of the bath is to be carried out in the presence of chromium without much of chromium oxidation. The problem of making stainless steel using stainless scrap and high-carbon ferro-chrome has been described in Chapter 14 as the Rustless process and in Chapter 15 using oxygen lancing in an arc furnace. The latter has the following disadvantages:

(*i*) Utilization factor of power is low.

(*ii*) Although cheap high carbon ferro-alloys are used to a large extent, costly low carbon ferro-alloys are still needed for final adjustments, and in terms of cost it is by no means negligible.

(*iii*) Oxygen lancing develops a high bath temperature (1800°C), which affects the lining life adversely, and at times it causes discontinuity in production.

(*iv*) Heterogeneity in composition of the bath.

(*v*) Recovery of chromium is low.

The fundamental understanding of the thermodynamic aspects of the reactions involved in stainless steel making finally led to evolve more efficient and economical techniques of decarburization of steel bath in the presence of chromium. The underlying principles can be understood from the following: The basic equilibria governing stainless steel making, although described earlier in Chapters 14 and 15, may be recapitulated as:

$$3\{CO\} + 2[Cr] = <Cr_2 O_3> + 3[C] \qquad (22.1)$$

When a stainless steel bath is decarburized using pure oxygen lancing, the atmosphere in equilibrium with the bath will be essentially pure CO gas at one atmosphere. This automatically decides the maximum level of chromium that can exist in the bath in equilibrium at a given level of carbon and temperature of the bath (refer to Chapters 14 and 15). Any excess chromium would be oxidized and which will either be lost with the slag or will have to be recovered from the slag. As the carbon content of the melt decreases, the equilibrium chromium level also decreases. These relations determine the extent to which the melt can be decarburized without significant oxidation of chromium. The improvement that is possible by raising the temperature of the bath is limited by the economics of the lining life of the furnace. The thermodynamic alternative is, therefore, to reduce the equilibrium partial pressure of CO gas in the ambient atmosphere of the melt so that further decarburization shall take place in preference to chromium oxidation, as shown in Figure 22.4.

This has been achieved on a commercial level in rather ingenious ways in the form of development of special decarburization techniques. In the argon-oxygen-decarburization technique, AOD for short, the partial pressure of CO in the ambient atmosphere is decreased by blowing an argon-oxygen mixture, in place of pure oxygen for refining. In the vacuum-oxygen-decarburization, VOD for short, oxygen is blown for carbon oxidation under vacuum. In the Cresesot-Loire-Uddeholm, CLU for short, process an oxygen, steam, and nitrogen mixture is blown for refining.

In the recently developed metal-refining process, MRP for short, the AOD design is changed from side-blown arrangement to bottom-blown design for its superiority. Blowing is carried out using oxygen and argon, both.

22.6.1. Stainless Steel Making Technology

Prior to the introduction of the Rustless process of steel making, stainless steel scrap could not be reused again for the recovery of chromium values. The Rustless process founded the basis for reuse of stainless steel scrap without many difficulties. The introduction of oxygen lancing for

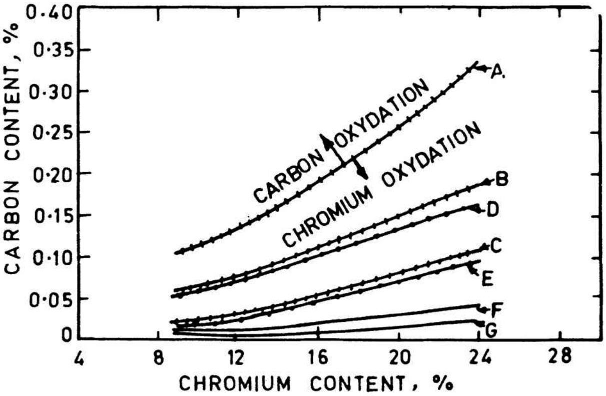


FIGURE 22.4. The inter-relationship of carbon and chromium dissolved in iron as a function of temperature and pressure.

A 1700°C/1 atmos.
B 1800°C/1 atmos.
C 1900°C/1 atmos.
D 1700°C/0.5 atmos.
E 1700°C/0.5 atmos.
F 1700°C/0.1 atmos.
G 1800°C/0.6 atmos.

preferential decarburization of a bath containing chromium paved the way for incorporation of high-carbon Fe-Cr in the charge mix for obtaining chromium values to some extent. The proportion of HC Fe-Cr in the charge could be to the extent of getting nearly 0.4% C at the melt-out stage for only this much could be decarburized by pure oxygen lancing without appreciable loss of chromium.

The modern decarburization techniques permit the use of much higher proportion of HC Fe-Cr (standard grade or even-charge chrome (low chromium Fe-Cr with high carbon) in the charges since they are capable of decarburizing the bath at a much higher level of carbon without losing appreciable chromium by way of oxidation. In the normal process, therefore, charge consisting of stainless scrap, high carbon Fe-Cr, or charge chrome along with ordinary scrap may be melted in an arc furnace and the molten metal, as free from slag as is feasible, is transferred to the decarburizing unit for further refining. The VOD technique can successfully decarburize the bath with transfer carbon at approximately 0.7%–0.8%. The AOD, CLU, and MRP techniques can deal with transfer carbon as high as 2.0%. These are the significant achievements in stainless steel making technology by adopting the scheme of duplex refining, melting in electric arc furnace, followed by decarburization and finishing by any one of the above-mentioned techniques of secondary refining. It is for these achievements that cost of stainless steel making has continuously decreased at a faster rate than that of any other steel making process.

In the absence of stainless steel scrap, even BOF processes have been used as primary processes along with this technique to produce stainless steel using charge chrome as a source of chromium.

These modern techniques of decarburization have decreased the Fe-Si consumption, and even if an additional cost item of a vacuum or argon is added, the overall process has proved to be more economical.

22.6.2. AOD Process

This is carried out in a special AOD converter which is solid bottom vessel with tuyeres provided on the sides. The tuyeres are few in number, depending upon the capacity. The vessel is lined with a basic magnesite refractory. Molten charge from an arc furnace is transferred to this vessel and blown, with argon-oxygen mixture, the proportion of which varies from 1:3 in the beginning through one or two discrete stages to 3:1 for the final part of the blow. The temperature of the bath rises at the end of first stage to nearly 1710°C, and coolants such as nickel, stainless scrap, other additions, HC Fe-Cr, etc., are added in predetermined proportion to finally arrive at the correct turn-down temperature. The heat is finished by deoxidation by Fe-Si addition. It also reduces back whatever chromium is in slag at the end. The chromium recovery is, therefore, plus 97%.

The average consumption of argon is 18–20 N-m^3/t. A major change in blowing practice has been substitution of argon by nitrogen in the first stage. Many stainless steels have high nitrogen specifications (0.025%) and hence nitrogen substitution of argon can be done even in the later stages also. Any nitrogen absorbed in the first stage is desorbed by argon in the later stages. With nitrogen substitution, argon consumption may come down to 8–10 N-m^3/t.

Total duration of an AOD heat is approximately 2 hours, and the lining life is approximately 80 heats. It is also effective in obtaining good sulfur removal by using argon stirring at the end.

22.6.3. VOD Process

The VOD system essentially consists of a vacuum tank, a ladle furnace with or without argon stirring, a lid with oxygen lancing facilities, et cetera. The ladle has a free board of approximately a meter to contain violent agitation of the bath during lancing. The charge ingredients are similar to those in the AOD process. The charge is melted in an arc furnace and the molten metal with approximately 0.7% –0.8% carbon transferred to the VOD system. Oxygen blowing and perhaps argon bubbling from the ladle bottom are commenced when the required vacuum is established. Argon stirring seems to be necessary, otherwise decarburization is delayed due to a lack of mass transport of carbon from the bottom portion to the surface where a carbon–oxygen reaction is feasible. The carbon can be lowered to approximately 0.02% at approximately 15% to 18% chromium by this technique at a temperature level of approximately 1600°C, as was shown in Figure 22.4.

At the end of refining, the vacuum is broken and the bath is deoxidized with Al and Fe-Si. Desulphurization can be carried out by putting synthetic slag of approximately 2% –3% by weight of the metal charge and argon purging would result in 80% of the sulfur removal, finally producing 0.01% sulfur content. The total VOD cycle lasts for approximately 2 to 2.5 hours. The chromium recovery is plus 97%.

Since many of the stainless steels are required to be vacuum-treated to decrease the gas content, the vacuum system can be easily modified to incorporate oxygen-lancing facilities and thereby VOD can be brought about for producing extra-low-carbon steels, without much extra additional investment.

22.6.4. CLU Process

The process was developed jointly by the Creusot-Loire of France and Uddeholm of Sweden and hence the name.

The charge, as usual, is melted in an electric arc furnace and then transferred to a Bessemer-type vessel with bottom blowing facilities. Refining is entirely carried out in this vessel, which has a chrome magnesite lining. A mixture of oxygen, steam, argon, nitrogen, and air is blown from the bottom. The compositions of the mixture are preset for the entire blow, depending upon the chemistry of the bath. The presence of steam controls the temperature to approximately 1650°C–1680°C. Whatever chromium is oxidized earlier is reduced back by the addition of Fe-Si and Cr-Si as in the usual practice. Carbon contents of approximately 0.015% have been achieved.

The major advantage of the process is its capacity to use a cheaper medium such as steam partly in place of costly argon. So far only one 70-t CLU vessel coupled with a 50-t arc furnace is in operation at the Uddeholm's Degerfors Works in Sweden.

The applicability of such a process using the already idle Bessemer converters (from the duplex plant of SMS3, TISCO) could be assessed to increase stainless steel production under Indian conditions. It might give an additional lease on life to the Bessemer shop, which otherwise is about to be dismantled.

Considering the ill-effects of blowing steam, the process has now been modified and now resembles the AOD process more.

22.6.5. MRP Process

The Germans have pioneered to put the OBM process in practice wherein oxygen is blown from the bottom using special designs of tuyeres. The chief achievement of this process has been the establishment of near-equilibrium conditions between slag and the bath, due to the excellent stirring caused by bottom blowing. The same technique has been used to make stainless steel by blowing oxygen and argon mixtures from the bottom. Although the metallurgy is similar to that of the AOD process, the efficiency is far better than that of AOD.

Two MRP vessels, each of 8-t capacity, have been successfully working, one each at M/s PHB Weserhute, St. Inbert/Sarr and M/s. C. Grossmann AG, Solingen-Wald in FDR Germany. These have been producing all varieties of plain, low-alloy, and high-alloy steels containing even very low carbon contents.

The blowing strategy, that is the proportion of oxygen and inert gas such as nitrogen and argon, is carried out automatically as per a predetermined program. The program is synchronized with the angular position of the vessel. Nitrogen is used as a diluent in the first stage and argon in later stages. Argon flush is used in the end to finally clean the bath, adjust the temperature, make effective additions for composition adjustments, and reduce the carbon to a level lower than 0.05%.

This MRP process is essentially the metal refining process recommended for refining and making steel from molten charge of almost any composition.

22.6.6. SR-KCB Process

The name is derived from the Kawasaki Steel Corporation where it was developed and put into practice. It means *Smelting Reduction Kawasaki Combined Blowing*.

This is a direct method of stainless steel making in which chrome-ore is used with hot metal as charge. The hot metal is dephosphorized beforehand. Coke is used as a reducer to reduce chrome-ore. This is essentially a reduction-smelting operation. A combined blowing vessel is used to refine the carbon in preference to Cr. Final finishing carbon removal and alloy adjustments are done in VOD. Several such units are in operation in Japan.

22.6.7. Nitrogen Problem in Stainless Steel Making

New and cheaper varieties (200 series) of stainless steel have recently been developed to substitute the traditional Cr-Ni stainless steels. The role of Ni in stabilizing austenite phase at room temperature in these steels is replaced, at least partially, by nitrogen, manganese, and copper. The cost reduction is maximum when the maximum amount of nitrogen is used for the substitution. The level of nitrogen is therefore raised to 0.2%–0.3%, or 2000–3000 ppm Nitrogen is known to cause strain brittleness in steel. That is the reason it was kept at a bare minimum for most steels. It creates serious problems in deformation during rolling. It is an interstitial element and strains the iron lattice. This was the reason that basic-Bessemer steels were not preferred for many applications, since it contained as high as 0.018 % nitrogen. It was difficult to reduce nitrogen content of steels then. It is equally difficult to get nitrogen at very high levels dissolved in steel for making stainless steels.

Standard grades given in the literature are given in Table 22.2.

Table 22.2

Shows the standard stainless steel specifications of the 200 series

Grade	*UNS Nos.*	*C*	*Mn*	*Si*	*Cr*	*Ni*	*P*	*S*	*others*
AISI 201	S20100	0.15	5.5–7.5	1.00	16–18	3.5–5.5	0.06	0.03	0.25N
AISI	S20500	0.12–0.25	14–15.5	1.00	16.5–18	1–1.75	0.06	0.03	0.32–0.40N

In the 200 stainless series, the following chemistry has also been developed, in India, and used:

Cr – 14-16%, Mn – 5-6%, Si much less than 1%, C – 0.1% and N_2 = 2500 ppm

Such local specifications are developed on the basis of achieving that chemistry that will somehow ensure austenitic structure at room temperature, when cooled. In the standard 300-series stainless steel, the austenitic structure is guaranteed by the presence of 9% Ni. Any substitution of nickel for economic or other reasons is judged on the basis of nickel equivalent. Various formulae are available and used for evaluating this equivalent. These are purely empirical in nature and established out of one's own practical experience. They need not be taken as universal truths.

One such formula is

$$\%\,\mathrm{Ni_{eq}} = \%\,\mathrm{Ni} + 0.31(\%\mathrm{Mn}) + 22\%(\mathrm{C}) + 14.2\,(\%\mathrm{N}) + \%\,\mathrm{Cu}$$

An excellent review[42] gives

$$\%\,\mathrm{Ni_{eq}} = \%\mathrm{Ni} + \%\mathrm{Co} + 0.1\%\,\mathrm{Mn} - 0.01\%\mathrm{Mo}^2 + 18\%\mathrm{N} + 30\%\mathrm{C}$$

Similarly minimum dissolved N required for austenitic structure is given by

$$\% N = -0.88\%C + 0.046\%Cr - 0.0009\%Mn + 0.038\%Mo - 0.053\%Si + 0.082\%Ni - 0.208\%Cu - 0.032\%W - 0.078$$

It has also been suggested that

$$(\%C + \%N) = 0.078\ (\%Cr - 12.5)$$

holds true for Mn contents in the range of 5% –14%.

This value of Ni must be 9%, made out of the value of the other elements in the proportion shown. Thus for the above-mentioned chemistry when 2500 ppm N is ensured at 5%–6% Mn, then no nickel would be required to ensure austenitic structure. The proportion of Ni on the one hand and that of the Mn and N on the other can thus be suitably adjusted to ensure austenitic structure.

The problem of making stainless steel of 200 series lies in getting required nitrogen dissolved in steel, as it is also prone to desorbs from steel equally readily (i.e., its reverse reaction).

At the temperature of interest in steel making, gaseous nitrogen exists primarily as diatomic molecules, which dissociate on dissolution in liquid iron as:

$$\{N2\} = 2N$$

and for which the equilibrium constant is

$$K_N = h_N^2/P_{N2}$$

If the dissolution obeys Henry's Law it can be expressed as

$$wt\%\ N = (K_N \times P_{N2})^{½}$$

where K_N is Equilibrium constant of Nitrogen dissolution as above and P_{N2} is Partial Pressure of Nitrogen

This is known as Sievert's Law, which states that the amount of nitrogen dissolved in iron will be proportional to the under root of the partial pressure of nitrogen gas in contact.

The solubility of nitrogen in steel is governed by

$$wt\%\ N = (K_N \times P_{N2})^{½}/f_N$$

where $f_{\underline{N}}$ is the Activity Coefficient of Nitrogen

$$\text{Log } K_N = -(188.1/T) - 1.246$$

It means that the amount of nitrogen dissolved in iron when pure nitrogen gas, at one atmosphere pressure, is bubbled through the bath, is 0.045% or 450 ppm, at best, no matter how long the operation is carried out. So in steel making in the initial stages of refining when the steel composition is tending towards pure iron the amount of nitrogen dissolved in the melt will never exceed 450 ppm. This has been observed in practice to be true. In fact, the actual value will be much less because of non-attainment of equilibrium. The non-attainment of equilibrium is due to a lack of sufficient volume of nitrogen blown and/or for insufficient duration of time.

The situation can be somewhat redeemed by taking advantage of thermodynamic interactions of nitrogen with other solute elements such as, in particular, Mn and Cr which are, even otherwise, present in sufficient amount and which reduce the activity coefficient of nitrogen in steel drastically and thereby increase the solubility of nitrogen in steel. The solubility is also affected by temperature. In general, solubility of nitrogen in steel increases with temperature. This factor can also be exploited in obtaining desired solubility of nitrogen in steel while making stainless steel. Nitrogen-bearing cheaper stainless steel can best be made by the AOD process only, wherein nitrogen is used in place of argon. In fact, it then becomes the *nitrogen-oxygen decarburization NOD* process.

22.6.8. Stainless Steel Making in EAF

Stainless steel was once routinely made in EAF, before the advent of the AOD process. The key lay in decarburizing the bath, in the presence of and in preference to chromium oxidation, by raising the temperature to nearly 1800°C. When top oxygen blowing is available in a modern EAF practice where hot metal is included in the charge, the same top blowing arrangement can be used for refining with little ingenuity. Instead of blowing pure oxygen alone, a mixture of air and oxygen may be blown from the top, in the same proportions, as in AOD, to achieve the same desired N_2/O_2 ratios in place of A/O_2 ratios for blowing. This will also result in generating the desired reduced partial pressure of CO in the ambient atmosphere to expedite decarburization in preference to oxidation of chromium. This is not only cheaper but it obviates the necessity of raising the temperature of the bath as is otherwise required under pure oxygen blowing. Pure oxygen blowing, from the top, without raising the temperature, is most likely to lead to excessive oxidation of chromium. It may be remembered that the solubility of Cr_2O_3 in basic slag is very poor (5% –8% only). Slag containing Cr_2O_3 even otherwise, is relatively more viscous because of the refractory nature of this oxide. The excess of the refractory oxide, when present in slag, makes the slag far more viscous and unworkable; in fact at times it behaves as a solid.

It is, however, not an easy task to get nitrogen dissolved in steel, even when it contains sufficient Cr and Mn. The rates of dissolution of nitrogen in iron as estimated using a CFD code and actually measured in experiments are shown in Table 22.3.

Table 22.3[43]

The Rates of Dissolution of Nitrogen in Iron

Sr No.	*Time (Min)*	*Gas Flow (m^3/min)*	*N_2 (kg)*	*Change in N (meas)*	*Change in N (theor)*
1	10	0.571	6.96	0.0065	0.0077
2	15	0.371	6.79	0.0060	0.0071
3	15	0.713	13.05	0.0140	0.0150
4	9	0.856	9.39	0.0095	0.0100

The table shows the amount of nitrogen picked up by molten iron as estimated by a CFD code and as actually measured by an experiment [43].

This will be a useful guide for those making stainless steel to assess the nitrogen pickup under a given set of conditions.

22.6.9. Concluding Remarks on Stainless Steel Making

The AOD process has now been universally accepted as the most efficient process of stainless steel making. This is universally used for regular stainless steel production of all different grades. This has therefore superseded all the rest of technologies that were in use, in terms of involved economies. Any new plant is bound to adopt the AOD technology only. Any other technology, however, may be used, but only for occasional heats when AOD is not available. Any alternative route (e.g., EAF), therefore has to have the same underlying principles employed in carrying out the stainless steel making heat. The EAF-VOD facilities can be employed for stainless steel making with the above in mind.

22.7. Injection Metallurgy

The ever-decreasing sulfur specification has always been a challenge to steelmakers but the introduction of injection techniques has solved it fairly satisfactorily. In injection metallurgy a strong desulfurizing reagent, in the form of fine powder, is injected in the refined steel bath held in the transfer ladle, along with an inert gas as carrier. The enormous interfacial area of contact of particles with the bath leads to very efficient interaction of reagent with the bath. Efficient desulfurization can be brought about in a short time. It takes approximately 8 to 10 minutes for this treatment, and the bath loses 30°C to 35°C in temperature.

The injecting reagents may be metallic, such as calcium, magnesium, et cetera, in one or the other form, such as calcium silicide and magnesium coke and so on or may be lime, calcium carbide, et cetera, in some suitable form to produce a fluid oxide melt product. It is now common practice to inject desulfurizing agents along with strong deoxidizing reagents since both together give excellent desulfurization as well as deoxidation. Consequently the improvement in toughness and elongation of the steel is not only due to the decreased sulfide inclusions but it is also due to the transformation of quality-impairing oxide and sulfide phases into less dangerous types of inclusions. The lines of sulfides and aluminosilicates in conventional steel transform to more globular in shape and change the chemistry from alumino to calcio silicates, which are non-deformable at rolling temperatures. Various other additions are also made in the injection material to achieve this end. Therefore, although the base composition of the reagents is known, the actual composition is a trade secret of the manufacturers.

Various injection systems are available in the market but the TN (Thyssen Niederrhein) system is more popular. In all these, the injecting lance is made of ceramic material, and argon is used as a carrier gas. The systems are similar in design and operation and hence can be adopted readily in an existing plant. The TN system is schematically shown in Figure 22.5. Injection metallurgy is practiced not only for removal of sulfur but also for inclusion modification, since it favors removal of non-metallics from a steel bath.

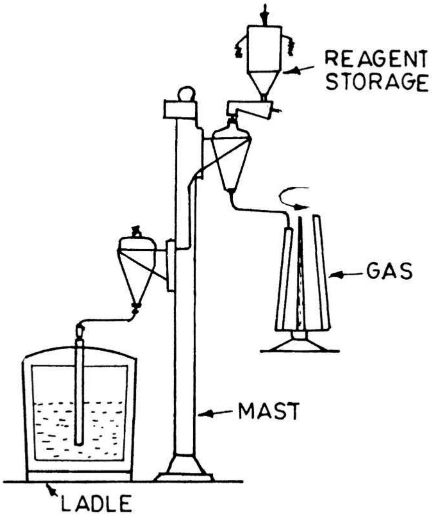


FIGURE 22.5. The TN system of injection.

For effective desulfurization, steel melt has to be deoxidized fully. In fact most of the desulfurizing agents are also very effective deoxidizers, viz., Ca, Mg, C, etc. Good calcined lime is also a good desulfurizing agent. The following table shows the details of these reagents, along with their normal rate of their injection:

Table 22.4

Desulfurizing Reagents

Reagent	*Composition %*	*Injected amount (kg/t steel)*
Ca-Si alloy	Ca-30, Si-62, Al-8	2–4
$CaO-CaF_2$	CaO-90, CaF_2- 10	3–6
$CaO-A12O3-CaF_2$	CaO-70, A1203- 20, CaF_2-10	2–5

Usually when sulfur specification levels are low, the residual Si and Mn contents that are specified are also very low. Even if they are not low, they by themselves do not deoxidize the bath to the extent required for effective desulfurization. In practice, therefore, the bath is invariably deoxidized by at least Al, if not by any other still stronger deoxidizer. Although Al is added in the form of big blocks or cubes, a sizable amount is still fed in the form of Al-wire. The rate at which it is fed into the bath is such that it travels in solid state well below the bath surface, before it melts, and is available for reaction with dissolved oxygen. Once it melts below the bath surface it is very effective in deoxidizing the bath as the droplets of Al, being lighter, travel upwards and react with dissolved oxygen. It is therefore a very effective deoxidizer when fed in wire form. However, it is costlier than charging block aluminum.

After deoxidation by Al is over, the products of deoxidation (i.e., Al_2O_3) remains largely dispersed in the melt as non-metallic inclusions, and in that form it is very detrimental in continuous casting, as it leads to the formation of alumina streaks in the product. It is equally detrimental to steel properties. Similarly, the products of desulfurization in the form of CaS may also partly be in suspended form, ultimately leading to non-clean steel. This problem of alumina and sulfide inclusions, along with other inclusions, in steel, is fairly satisfactorily subsided by adding Ca or Ca-Al at the end of the desulfurization operation. This is done by introducing Ca or Ca-Al wire in the bath in the same way as Al-wire is introduced. This modifies the morphology of alumina-sulfide inclusions from their odd acicular shapes to globular ones, which do not pose that serious a problem in rolling as the odd shapes do.

22.8. Plunging Techniques

Some simpler techniques have recently been developed to either desulfurize the steel bath in the transfer ladle or to alloy it with some micro-alloy additions. This is achieved by what is called as plunging techniques. In this a small crucible containing the reagent is attached at the end of a refractory-protected rod. The crucible is plunged in the bath upside down and held inside the bath for the required duration, during which the reagent is expected to interact with the steel. This technique is adopted when the total amount of reagent to be added is so small that it cannot be injected with a carrier gas or the addition is not a routine process required in the plant. The injection process is now more popular since it has become a routine matter to desulfurize the steel bath.

Reagents such as magnesium-coke briquettes or REM (rare earth metals) are plunged in refined steel bath for desulfurization on a commercial level. Recently the NML, Jamshedpur, has developed briquettes of magnesia plus coke for similar purposes. This substitutes the costly magnesium by cheaper magnesia and hence is much cheaper.

Virgin metals or ferro-alloys can similarly be added in the steel bath for achieving micro-alloying additions.

22.9. Post-Solidification Treatments

Quality improvement of steel can also be brought about after steel is refined and cast into ingots from the primary refining furnace, by re-melting and casting once again. A typical example of this type is zone refining, which is adopted to produce purer metals.

The other two techniques that have been developed are meant for the production of not pure metals, but alloy steel of better cleanliness and low sulfur contents. The vacuum arc re-melting, VAR for short, and the electro slag refining, ESR for short, are commercially used for further refining of steel after these are cast into ingots.

Although ESR was used as a commercial process for some special applications for a while, finally both of these techniques have remained confined for research and very special applications only.

22.9.1. VAR and ESR Processes

In both of these processes, the steel ingot produced by the primary refining forms the electrode to be drip-melted into a water cooled copper mold. In VAR, melting is carried out under vacuum and in ESR it is in open atmosphere. In VAR, arc is struck between the electrode and the mold, and it generates the heat required for melting the electrode. In ESR, a slag layer is used to act as a resistor between the electrode and the mold and which is responsible for melting the electrode. The slag also acts as a refining agent. In both of these processes the electrode melts progressively and is resolidified on the mold, nearly unidirectionally. Because of the high temperature, a small pool of molten metal and almost unidirectional solidification, both of these processes can produce sound ingots of high density. The composition of the product is nearly the same as that of the original material but with improved cleanliness, decreased segregation, and with practically no cavities. The ingot size ranges from about 200 to 1500 mm on an industrial level.

The product of both of these processes is exceptionally suited for the production of forgings of high-alloy steels. But because of the high cost of such a process, applications are limited to specialty products such as turbo rotor shafts and so on.

In VAR, the hydrogen and oxygen contents are very low, but in ESR they are like ordinary steel. In ESR, the choice of the slag composition is fairly critical since it has to act as a resistor as well as a refining agent. These are essentially oxy-fluoride-type reducing slags such as $CaO-CaF_2$.

The ESR, however, has some advantages over VAR, and these are given below:

1. Multiple electrode can be melted into a single electrode.
2. Spacing between the mold wall and the electrode is not critical.
3. Surface quality is superior, requiring little or no conditioning.
4. Steel can be desulfurized to as low as 0.002% sulfur.
5. Round, square, hollow, and rectangular shapes of ingots can be produced.
6. Ingots of much larger weight can be produced.

The ESR, therefore, gained popularity during the seventies, and several such units were installed the world over. In India an ESR unit capable of producing 250-mm diameter round ingot of high-alloy steel was installed at the Firth (India) Steel Co., Nagpur, in the early 1970s. Another unit was installed at TISCO, Jamshedpur. Even these units find difficulties in selling their product because of its prohibitive cost. On an average, the ESR steel costs almost two to three times that of the same steel produced by the conventional primary process. Much of the earlier VAR and ESR market has, therefore, been captured by the other secondary processes.

22.10. Special Improvements in Steel Making Practice

It is alright to assume in theory that after steel is fully refined and finished to the extent possible in a steel making furnace it is tapped free of any refining slag. In reality it is most difficult to tap steel free of refining slag. This slag is oxidizing in nature and may contain phosphorus. If deoxidation is carried out, of steel in contact with such slag in the ladle, phosphorus may revert and create problems beyond repair.

It is estimated that the slag carryover from a BOF converter may be around 1.2% –1.5% of the steel weight. If proper precautions by way of use of plug-slag ball and plug-slag skimming machine are used, then this can be reduced to 0.5% but not to less than that. The ill effect of this slag to some extent can be offset by using useful slag cover made by additions of suitable components such as CaO, $A1_2O_3$, CaF_2, and so on, which enhances the state of deoxidation of the bath. Such a slag may amount to nearly 15–20 kg/metric ton steel. This slag has capacity to absorb sulfides and oxides generated by deoxidation. All these help to obtain better clean steels and minimize deoxidizer consumption. Similarly, silicon from slag may also revert if deoxidation is very deep and a considerable amount of Al is left in the metal bath.

22.11. Tundish Metallurgy

Tundish is that simple device where liquid steel stays for a while before entering the mold. Liquid steel is tapped in teeming ladles where much of the finishing operations such as deoxidation and alloying are carried out and where the steel is held for some time to allow the products of deoxidation to rise and join the slag cover at the top. Steel is then teemed into the mold via a tundish. Earlier it was thought to be only a device to maintain standard ferrostatic head of steel for teeming and at best a metal distributor for multiple strand continuous casting. Soon it had been realized that during the stay of steel in tundish it can be utilized for further removal of inclusions with or without additional deoxidation. The tundish design and operation should therefore be aimed to carry out this job. It is known that gentle stirring and increasing the residence time of steel in tundish can lead to this achievement to some extent.

The tundish is therefore designed to improve steel flow patterns to help coalescence and floating of inclusions. It is also provided with synthetic slag cover to absorb inclusions. Ceramic filters may be used to trap inclusions. Vortexing is avoided by having suitable depth of steel in the tundish. Suitable flow control devices such as dams, weirs, and slotted baffles are also used.

Besides this, some metallurgical operations such as final deoxidation adjustments, desulfurization, trimming alloy additions, inclusion modifications, and temperature homogenization and control, etc., are also possible to be carried out to obtain better chemistry, inclusion control, and temperature to finally produce better-quality casting. All these metallurgical operations are now clubbed as “tundish metallurgy.”

Tundish can also be a source of silicon pickup if proper precautions are not taken. It is general practice to use some insulating material layer on the steel bath in tundish to decrease heat losses and thereby to reduce the temperature loss of steel at the point of teeming. The lower this loss is, the better it is for casting. Many times rice husk is used for this insulation.

It is very rich in silica, and being very active, it somewhat gets reduced, most probably by the Al in the melt, causing silicon content to rise and spoil the end chemistry. If insulation is to be used, then it is better to use a thin liquid slag layer first on the bath surface and then use rice husk for insulation. This way, the silica has no chance to get reduced by dissolved aluminum.

22.12. Secondary Refining Furnaces

Secondary refining is carried out even in a transfer ladle fitted with a porous plug for argon purging. An argon supply line is connected to this plug, just before tapping. A refractory lance can be inserted in the transfer ladle to supply argon if a plug is not provided. Similarly, injection treatment can be carried out in the transfer ladle itself. However, the bath tends to lose temperature in all these treatments and hence these cannot be carried out for too long, even if required. A ladle furnace has been designed in the recent past to carry out most of the secondary refining and has proved to be most suitable and economical as well.

22.12.1. Ladle Furnace (LF)

It is a simple ladle-like furnace provided with a bottom plug for argon purging and a lid with electrodes to become an arc furnace for heating the bath. Another lid may be provided to connect it to a vacuum line, if required. Chutes are provided for additions and an opening even for injection. In short, it is capable of carrying out stirring, vacuum treatment, synthetic slag refining, plunging, injection, etc, all in one unit without restraint of temperature loss, since it is capable of being heated independently. Every ladle furnace need not be equipped with all these arrangements. As per the requirements of refining, the ladle furnace may be provided with the necessary facilities. For example, if gas content is not a consideration, the vacuum attachment may be eliminated. The principal component of the facilities are shown in Figure 22.6 schematically.

Several LF designs are, therefore, available in the market. The ASEA-SKF furnace is a special variety of LF furnace only.

The SKF furnace is essentially a teeming ladle for which additional fittings are provided. The metal in the ladle is stirred by an electromagnetic stirrer provided from outside. The ladle shell is made of austenitic stainless steel for this reason. Two ladle covers are employed. One of these fits tightly

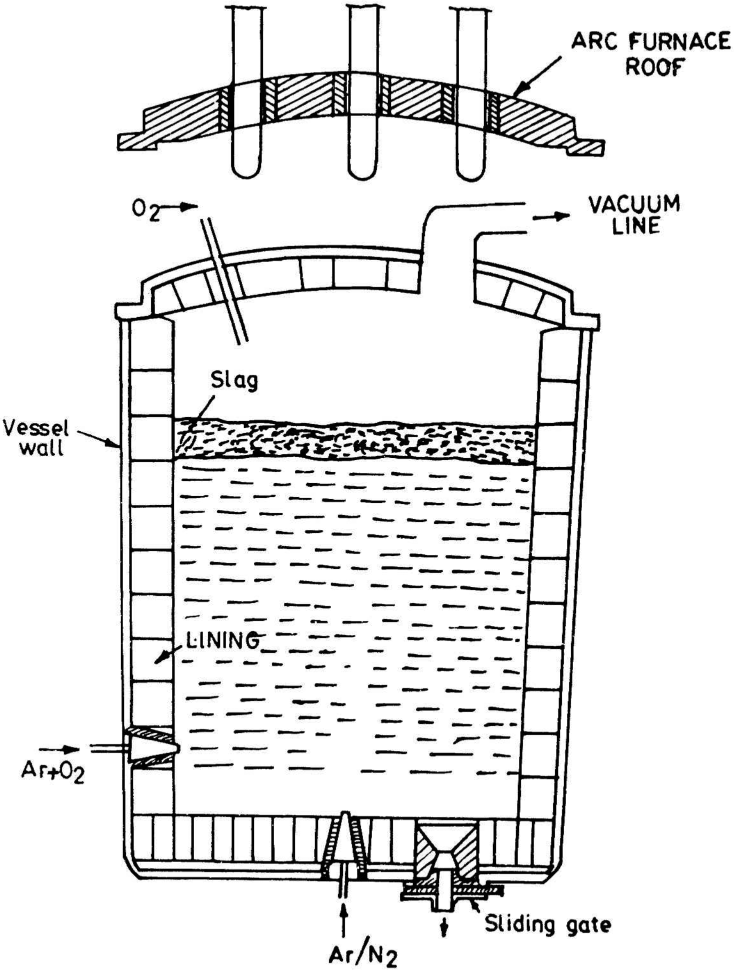


FIGURE 22.6. Principle components of a ladle furnace facility.

on to the ladle, forming a vacuum seal, and is connected to a steam ejector unit for evacuation of the ladle chamber.

For vacuum decarburization, an oxygen lance is introduced through a vacuum-sealed port located in the cover.

When the decarburization and vacuum degassing is over, the first cover is replaced by the second cover, which contains three electrodes. Final alloying and temperature adjustments are then made. Steel can also be desulfurized by preparing a reducing basic slag under the electrode cover.

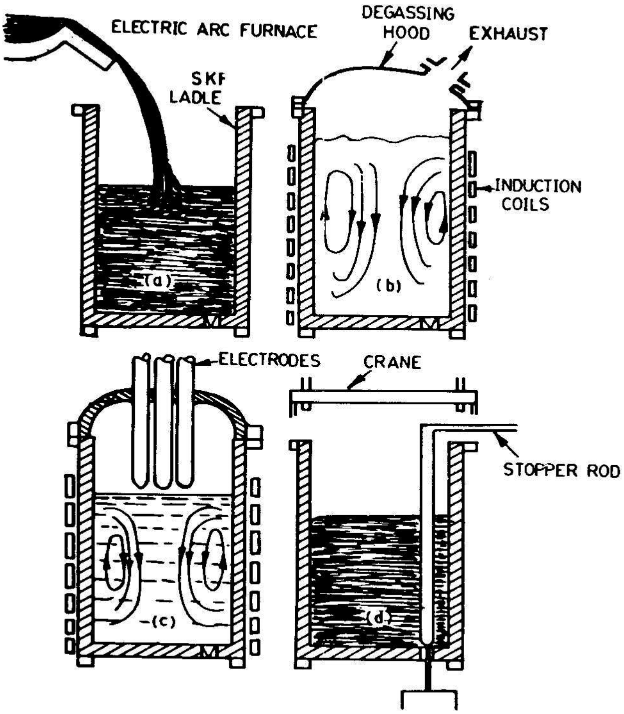


FIGURE 22.7. The scheme of operation of an SKF furnace.

The process is schematically shown in Figure 22.7. The nearly refined steel in only one of the primary steel making processes can be treated in this furnace by carrying out the following operations:

(*i*) Tapping primary furnace into the SKF ladle directly.

(*ii*) Controlled stirring during the entire secondary processing.

(*iii*) Vacuum treatment, including minor decarburization.

(*iv*) Extensive decarburization for stainless steel making.

(*v*) Deoxidation.

(*vi*) Desulfurization and deslagging.

(*vii*) Alloying to desired extent.

(*viii*) Temperature adjustment.

(*ix*) Teeming from the same SKF ladle.

22.13. Concluding Remarks

The secondary refining processes that can be carried out in the transfer ladle itself can be coupled with any of the primary processes of steel making even in the existing plants without many problems. These treatments improve qualities of steel product in one or more ways. Although it is possible to couple other secondary treatments with any of the primary processes of steel making, the coupling is relatively easier with electric arc furnace practice. Coupling with modern BOF processes is likely to pose problems of synchronization of high-speed refining in these primary processes with relatively slow secondary processes. Such a high rate of production of quality steels may not even be necessary. Coupling with large-capacity primary furnaces may be feasible but would need equally large secondary furnaces. In spite of these difficulties, several small LD vessels have been coupled for secondary treatments at some of the well-known plants. The secondary refining in such cases must be synchronized with the primary unit, and in that sense whatever time is available, the secondary refining strategy will have to be completed within that available time.

The coupling of secondary refining with electric arc furnace has not only improved the quality of product but it has also resulted in the following advantages:

1. Better utilization of the arc furnace, preferably using HP or UHP transformer, as a melting unit alone and thereby raising the overall production rate of the arc furnace by investing proportionately a small amount in installing the secondary unit.
2. The melting and refining are carried out in two different units, each powered by just the required size transformer and hence the overall power factor is more favorable.
3. Better furnace life.
4. The electrode consumption is low.
5. The sulfur specification of the charge material in general and of carbon for stainless steel making is much more relaxed.
6. Special desires of customers and local conditions in the steel making shop can be accommodated.

In general, the effects obtainable by secondary treatments are varied in terms of cleanliness, gas content, decarburization, desulfurization, deoxidation, composition range, temperature, etc. This is essentially due to the type of treatment that is adopted. There, therefore, exists a large number of combinations of requirements and accordingly a very wide range of equipment for secondary treatments is available in the market for the purchaser.

The extremely rapid slag-metal and gas-metal reactions in the bottom blown processes may be viewed in proper perspective for its application in secondary treatments. The MRP technique will be watched with great interest in this respect.

CHAPTER 23

CASTING PIT PRACTICE

In This Chapter

- Introduction
- Deoxidation Practice
- Casting Pit Design
- Teeming Ladle
- Ingot Mold
- Teeming Methods
- Pitside Practice at Rourkela Steel Plant (India)
- Production of Large Ingots

23.1. Introduction

Until only a couple of decades ago, much of the refined and finished steel was cast into permanent metallic (made from iron) molds to produce ingots for their subsequent rolling or forging. In a steel foundry it was poured into metallic or sand molds to produce finished or semi-finished castings, the proportion of which, in the total liquid steel produced, was and is still very small. This is known as the conventional casting process. This process of casting into ingots and their subsequent mechanical working involved use of soaking or reheating furnaces to maintain the proper temperature of the ingot before its working and it also resulted in great loss of metallic yield. This process has almost been replaced in integrated steel plants and even in mini-steel plants almost completely by the modern, more efficient, and high-yielding continuous casting of steel into billets, blooms, or slabs and now even in the form of thin slabs and sheets. The conventional process of casting is now seen only in old small steel plants using scrap and/or DRI as charge materials. The ingots so produced are small pencil ingots of cross section 100 by 100 mm or a little more.

The liquid steel refined, particularly in an open hearth–type furnace was then deoxidized during tapping in special ladles, called teeming ladles. It was also alloyed during tapping to a small extent, if required. Many times the liquid steel was also exposed to vacuum either in the ladle or during casting. The entire processing of liquid steel from the time of tapping to its solidification in ingot molds, including the reconditioning of the molds for its subsequent use and maintenance of relevant equipments in the shop was then known as *casting pit practice* or *pit-side practice.* The term *pit* refers to the old designs of sunken pits in which the molds were kept for casting ingots and the operation on this side of the steel melting shop was called pit-side practice. This old type of operation is almost extinct, but the entire operation of liquid steel after tapping till solidification continues to be called in the same traditional context.

The procedure for casting steel in a continuous casting machine is quite different from the traditional ingot casting process. The teeming ladle in principle remains the same. The solidification process cannot change by such a change. Many of the defects that arise in solidified steel are also common in both. The traditional process is therefore described, in fair details here, to understand the underlying principles.

23.2. Deoxidation Practice

The theory of deoxidation has already been described in Chapter 4, and the various deoxidizers and their applications are discussed in Chapter 5.

Complete deoxidation of a refined steel bath inside the furnace is possible only in an electric furnace process and even in this process a small amount of strong deoxidizer(s) may still be added in the ladle during tapping. In the rest of the processes a certain proportion (minor) of the deoxidation may be carried out in the furnace and the remainder (major) in the ladle during tapping. The required amounts of deoxidizers, recarburizers, and alloying elements are weighed and kept ready in bags for charging during tapping. The first bag is not thrown in the ladle until it is one-third full. All the additions must be over by the time two-thirds of the ladle is full. The turbulence created by the tapping stream helps to homogenize the composition of the metal in the ladle.

In general, blocks of aluminium and shots and lumps of Fe-Mn and Fe-Si are added as deoxidizers. The other additions may be as per requirements to meet chemical specifications.

Steel is held in the ladle for nearly 10–20 minutes after tapping is over and before teeming commences. By this time the products of deoxidation are expected to rise and stratify at the surface of the metal. The tapping temperature should be high enough to allow the additions and holding period and still be left with adequate superheat to obtain good castings after teeming.

23.3. Casting Pit Design

In the earlier designs, the molds were arranged in a row in a fixed refractory–lined pit in front of the furnace. The depth of the pit was a little over the height of the mold. The term *pitside* originated from these sunken pit designs. The molds were prepared *in situ* and kept there until they were stripped. Such a design involved a great deal of wastage of space, and the heat retained by the pit made conditions around it more difficult to work. The design was alright for slow processes such as conventional open hearth and electric furnaces. In modern steel making, as much as a few hundred metric tons of steel are to be teemed every hour. The pitside area required for fixed pits to cater for such large production rates would be exceptionally large and would amount to wastage of a great deal of space in the shop.

The only feasible alternative for a modern steel-melting shop is to have a moving pit in the form of open bogies. The molds are set on these bogies, which are brought to the teeming platform just before teeming. The level of the platform is slightly higher than the mold top level so that the teemer standing on the platform can easily watch the mold during teeming, add the additives to the mold, and handle the lids for mechanical capping of the mold. A set of molds are brought to the teeming platform for teeming one full tapping of the furnace. It includes molds equal to the number of full ingots expected from a heat and a few additional molds to teem the left over metal (*i.e.*, to produce short or spare metal ingots). Before teeming commences, the teemer should know the number of full ingots he is expected to teem. Short ingots are not desirable as they may not cater the order. Efficiency of a teemer can be assessed in terms of how few short ingots are produced during teeming; the more there are, the less is the efficiency.

The teeming ladle is either held in a overhead crane, a ladle carriage, or a ladle stand. These are positioned in such a way that the ladle stands just above the molds. The ladle carriage is moved on rails parallel to that on which pit bogies are brought. The use of a ladle carriage or a ladle stand is desirable if a large heat is tapped in more than one ladle for teeming. It is also used if an ingot is to be cast from more than one tappings in the shop. The teemer maneuvers the stopper rod of the teeming ladle from the teeming platform.

After the heat is fully teemed, the bogies are carried away to the stripper yard where molds are stripped and the ingots are transferred to the soaking pits for conditioning before rolling or forging. It is not necessary to wait for stripping until the ingot is entirely solid. Many types of steel ingots are stripped when the core is steel molten and the temperature is equalized in the soaking pits. The molds are reconditioned for use again. A modern pit-side, therefore, includes a stripper yard and, mold reconditioning, and assembly bay. Readymade molds are sent to the teeming platform after reconditioning and assembly on bogies as per requirements. This is also known as a *bogie casting* system as against the old sunken-pit casting. The stripper yard and the reconditioning and assembly bays may be located under separate shade or at one end of the steel-melting shop. A very efficient system of railway tracks is essential to achieve to and from movements of the bogies. In spite of all care, accidents may occur and repair work may go on, but the normal working of the pitside should continue unhindered, and this is the acid test of a well-designed pitside layout.

23.4. Teeming Ladle

Although sometimes a heat is tapped in more than one ladle and more than one heats may be tapped in the same ladle, by and large a heat is tapped in just one ladle. The ladle is lined with refractory to withstand the tapping temperatures. The ladle design and construction must be rugged to stand the weight of the heat at such high temperatures for over an hour without imparting any undesirable effect on the steel. The general design of a teeming ladle is shown in Figure 23.1. It is a welded or riveted steel

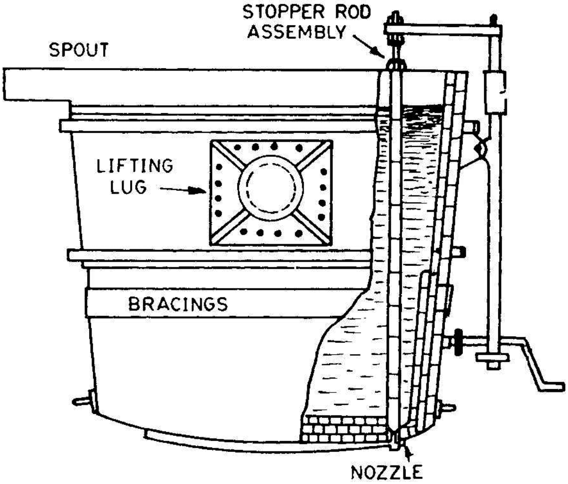


FIGURE 23.1. Teeming ladle for steel.

shell with external reinforcement to strengthen the shell. The refractory lining may be bricked, rammed, or sprayed; bricking is by far most commonly adopted. The teeming ladle has an off center opening(s) in its bottom, equipped with a nozzle(s), a stopper, and a mechanism for raising and lowering the stopper to open or close the opening.

In general, ordinary-grade fire bricks are used to line the ladle. The lining is in two parts viz. bottom and side walls. The lining is laid in such a way as to be able to repair the walls or the bottom, if damaged, independently of each other. The bottom is either flat or sloping towards the nozzle. Several brick shapes are used, for example, normal rectangular, tapered, or curved with interlocking arrangement. Good-quality cement must be used as a mortar while laying the bricks. The joints between the bottom and the side wall happens to be the most vulnerable spot for failure of the lining since more than one layer are used in lining the ladle. The brick joints should not continuously run to the surface, as it is more vulnerable for metal penetration.

The ladle may have one or two nozzles. A nozzle seating block is used to make the opening. The seating block profile is extremely important as it decides the rate of teeming. The nozzle may be fixed in the seating block from inside or outside.

The ladle wall is lined in three courses. A thin layer of ganister next to the shell is followed by a layer of safety lining and then the working lining. Sometimes the working lining at the usual metal surface level is made of high-alumina bricks rather than usual fire bricks, to resist the attack of oxidizing slag, a little of which always comes in the ladle from the furnace. Some ladles have lips to allow excess slag to run over.

Recently, spraying of refractory material to build up the ladle lining has been developed. Similarly, remaining siliceous lining has been used and found satisfactory. The ordinary fireclay brick–lined ladle still offers the best choice.

A freshly lined ladle is thoroughly dried to achieve good life and to eliminate the possibility of hydrogen pick-up by the metal.

There is no sure way to decide that the ladle is due for relining. To be on the safer side these are relined before the due time. A ladle may be used for nearly 6–8 tapping before it is relined. In each tap some metal solidifies in the ladle and is known as skull, which should be carefully removed after the ladle is empty, to prolong ladle life. If the skull is not removed it continues to grow in thickness.

The ladle must be used for the maximum amount of time and their individual use should be so spread that not more than the designed number of ladles are sent at a time for repairs.

The nozzle diameter decides the time required to teem a given-size mold. The time required to teem the required number of molds is thus calculated. The initial teeming temperature should be such that adequate superheat is still left at the time of teeming the last mold (see Figure 4.12). If ladles with a single nozzle prolong the teeming period beyond nearly 45 minutes two-nozzle ladles are used to teem two molds at a time. This helps to have proper metal temperature right through the entire teeming period. The inter-relationships of mold size, nozzle diameter, number of nozzles, initial teeming temperature, et cetera, are established to fix up the correct practice in relation to the ladle size, which is generally equal to the furnace capacity.

23.4.1. The Stopper Assembly

The stopper assembly consists of a stopper, which sits on the nozzle seat from inside the ladle, and the stopper lever assembly to control the life of the stopper for regulating the flow of metal during teeming (*i.e.*, teeming rate). Efficient stopper assembly design is essential for proper teeming practice. One such is shown in Figure 23.2 (*a*).

Stopper. The stopper is a refractory-covered steel rod held vertically from the top end by the crossarm of the stopper lever assembly.

Anything from 6–12 pieces of refractory sleeves of circular cross section are used to cover the steel rod. The bottom sleeve forms the plug and is

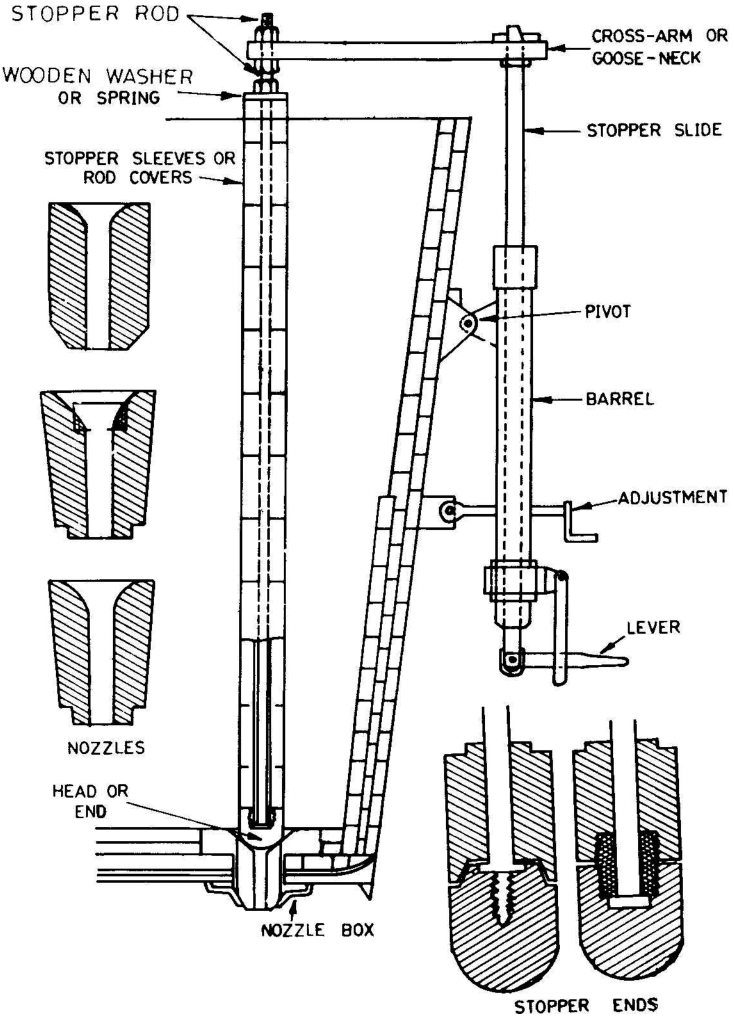


FIGURE 23.2. (a) Stopper assembly for teeming ladle. The detail of the nozzle and the stopper end are shown.

firmly secured in the flanged bottom end of the steel rod. The plug may be made of ordinary fireclay, high-alumina clay, graphite, chrome-magnesite, graphitized clay et cetera. Graphitized stopper end plugs are quite popular. At the slag line the sleeve should stand the action of oxidizing slag at the high temperatures.

A piece of sleeve may be approximately 30 cm in length and 15–20 cm in diameter. The sleeve must protect the steel rod and thereby retain full control on the stopper end plug, which controls the teeming rate. The sleeves are, therefore, shaped to interlock one into the next, and good-quality cement is used to bind them.

The stopper is held firmly against the nozzle seat during tapping by a locking device in the lever assembly. A series of washers, springs, loose dry sand, etc. is used to allow expansion of the rod at the top end without allowing it to lift up from the nozzle seat.

The stopper rod must be lined afresh before using it, and a regular supply of these is, therefore, essential. The sleeved rod must be dried thoroughly before use.

Stopper Lever Assembly. It includes all the mechanical arrangement meant for raising and lowering the stopper to open or close the opening through which metal flows out from the ladle. Although the arrangement varies from one design to another, every arrangement essentially has a cross arm to hold the stopper vertically against the nozzle seat, and a barrel and slide lever to allow smooth maneuvering of the stopper rod. In spite of the developments in automated and mechanized control of the stopper assembly, the manually operated design, as shown in Figure. 23.2, is still very commonly used.

Nozzle. It is essential to teem the ingot mold with a pipe-like stream of metal flowing at a constant rate to minimize possible ingot defects. This is achieved by fitting a nozzle in the seating block from either inside or outside the ladle. Those fitted from outside need to be locked in position. The use of a seat block facilitates the fixing up of the nozzle. The choice of the material for making nozzles depends upon the type of steel to be cast. Ordinary fireclay nozzles are used to teem fully deoxidized or killed steels. High-alumina clay nozzles are used for teeming un-deoxidized or rimming steels.

The metal flow is principally controlled by the nozzle shape and size rather than the stopper position. Several nozzle shapes are in use and are also shown in Figure 23.2. The nozzle should last without excessive erosion and appreciable change of dimension, until all the metal in the ladle is teemed.

23.4.2. Preparation of Teeming Ladle

After emptying the ladle it must be freed of any solidified metal, (*i.e.*, skull and slag). Any worn-out part of the wall must be made good by ramming clay or ganister. Careful patching after each teeming improves the lining life considerably. Bottom patching is not desirable as it tends to affect the cleanliness of the steel adversely. The life of the ladle lining very much depends upon the amount of slag that flows in with metal during tapping for it has the most corrosive effect on fireclay lining. Bad joints allow the skull to grow in these cracks, and the lining is damaged while removing the skull.

A newly lined ladle or a repaired ladle must be thoroughly dried before use. An ordinary fire bucket is placed in the ladle for this purpose. In modern plants special drying equipment using gaseous or liquid fuels are used.

23.4.3. Slide Gate Valve

An alternative to the stopper-rod assembly has been recently developed for flow of liquid steel during teeming. It is very simple to make, fit on the ladle, and operate for steel flow control. It is more reliable and sturdy, and being not exposed to liquid steel as the stopper rod is, it gives much better life and reliability. As a result, it is quite extensively adopted now in place of a conventional stopper-rod assembly. It is shown in Figure 23.2 (*b*). This valve assembly is fitted on the teeming ladle from outside below the nozzle in the ladle lining. The valve is a simple plate construction but supplied in ready-to-use preassembled cassette form to be introduced in the ladle shell socket from outside as easily as a cassette is introduced in a cassette player. This arrangement allows easy repairs and thereby overall economy in refractory consumption. The hinged opening system allows easy inspection and quick changing, if necessary. Symmetrical plates of two working faces ensure that by 180° rotation a new working face can be obtained to ensure increased life.

The success of these valves on teeming ladles led to its development and adoption even on steel making furnaces for tapping liquid steel more free of slag.

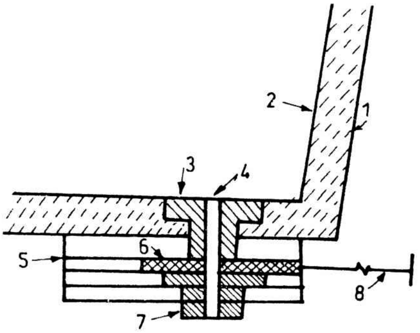


FIGURE 23.2(*b*). The slide gate assembly for a teeming ladle:
1. Teeming ladle shell
2. Ladle lining
3. Nozzle
4. Nozzle hole
5. Socket for slide gate assembly
6. Upper plate of slide valve
7. Lower plate
8. Handle

23.5. Ingot Mold

Except in a steel foundry, molten steel is poured into molds to produce ingots for rolling, forging, etc. The molds are themselves massive castings of more uniform shapes with a cross section such as square, rectangular, round, polygonal, etc., which are used according to the subsequent mechanical working operations that are to be carried out on the ingots. Ingot with a square cross section is used for rolling into billets, rails, structural sections, girders, merchant products, etc.. Rectangular shape, also known as a slab, is suitable for rolling into flat products such as sheets, strips, and plates. Round ingots are not used much but can be used for tube making. Polygonal ingots are used for tires, wheels, forgings, etc.

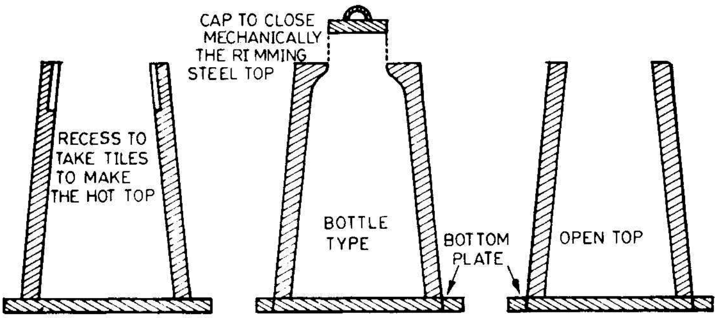
The weight of an ingot cast in a mold varies considerably, depending upon the rolling and forging mill designs. A 3- to 5-t ingot of mild steel was once commonly used for rolling, but its place has now been taken over by 5- to 20-t ingots. Some of the alloy steel ingots, for rolling as well as forging, may be as small as few hundred kilograms only. The size of the ingot for forging, in general, varies considerably from a few metric tons to few hundred metric tons as per the job requirements. Casting of an approximately 300- to 400-t forging ingot is not entirely an exception anymore. It does involve highly developed technology.

To help strip the mold off the ingot, the mold walls are tapered from one to the other end, which divides the molds into two basic types, viz.

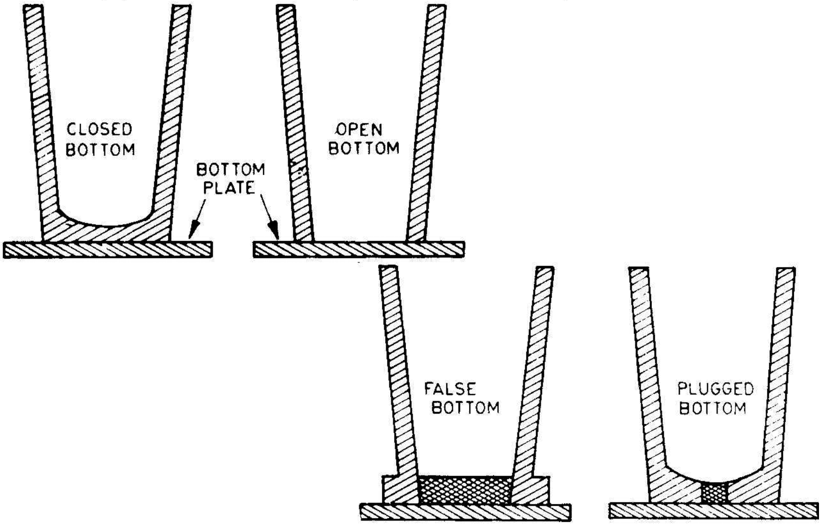
(*i*) Narrow-end-up (NEU) or big-end-down, as shown in Figure 23.3 (*a*).

(*ii*) Wide-end-up (WEU) or narrow-end-down, as shown in Figure 23.3 (*b*).

Narrow-end-up molds can be lifted from the lugs to strip the ingot off the mold easily and efficiently. This is used very commonly to produce



(*a*) NARROW END UP (WIDE END DOWN) MOULDS.



(*b*) WIDE END UP MOULDS (NARROW END DOWN)

FIGURE 23.3. Ingot mold types.

rimming and semi-killed steel ingots for rolling. Some form of the insulated head to feed the pipe may be used, or alternatively the head is left open to make an open-top mold for semi-killed steel ingots. For making a capped rimming steel ingot, a bottle-top mold is used.

Wide-end-up molds cannot be stripped off without the aid of additional mechanical arrangement. These are used to produce forging ingots of killed plain carbon or alloy steels. Open-bottom, closed-bottom, and plugged-bottom types of wide-end-up molds are also in use.

Forging ingot mold differs from rolling ingot molds. The forging ingot has two parts, the rolling ingot, only one. The main portion that is actually forged is called *chill.* The portion to be held in manipulator grips (for small forgings), or in a porter bar (for large forgings), while the chill is being forged, is known as *head.* Usually a wide-end-up mold is used with a feeder box of narrow-end-up shape to produce the head. Alternatively, a false head may be dropped in the mold, with or without a box.

Wide-end-up molds may have a solid bottom as against the open bottom of narrow-end-up molds. In some designs, an opening of 6–10 cm in diameter is left in the solid bottom, which is plugged by a metal or refractory plug just before the mold is poured. Although the erosion due to impingement of the metal stream is expected to be confined to the plug, the mold edge around the plug does get damaged and may cause difficulties during stripping. The hole is useful for cleaning and draining of the mold. A ram may be inserted through this hole to push up sticky ingots during stripping. There are strong advocates for both of these types of molds.

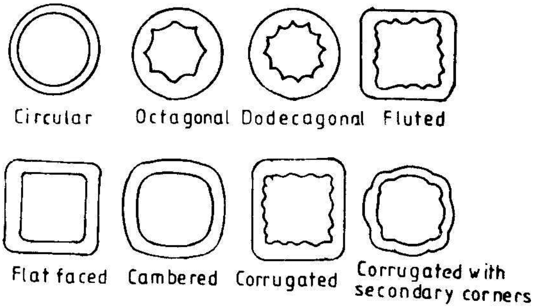


FIGURE 23.4. Cross-sectional shapes of molds.

The inner walls of the mold may be plane, cambered, corrugated, or fluted, as shown in Figure 23.4. The plane-walled mold has a minimum of surface per unit weight of ingot it produces, and hence the cooling rate of steel is least. The specific surface area of the mold walls increases in the order described above, from plane to fluted, which has the effect of increasing the initial ingot skin thickness by promoting faster cooling. Corrugated molds are more commonly used than fluted molds. The height of corrugation varies from 10–20 mm and the wavelength of corrugated shape varies between 75–150 mm.

Some separate molds cannot be used for every weight of ingot required for forging; these can be produced from a bigger mold by using either a false bottom or by inserting a head box inside the mold at an appropriate depth. The cross section of the ingot remains the same, only the height is reduced. The shop should carry the necessary designs to cater such demands.

The mold in general should be massive enough to possess sufficient heat capacity, so that the heat evolved during solidification of ingot does not overheat the mold. The massiveness is measured in terms of the mold wall thickness. Small molds may have walls as thin as 8 cm, which increases up to 30 cm for big molds. Since mold consumption is an important cost item in the production of ingots, the mold should have, commensurate with adequate mold life, as thin a wall as is possible. In general, the ratio of mold weight to ingot weight that it makes is usually between 1.1 to 1.5 for open-top molds, and the corresponding values are 1.2 to 2.0 for hot-topped forging ingots. Larger weight of a mold may be required for quality requirements of the ingots and better mold life. The height of the mold in general varies between 1.5 to 2.5 m. Forging ingot molds may be much taller than the same size rolling ingot molds.

Rimming steels rise during solidification by about 10–25 cm and hence this much extra height of the mold is left after teeming is over. The corresponding figure is just a few cm for semi-killed steels. In the case of wide-end-up hot-topped molds, the height of the mold is more than that of the ingot because of the solid bottom.

The design of the mold has a profound influence on the solidification characteristics of the ingot, and thereby it affects not only the quality of the ingot but also its general mill processing and the product yield. Since the ingot shrinks on solidification, an extra allowance of 2% in the mold cross section is allowed in the design. Similarly, a 1%–2% taper is provided in the mold walls in almost all types of molds. The corner radius of the mold may be very critical since there the cooling rate is much faster than that at the flat surfaces.

23.5.1. Mold Material

Cast iron has good thermal shock resistance and is hard enough to resist grazing. Earlier molds were made of cupola metal, but now these are increasingly being made of blast furnace hot metal after careful control of its composition and temperature. These are cheaper and lower in sulfur content than those of the cupola metal.

Cast iron has a different coefficient of expansion from that of the steels and hence the mold shrinks away from the ingot face on solidification, thereby separating the two readily. The sticking of ingot to the mold is further minimized by applying mold dressings.

23.5.2. Hot-Top

Fully deoxidized or killed steel such as that used for high-quality forgings shrink deeply on solidification and may lead to the formation of a pipe (see Section 26.1). The use of hot-top acts as a reservoir to feed metal to the main part of the ingot and avoid the formation of such a pipe, which otherwise leads to excessive loss of ingot yield during working. The shrinkage cavity is then mainly confined to the hot-top region. Hot-top is also called a feeder head.

Ho- tops are made in two ways, as shown in Figure 23.5. In one, fireclay lining is given to the mold top from inside. It is used only once and needs to be made anew for every casting since it is broken during stripping of the ingot. The other uses a cast iron box that is lined from inside with fire bricks and is generally placed on top of the wide-end-up molds. The brick surface is prepared after each casting of clay and water. It is fixed to the mold top by clamps or by wooden wedges. The feeder box should be placed in vertical position and the longitudinal axes of the mold and the box must be the same. It may be nearly 30–60 cm in height.

The box is stripped first, as it has its narrow end upwards. The ingot may be withdrawn from the wide-end-up mold by holding the head (*i.e.*, the portion solidified in the feeder box).

Exothermic or insulating materials are often added in the hot-top, immediately after the teeming is complete, to retard solidification in the hot-top portion since it must solidify at the end.

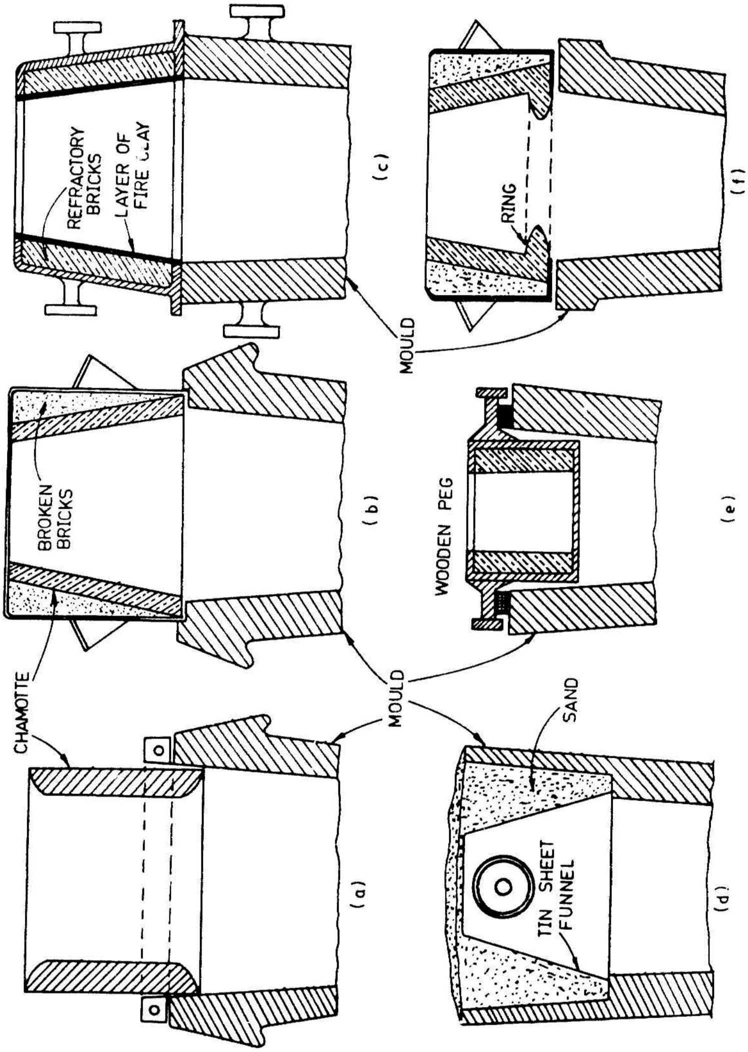


FIGURE 23.5. Hot-tops of various designs.

23.5.3. Bottom Plate

Rimming and semi-killed steels are cast in molds open at the bottom also. The top and the bottom of these molds must be machined smooth and level in order to sit squarely on a level plate called the bottom plate or the stool. They may have pins to locate molds correctly on the stool. The assembly of molds mounted on a stool is placed on a casting bogie. The mold must stand vertically on a horizontal bottom plate. Such an arrangement prevents leakage of metal at the bottom.

The bottom plates are made of cast iron. Although the design is simple, its thickness, and thereby the weight, should be so designed as to avoid excess weight for maximum of life. As with the molds, the bottom plate also fails more by cracking rather than by erosion. A careful record of its past performance helps to design it properly for better life in any plant.

Most hot-topped molds have a closed bottom, but these are also placed on flat stools to keep them vertical during teeming.

23.5.4. Mold Life

Mold consumption is an important item in the overall cost of ingot production. The total requirement of molds generally is approximately 2% of the weight of ingot production. It means that the mold can be used for approximately 50 pours, assuming that the weight of the mold itself is nearly equal to that of the ingot it produces. In some very efficient plants the mold is used for as many as 100 pours with usual intermediate repairs.

Molds fail in service primarily due to *fire-cracking* of the mold chamber walls. Usually such cracks are initiated near the bottom and spread upwards till the entire surface may show a network of these cracks. Some of these cracks may be deep enough to obstruct smooth stripping of the mold. It then becomes necessary to discard the mold. Fire-cracking also affects the surface quality of ingots, and even though mold stripping may not be a problem, the mold has to be scrapped to maintain good surface quality of the product. This is truer in the case of hot-topped molds. Forging ingot molds are discarded, even when mold walls are in proper shape, simply because of a spoiled plug hole at the bottom.

Defective practices such as late stripping, inadequate cooling between two consecutive pours, and fast teeming contribute to fire-cracking of molds. Accidental bumping and dropping of the mold may aggravate the problem of cracking, leading to early discarding of the mold.

The mold life does vary with its own composition and is decreased with increasing sulfur and phosphorus contents. Alloy steels may be teemed at higher temperatures than that for mild steels. It affects the mold life adversely.

23.5.5. Mold Preparation

After the solidification of metal, the molds are stripped off the ingot. The ingots are sent to the soaking pit and the molds are reconditioned for the next hour. The reconditioning includes cooling of mold by water sprayers, their assembly on the stools, and mold washing, generally with tar or pitch. In the case of bottom-poured molds, a new set of hollow refractory guides is used during reassembly of the molds. The hot-tops are also built up anew or reconditioned for the next pour. The general arrangement of the shop is shown in Figure 23.6.

If molds are to be cooled, they are taken on the bogie car to the water spray area. After spray cooling, a tar or pitch coating is applied with the help

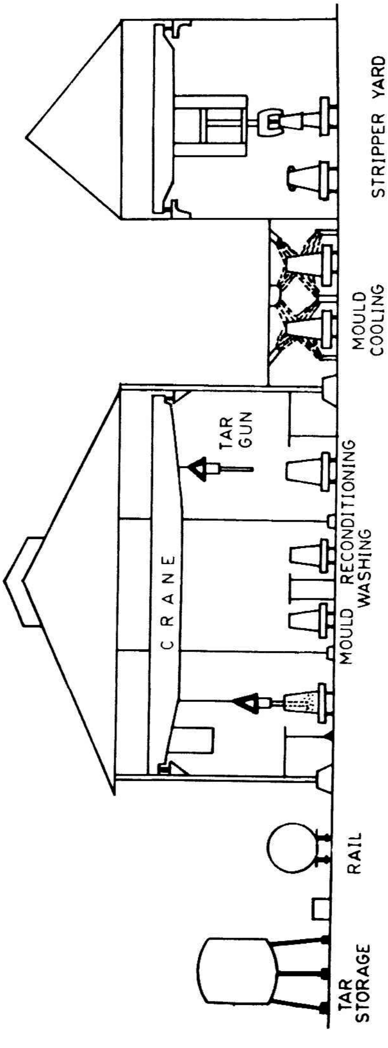


FIGURE 23.6. Mold stripper yard, mold reconditioning and assembly shop.

of a portable sprayer, which is lowered in the mold from above. Tar or pitch is best coated when the mold surface temperature is above 95°C and below 230°C. A lower temperature in this range is suitable for tar coating and a higher temperature is preferred for pitch coating. As the molds get older these temperatures are increased to 150°C–230°C so that excess volatile material does not get entrapped in the surface cracks.

The carbonaceous coating prevents splashed steel from adhering to the walls, and due to its gasification, pushes the metal away from the walls as it rises in the mold. The thickness of coating should be adequate to generate necessary gas pressure to push steel away from the mold wall.

23.6. Teeming Methods

Teeming means pouring of liquid steel in an ingot mold. The method of teeming affects the ingot quality. Three different methods are used for teeming to produce ingots.

23.6.1. Direct Pouring

The metal is teemed from the ladle directly in the mold, as shown in Figure 4.15. The rate of pouring can be controlled by the use of different sizes and designs of nozzles. The rate of teeming increases as the nozzle diameter increases due to erosion. This increase in the rate is to some extent neutralized by the decreasing ferro-static head in the ladle with the progress of teeming. Magnesite and graphite nozzles are better than fireclay nozzle in this respect. The size of the nozzle employed varies with the type of steel to be teemed. Soft steels are more viscous, and hence a slightly larger diameter (≅ 35 mm) of the nozzle is preferred; for medium and high carbon steels nozzle of 20-mm diameter is used for ingots of nearly 5-t sizes.

Since the metal stream directly hits the bottom plate of the mold, the wear of the bottom plate is quite severe in direct teeming.

This is used for teeming rolling ingots.

23.6.2. Tundish Teeming

The ingot should be teemed by a pipe-like metal stream at a uniform rate to minimize ingot defects. A tundish is, therefore, inserted between the ladle and the ingot mold to ensure a uniform metal stream while teeming from top. The tundish has its own nozzle to regulate the flow. A stopper may be provided in the tundish to further regulate the flow. The tundish is filled to a certain depth that is maintained the same throughout the teeming period to eliminate the flow variation due to the *varying* ferro-static head in the ladle. The tundish nozzle size is slightly bigger than the ladle nozzle to compensate for the lower ferro-static head in tundish than that in the ladle.

Tundishes with one or more, up to eight, nozzles are employed to distribute the metal evenly in that many molds at a time. This reduces the total teeming time of a ladle and, the available superheat in the metal can be fully utilized. This is not possible in direct teeming.

Tundish is used for teeming forging ingots and special alloy steel ingots.

23.6.3. Bottom Teeming

This is also known as uphill, or indirect, teeming. Steel is teemed into a vertical runner that is connected at the bottom to a horizontal through-runner, the end of which is an elbow shape that opens up in the bottom of the mold, as shown in Figure 23.7. The top of the vertical runner is shaped like a trumpet or bell to make teeming easy. The height of the vertical runner is more than that of the mold to ensure complete filling of the mold. In general, one vertical runner is meant to feed at least two and as many as twelve (still more if pencil ingots are cast) molds at a time, with four as a more popular figure. All the molds are set on the same bottom plate having the required number of through-runner channels.

The vertical runner is a cast iron pipe. The horizontal runner is also a cast iron pipe but in two halves cottered together. In some designs a square shape from outside is used. The through-runners are set in the bottom plate in slots. The runner boxes are machined from inside to a closed tolerance to fit the hollow refractory guides tightly in them and safeguard these boxes. The refractory guide at the bottom of the vertical runner has a flange to tightly secure it in the box. The refractory block that divides the metal into multiple streams is located at the bottom of the vertical runner and is called the center-pot. The vertical and the horizontal refractory guides are tightly fitted in the center-pot. The end runner is designed to smoothly change the direction of metal flow from horizontal to vertical. It is a normal practice to decrease the velocity of metal as it enters the mold. This is achieved by making the diameter of the end runner opening more than that of the through-runner.

In general, fireclay is used to make the hollow refractory guides for its low cost. Either cement or dry slag dust is used to close the joints to prevent possible breakouts.

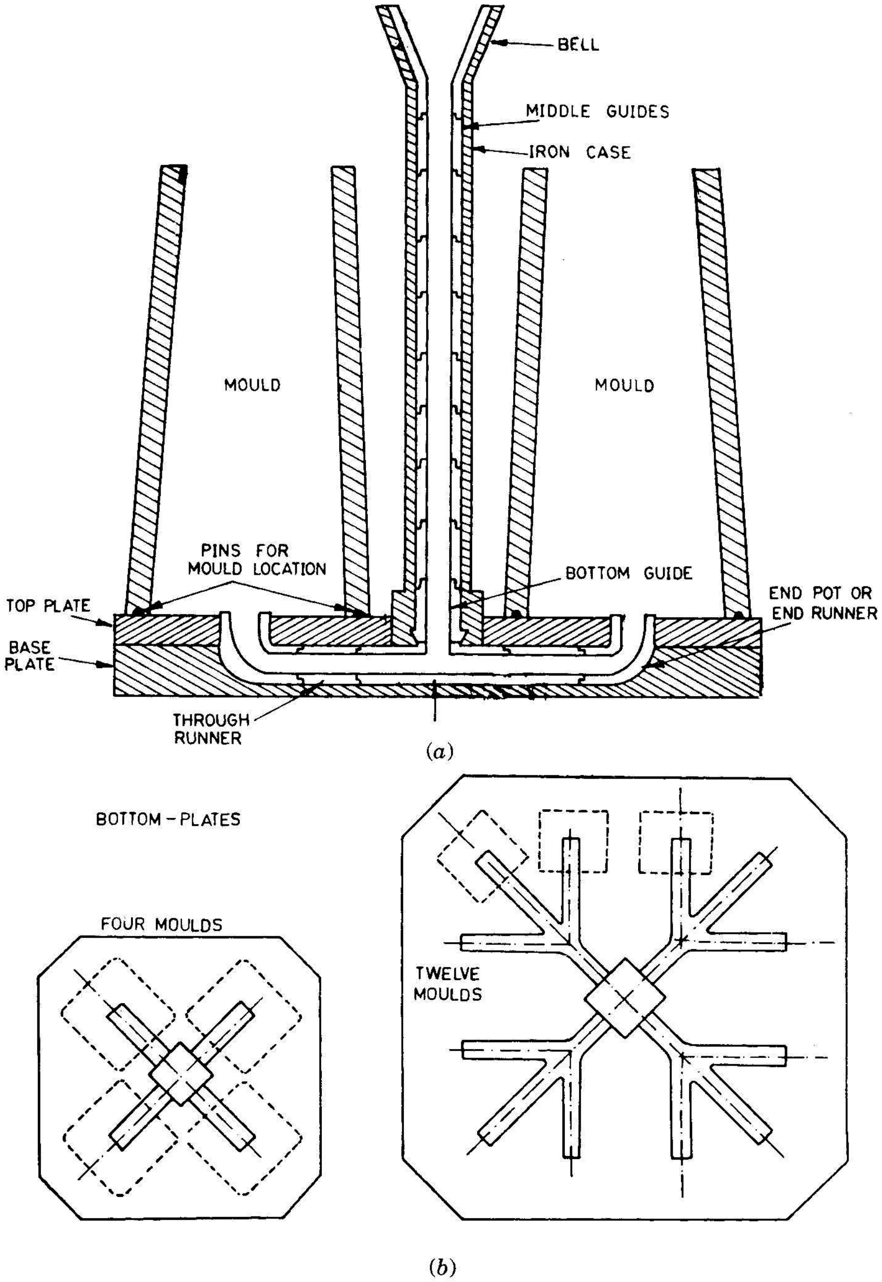


FIGURE 23.7. Constructional details of bottom teeming of heights. Vertical section is shown in (a), and (b) shows the details of bottom plates of two different designs.

For small ingots, the weight of the mold ensures adequate weight on the horizontal runner to counterbalance ferro-static pressure and to keep them in place. For larger ingots, additional plates are used. The base plate or these additional top plates have pins to assist in the setting of the mold in the correct vertical position. Compressed air is used to blow off any loose particles in the runners before teeming.

The quality of the bottom teemed ingot is much superior, and the bottom plate wear is much less as compared to top teemed ingots. Bottom teeming, however, uses more refractory ware and involves elaborate preparation of the mold assembly. Steel solidified in the runners has to be scrapped. The extent of preparation of the mold assembly and the amount of metal scrapped as runners are minimized is to some extent accomplished by using more number of molds in the assembly.

Use of bottom teeming is economically justifiable only if the superior quality of the ingot is necessarily required.

23.7. Pitside Practice at Rourkela Steel Plant (India)

It is being described here as an illustration. The steel making shop has three 45- to 50- t and two 60- to 66-t LD converters, and five 80-t BOH furnaces producing on average 80–90 heats daily to be handled by the pitside. This involves handling of 80–90 mold trains daily, one for each heat. Steel is teemed into four sizes of narrow-end-up molds of 8.3 t, 10 t, 12.5 t, and 16 t. Fireclay nozzles of 40 mm with a pouring rate of about 45–50 kg/second are used for the 8.3-t and 10-t molds, whereas magnesite nozzles of 60 mm with a pouring rate of 75–80 kg/second are preferred for the two bigger sizes.

The 40-mm size fireclay nozzle is used for making rimming ingots. The pouring rate is 9–11 mm/second. The rise of ingot (20–30 mm) is controlled by the addition of 0.5 kg aluminum (92%–95% purity) shorts per metric ton of steel. Rimming is allowed between 12–15 mm. The ingot is capped chemically using 0.36 kg of aluminum per metric ton.

Killed steels are teemed in recessed hot-top molds with a feeder volume of approximately 12% of the ingot volume. A commercial anti-piping compound is spread on the top of the ingot 15 seconds after teeming is over. Killed steels are also being teemed to usual molds with hot inserts made from semi-exothermic compounds with insulating properties. The latter has proved to be better than the former.

The smaller-size molds are given a 12-hour cycle, and the bigger-size molds are given a 18- to 20-hour cycle to cool them to 100°C on mold-cooling beds in the mold preparation yard.

23.8. Production of Large Ingots

Many of times it is necessary to produce large ingots weighing up to a few hundred metric tons. The molds employed for this usually consist of three parts viz., the bottom, the central chill, and the top feeder head as shown in Figure 23.8. These are almost invariably wide-end-up molds and are top teemed via a tundish or pony ladle. If the ingot weight is up to 50–100 t it can be poured from a single heat, but quite often several heats are required for larger ingots. This can be achieved by proper coordination of several furnaces in the shop. The mold is prepared, set up, and the first ladle is brought over the mold. Teeming is commenced only when the next furnace is being tapped. In this way, teeming continues without a break. The tundish or pony ladle must always remain full to a desired level until the teeming is complete.

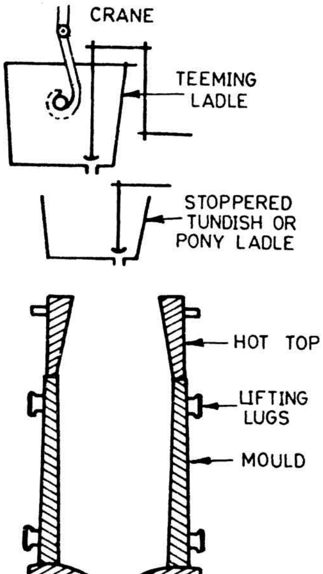


FIGURE 23.8. Casting of a large ingot weighing up to a few hundred metric tons.

The sizes of nozzles in the ladle and tundish need to be carefully calculated to ensure correct rate of flow.

The feeder head should be of adequate size not only to prevent pipe formation in the main part of the ingot but also to form adequate manipulative head required during subsequent forging. The feeder head may be as much as 20%–30% of the total ingot weight. The usual precautions should be taken more seriously to minimize defects in ingots, for a rejection of an ingot of this type means a tremendous loss.

A 400-t ingot was cast under vacuum at the Japan Steel Works Ltd[41] for making a rotor shaft of a nuclear power plant. Four furnaces of 100-t and 120-t individual capacities were used for pouring. The ingot was 4.0 meters in height and 3.6 meters in diameter.

CHAPTER 24

SOLIDIFICATION OF STEEL IN INGOT MOLDS

In This Chapter

- Types of Steel
- Mechanism of Solidification
- Ingot Types
- Control of Ingot Structure

24.1. Types of Steel

Molten steel contains dissolved gases, in particular oxygen. During cooling of steel the solubility of dissolved gases decreases and the excess comes out of solution. Of greater significance is the chemical equilibrium between carbon and oxygen dissolved in steel (see Section 4.2.1). The equilibrium shifts during cooling in such a way that the two react to form carbon monoxide gas which is expelled from steel. The amount of oxygen in solution and the amount that is expelled as CO is decided by its carbon content, the type and amount of deoxidizer added to steel prior to solidification. Steel that is fully deoxidized by strong deoxidizers is called killed steel since no activity is observed, in this case by way of gas evolution during solidification. It remains quiet in the mold as if it is dead, and hence the term killed. Killed steel solidifies progressively from the sides and bottom of the mold until finally a shrinkage cavity, known as pipe, is formed in the top central portion of the ingot. As against this, if the steel contains oxygen so that CO evolution is possible during solidification, depending upon the amount of gas evolved, a series of ingot structures are produced. If the evolution of gas is appreciable, in other words deoxidation is not fully carried out, it gives the appearance of boiling to liquid steel in the mold. This boiling action is termed as *rimming*, and the steel is known as rimming steel. In between violently rimming and killed steels lie the semi-killed steels, which are only partially deoxidized such that some gas evolution takes place during later stages of solidification. The capped steel is only a special variety of rimming steels in which the rimming action is less violent.

24.2. Mechanism of Solidification

Killed steel solidifies in three zones in an ingot form. The metal next to mold walls and bottom is chilled by the cold mold surfaces. This is a thin layer and is known as a chill, shell, or skin of an ingot and has fine equiaxed grains. The rate of solidification is very high in forming the skin. However, the rate of solidification soon slows down. The mold expands on heating and the skin contracts on solidification, resulting in separation of the two and formation of an air gap in between. It reduces the rate of heat flow and thereby slows down the cooling of an ingot. The solidification front moves inwards perpendicular to the mold faces, resulting in columnar grains next to the chill. Their lateral growth is restricted due to the adjacent crystals. The columnar crystals rarely extend to the center of the mold. The central portion solidifies as equiaxed grains of bigger sizes than those in the chill due to slow cooling. Although these zones exist, their boundaries are not as distinct; as shown in Figure 24.1, one zone blends into the next gradually. The extent of each zone varies with the composition and the temperature of liquid steel, mold design, and its temperature at the time of teeming.

The three-zone solidification mechanism is altered due to the evolution of gas. If steel has not been deoxidized fully, the reaction

$$[C] + [0] \qquad \{CO\}$$

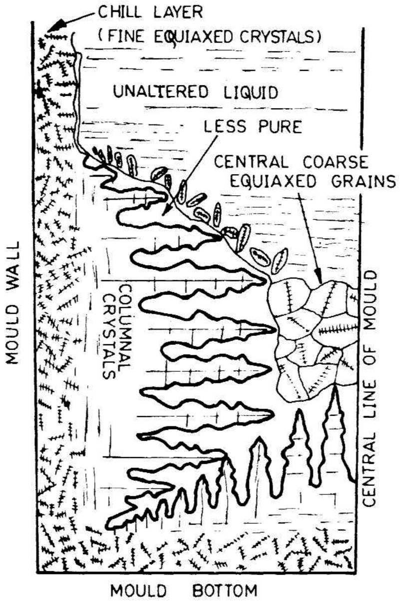


FIGURE 24.1. Structure of a killed ingot showing the three zones of solidification.

mainly forms the basis of gas evolution during cooling and solidification (see Figure 24.1). Carbon and oxygen react to form carbon monoxide only if the product of concentrations of carbon and oxygen in steel exceeds the equilibrium value, which is a function of temperature and pressure. As the temperature and/or pressure decreases, the equilibrium shifts to the right.

If oxygen content of steel is very low, then the necessary super-saturation of carbon and oxygen for CO formation is not achieved until towards the end of solidification. The ferro-static head also prevents it from attaining the same at the bottom portion of the ingot. It means that the formation of chill layer and the columnar crystals are not much disturbed. Only the central zone of equiaxed crystals is disturbed, by way of formation of blow holes in the top middle portion of the ingot. This is the way a semi-killed steel ingot will solidify.

As the oxygen content of steel increases, CO evolution takes place earlier and the amount of evolved gas also increases. When oxygen content is appreciable, marked gas evolution is seen immediately after pouring steel in the mold. The gas is evolved at the solid-liquid interface, and the rising bubbles cause metal to circulate, which brings hot metal to the surface, and freezing of the top is delayed. It prevents formation of columnar grains, and the interior of an ingot is at a more uniform temperature. This finally gives rise to the capped or rimming ingots in which gas that is evolved is mechanically entrapped as blow holes. In a rimming ingot, pipe is very small and irregular. The elongated or honeycomb blow holes are called primary blow holes, and the circular blow holes are called secondary blow holes.

24.3. Ingot Types

A series of ingot types are produced, depending upon the extent of deoxidation carried out prior to solidification of steel in ingot molds. These are shown in Figure 24.2 in the form of eight different types. The amount of gas evolved during solidification progressively increases from No. 1 to 8, (*i.e.*, steels are deoxidized to lesser degree in the same order). The details of the ingot types thus produced are discussed below.

No. 1. This is a typical killed steel ingot with intermittently bridged pipe. The killed steels are almost invariably cast in wide-end-up molds with a hot-top so that the pipe is confined to the feeder head as far as possible.

No. 2. This is a partially deoxidized steel. The gas is not evolved in the early stages of solidification. The ferro-static head prevents it from evolving at the bottom of the ingot. The gas is evolved, in a small amount, and that too towards the end of solidification, (*i.e.*, at the middle portion of the ingot). It is enough in volume to compensate for shrinkage and thereby reduce or nearly eliminate the pipe. The pressure developed by the gases is sufficient to cause the top to bulge out. This is a typical semi-killed steel ingot.

No. 3. The gas is still evolved only towards the end of solidification. Since the amount of gas evolved is more than that in No. 2, the pressure developed is high enough to burst the solidified top surface. This is called bleeding. The blow holes are near the walls and are confined to the upper portion of the ingot.

No. 4. The gas evolution is appreciable and commences early to cause the liquid steel to rise in the mold. This rimming action does not allow the top to solidify immediately after pouring. The rising level of the metal is stopped by putting a metal cap at the top. The remainder of the evolved gas is thereby entrapped to form a series of honeycomb-shaped blow holes all along the walls and bottom.

No. 5. The gas evolution is so large and is evolved so early that a great deal of it escapes out from the ingot. The ingot slowly rises in the mold and

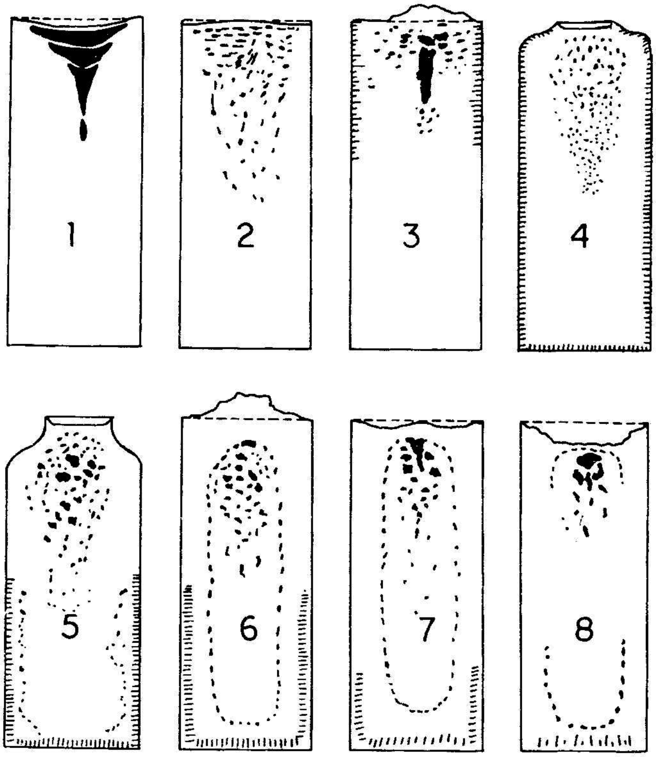


FIGURE 24.2. Typical ingot structures as resulting from varying amount of gas evolution during solidification. The amount of gas evolved increases progressively from No. 1 to No. 8. Typical commercial ingots are No. 1 killed No. 2 semi-killed, No. 5 capped, and No. 7 rimming steels.

is capped to produce a typical capped ingot. The ingot is capped after the rimming action slows down. The net result is that a thick skin is formed first and then the honeycomb blow holes, which are less than those in No. 4.

No. 6. A great deal of evolved gas escapes out. The ingot, therefore, does not rise in the mold as much as in No. 5. The honeycomb blow holes are formed after thick skin formation and are confined to the lower portion of the ingot.

No. 7. The brisk gas evolution finally leads to the formation of a very thick skin and honeycomb blow holes in the bottom quarter of the ingot. The ingot does not rise in the mold to any appreciable extent. The volume of blow holes just compensates for the shrinkage. This is a typical rimming ingot.

No. 8. This gives a very violent rimming action such that no blow hole formation is possible. The ingot in fact sinks to some extent during solidification.

The main four types of ingots that are commercially produced are included typically in the above illustration, killed in No.1, semi-killed in No. 2, capped in No. 5, and rimming in No. 7.

24.4. Control of Ingot Structure

The final structure of an ingot is almost entirely determined by the total degree of deoxidation carried out prior to solidification of steel in a mold. The residual oxygen in steel at the end of refining is determined by the steel making practice and the type of steel produced. For a given type of steel, the steel making as well as deoxidation practices have to be properly adjusted to finally obtain the desired ingot structure. These control aspects are described below.

24.4.1. Rimming Steels

Rimming steels require a great deal of gas evolution during solidification. The steel, therefore, must contain enough dissolved oxygen and that is possible only in low-carbon steel (see Figure 4.1). Hence, rimming steels are produced only if the carbon level is less than 0.15%.

The heat must be finished in a furnace with respect to iron oxide content of the slag in such a way that the bath contains the desired level of oxygen at a carbon level less than 0.15%. In general, no deoxidation is carried out inside the steel making furnace. Only a small amount of deoxidation, if needed, is carried out using Fe-Mn or Al as deoxidizers, in the ladle or preferably in the mold itself. The exact procedure that is followed depends on whether the carbon content of steel is high (0.12%–0.15%), low (0.06%–0.12%), or below 0.06%.

The correct thickness of ingot skin without any blow holes can be obtained by carefully controlling the oxygen content and temperature of liquid steel while teeming, so that the necessary gas evolution takes place in the mold. The rimming action of the first ingot that is teemed is observed to decide whether rimming is less correct or more. If teeming is correct it is continued but if more, aluminum is added to the mold to decrease it and if it is less, then gas-producing materials are added to increase gas formation.

In a properly cast rimming steel or even capped steel, primary blow holes formed next to the ingot skin of adequate thickness, and the secondary blow holes are formed still inside. The ingot has a minimum of pipe. The zone between primary and secondary blow holes is known as a *rim*, which is characteristic of a rimming ingot. Rimming ingot has a smooth surface that is relatively clean due to the brisk evolution at the beginning of solidification. Such a structure is most desirable for rolling of products wherein surface finish is most important (*e.g.*, in flat products). Rimming ingots are cast in narrow-end-up molds.

24.4.2. Capped Steels

This is another variety of rimming steels in which the gas evolution is much less brisk than in usual rimming steels; that is, it is less active. Steel is cast in bottle-shaped narrow-end-up molds in which the constricted top facilitates mechanical capping of the ingot. Early gas evolution is prohibited by adding some aluminum shots in the mold during teeming. The gas evolved in the later stages raises the metal level, which is stopped by capping. A cast iron cap is mechanically fastened down on to the mold top. The capping stops rimming and hence a thin skin is formed in the ingot, as compared to usual rimming steels. Usual rimming action in rimmed ingots is accompanied by a marked segregation that is, however, less pronounced in capped steel due to short rimming time (a minute or so before it is capped), which produces less concentration of impurities in the core. Capping can also be carried out chemically by adding strong deoxidizers at the top of the ingot.

Capped ingots are produced from the steels containing approximately 0.15% carbon. These are used for producing flats, wires, and bars.

24.4.3. Semi-Killed Steels

These are partially deoxidized steels such that only a small amount of gas is evolved during solidification. The carbon content has to be in the range of 0.15%–0.30%. The required deoxidation may be carried out, at least partially and, if permitted, in the furnace itself. Ferro-manganese, ferro-silicon, and aluminum may be used as deoxidizers. Gas is not evolved immediately after pouring of steel in a mold. In general, the top level freezes before gas evolution commences. This is ensured by adding aluminum in the mold as it is filled. The gas is evolved towards the end of ingot solidification. The shrinkage of steel on solidification relieves the pressure developed in the liquid core of the ingot due to gas evolution. The pipe is thus automatically compensated, by the gas evolution and its entrapment, if carbon and oxygen contents of liquid steel are carefully adjusted. Such an accurate adjustment is not readily obtainable in practice. Many times the steel contains more oxygen than desirable and hence, gas evolution takes place much earlier than required. This situation can be remedied by thoroughly deoxidizing the top of the ingot by aluminum that is added just before completion of pouring or immediately thereafter. It makes the top freeze quickly. The blow holes are, therefore, formed mainly in the middle part of the ingot with very little in the lower part. The blow holes are absent in the top portion because that portion is fully killed, and whatever CO rises from lower levels gets dissolved in this region because it shifts the equilibrium in reverse direction. Aluminum should be added carefully so that it is not carried down in the lower portion of the ingot.

Semi-killed steels find a very wide use in the manufacture of structural shapes, plates, and merchant products.

24.4.4. Killed Steel

No gas evolution takes place in killed steels during solidification. All steels containing more than 0.3% carbon are killed.

The heat is worked in such a way that by the time the carbon level drops close to the specification level, the refining should be over. In general, the heat is then blocked by adding Fe-Si, high silicon pig iron, or silicomanganese to the bath in the furnace. Blocking stops the carbon-oxygen reaction by lowering the oxygen content of the bath. Alloying additions that are susceptible to oxidation can be made to the bath at this stage if otherwise permissible. Ferro-alloys are added in the ladle while tapping to fully deoxidize the steel. The ladle additions, deoxidizers as well as alloying additions, should not be so large as to chill the liquid steel beyond a safe limit. The deoxidation products should be given adequate time to rise to the surface or else it will form nonmetallic inclusions in steel. Killed steels are almost always cast in wide-end-up molds with hot-tops. Although solidification of killed steel is accompanied by marked V and A type segregation, the killed steel structure is quite sound and dense, which is particularly suitable for forgings. Alloy steels are, in general, fully killed steels because soundness of the ingot is what is essentially required.

In general, the addition of deoxidizers to the mold is not recommended because that decreases the cleanliness of the steel. The deoxidizers added to the ladle can also be used to meet the relevant chemical composition in the specifications.

CHAPTER 25

INGOT DEFECTS AND REMEDIES

In This Chapter

- Pipe
- Blow Holes
- Columnar Structure or Ingotism
- Segregation
- Non-metallic Inclusions
- Internal Fissures and Hairline Cracking
- Surface Defects

It is always an endeavor of the operator to produce, both physically and chemically, a homogeneous ingot that would have a fine equiaxed crystal structure; would be free of chemical segregation, non-metallic inclusions, cavities, etc., and would have a smooth-surface finish. Unfortunately, this is not possible in practice because of the natural laws that govern pouring and solidification of steel as an ingot. Within itself an ingot develops pipe, blow holes, chemical segregation, dirtiness, columnar crystals, and internal fissures. Externally it exhibits surface cracks, seams, scabs, etc. The factors that lead to the formation of these defects and the possible remedies to minimize them are described here.

25.1. Pipe

The volumetric contraction resulting on solidification appears in the form of a cavity known as pipe, which amounts to approximately 2.5%–3.0% of the total apparent volume of the ingot. Rimming and semi-killed steels show slight tendency for piping, which can be eliminated by careful practice. Capped steel is practically free of pipe. The problem of pipe formation is, however, serious in the case of killed steels. The shape and location of pipe in killed steels depends upon the mold type, as is shown in Figure 25.1. In a wide-end-up mold the pipe is short and wide. In a narrow-end-up mold it is narrow and long and, a secondary pipe may also be formed.

Primary or open pipe gets oxidized and does not weld during rolling. As a result, that much of the ingot portion has to be discarded. It decreases the ingot yield. Being deep seated, the secondary pipe does not get oxidized and is welded up during rolling. The longer the primary pipe is, the lesser the ingot yield. Wide-end-up molds are, therefore, preferred to obtain better yield during rolling.

The detrimental effect of pipe formation on the yield is reduced by adopting a hot-top feeder head so that the pipe is confined to the feeder box of a wide-end-up mold, as shown in Figure 25.1. The volume of the feeder box is approximately 15% of the ingot volume. Use of insulating or

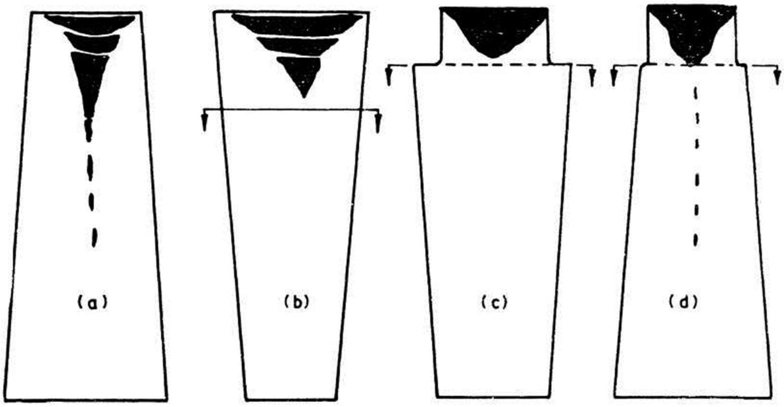


FIGURE 25.1. Killed steel ingots showing pipe formation. Secondary pipe is evident in narrow-end-up molds:
(*a*) Narrow-end-up mold showing long primary and secondary pipe.
(*b*) Wide-end-up mold showing short pipe.
(*c*) Wide-end-up mold with hot-top.
(*d*) Narrow-end-up with hot-top showing pipe confined to hot-top alone.

exothermic materials on the top of the ingot further ensures, by keeping the metal in the hot-top molten for a long period, that the pipe will be confined to the feeder head without extending the main part of the ingot.

Another method of reducing pipe in an ingot is to pour a little more metal after partial solidification of the ingot to compensate for part of the shrinkage. This is, however, not a very common practice.

25.2. Blow Holes

The entrapment of gas evolved during solidification of steel produces cavities known as blow holes in all except killed steels. The blow holes are of two types. The primary blow holes are elongated or like honeycomb and are located next to the ingot skin. The secondary blow holes are more spherical and are located further in. Formation of blow holes partially or fully eliminates the pipe and thereby increases the ingot yield during rolling, provided that these are located at the proper depth from the surface from all sides of the ingot. Deep-seated blow holes do not open up and thus do not get oxidized during soaking (due to scaling) and rolling of the ingot. Such blow holes are welded up during rolling and do not leave any mark of theirs in the product. Blow holes that are closer to the surface often get oxidized during soaking or get first punctured and then oxidized during rolling. Oxidized blow holes do not heal up during rolling and leave surface defects (seams) on the product.

Gas evolution during solidification should, therefore, be controlled to obtain these blow holes only after an ingot skin of adequate thickness is formed.

25.3. Columnar Structure or Ingotism

Steel is a crystalline solid. After the formation of an initial chill layer, further solidification results in the formation of dendrites that grow along their principal axis perpendicular to the mold walls. Their lateral growth is restricted due to the growth of adjoining dendrites, giving rise to elongated crystals. If the length of these is appreciable, it is known as columnar structure and in an exaggerated form, ingotism. Ingots possessing ingotism tend to crack during rolling unless in the first few passes the reduction in cross section (draft) is kept low.

In general, columnar structure does not extend to the center of the ingot. The middle portion of the ingot solidifies as equiaxed grains. The relative proportion of columnar and equiaxed grains is adjusted to keep the ingotism to a minimum. The adjustment is carried out with respect to the composition of steel, its temperature while pouring, mold temperature, and gas evolution during solidification.

25.4. Segregation

Segregation means departure from the average composition. If the concentration is greater than the average, it is called positive, and if it is less than the average, it is called negative segregation. It is often estimated as percentage departure from the average composition. Segregation is the result of differential solidification, a characteristic of all liquid solutions. Steel is a liquid solution of S, Si, C, P, Mn, etc., in iron and hence is prone to segregation during solidification. The initial chill layer of the ingot has practically the same composition as that of the steel poured in the mold (*i.e.,* there is no segregation in the chill layer because of the very rapid rate of solidification). The progressive solidification thereafter results in solidifications of a purer phase (rich in iron), while the remaining liquid gets richer in impurity contents.

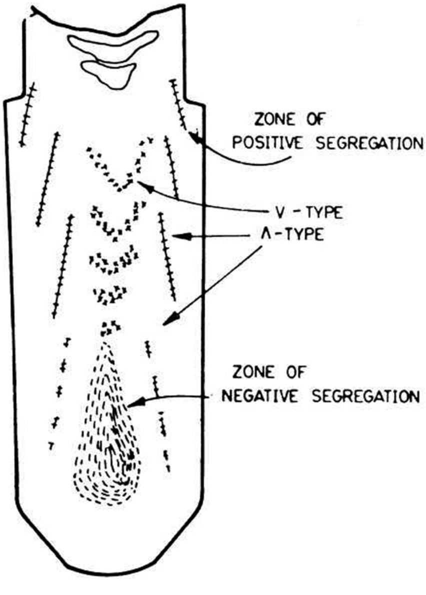


FIGURE 25.2. Killed steel ingot showing segregation.

A killed ingot cast in wide-end-up mold shows two types of segregation, as shown in Figure 25.2. The impurity segregation at the top follows the shape of the pipe and is known as V segregation. Side-by-side inversed V, or ∧-shaped, segregation is also observed at the top. It may be due to the sinking of purer crystals down and the rising up of the impure liquid in the upper part. The impurities get entrapped in the impure part at the end of solidification. This is the positive segregation. The negative segregation is confined to the lower central portion of the ingot. In the actual ingot these zones are not as sharp as are shown in Figure 25.2; these are quite diffused.

The tendency of an element to segregate depends upon the shape of the binary liquidus curve of the element with iron. The more shallow the liquidus (*i.e*,. the more is the difference in concentrations of the first and the last crystals to solidify), the more is the segregation. In this respect the segregation tendency increases in the order of Mn, Si, C, P, and S. In other words, segregation depends on the composition of steel.

Segregation increases with increasing time of solidification required for an ingot, so that large ingots tend to segregate more than small ingots.

Segregation in ingots of the same cross section is increased by the convention and turbulence caused by gas evolution during cooling and solidification of steel. Segregation, therefore, increases in the order of killed, semi-killed, capped, and rimming steels. In the semi-killed steels, the V segregate is less sharp and is pushed downwards. The V segregate is also at a lower position than in killed steel. The rimmed zone of a rimming ingot exhibits negative segregation. The boundary of these two zones is at times so sharp that they appear as steels of two different compositions.

Besides the above, the micro segregation of sulfur as revealed by sulfur print of the section is quite important. Sulfur segregation values are quite critical in certain areas of the structural shapes.

Segregation can be minimized by prolonged soaking of ingots before working.

25.5. Non-Metallic Inclusions

The term cleanliness is used to refer to the relative freedom from the entrapped non-metallic particles of solid ingot. In some steels this is the most important criteria in judging their quality. The fact that it is non-metallic and, therefore, incongruent with the metal lattice, has often been considered as *prima facie* evidence of its undesirability. Inclusions are of two types:

Indigenous–those arising in the course of steel making.

Exogenous–those arising from mechanical erosion of contacting refractory lining.

Indigenous inclusions are comprised of deoxidation products such as oxides, oxy-sulfides, or precipitates such as sulfides, carbides, nitrides, etc.. Almost all sulfur is precipitated as iron or manganese sulfides. The best way is to keep sulfur as low as possible to avoid formation of inclusions. Oxide inclusions can be kept to a minimum by a suitable deoxidation practice. Enough time is allowed for them to rise to the surface in the ladle. Alternatively, if economically permissible, vacuum treatment may be adopted to decrease the oxygen content of steel. Deoxidation carried out in the furnace also helps to minimize them.

Nitrides form only if the dissolved nitrogen level is high. The addition of Zr, Ti, and V form stable nitrides. Titanium carbide may act as an inclusion. Iron carbide is a part of the desirable microstructure of steel.

In general, proper care during refining and more particularly during deoxidation can minimize the indigenous inclusions.

The exogenous inclusions arise from the mechanical erosion of refractory lining with which metal comes in contact during its processing. In particular, the erosion of ladle and moreso of the refractories used in the assembly of the mold contribute most to the formation of exogenous inclusions. The best way to keep them down is to use a minimum of such refractory channels through which metal has to flow before solidification. Use of a strong refractory for such places is definitely beneficial.

Inclusions are not always undesirable. These are purposely introduced to gain a certain desirable effect. For example, sulfur is added, to obtain sulfide inclusions, to improve machinability. Exogenous inclusions are used for dispersion strengthening of steels. Stable nitrides are useful since the ill effects of dissolved excess nitrogen are thereby eliminated.

25.6. Internal Fissures and Hairline Cracking

The term *Clinked* ingot is used to denote internally cracked ingot. These cracks or fissures arise due to two causes.

1. Too rapid reheating of an ingot such that the outer layers expand more rapidly than the core, giving rise to internal rupture. Certain classes of alloy steels are prone to such cracking because of their coarse and weak crystal structure.
2. Too rapid cooling of an ingot after stripping the mold causes uneven contraction at the surface and in the core, finally resulting in internal fissures.

Internal fissures may extend to the surface and cause surface cracking or may open up during soaking and working. These can be eliminated by preventing too rapid cooling and reheating of an ingot.

Hairline cracks are formed all through the section and are revealed only after deep etching. These are oriented at random. Desorption of dissolved hydrogen is believed to be the principal cause of formation of hairline cracks in large sections. The solubility of hydrogen is decreased during solidification and further cooling of steel. Hydrogen is desorbed very slowly even after cooling the steel, for days or even weeks, depending upon the type of steel, the cross section, the residual stresses, and the hydrogen content of steel when it was molten. Alloy steels are more susceptible to hairline cracking. Since the hydrogen content of liquid steel is determined by the type of raw materials used and the steel making process adopted, these factors seem to influence hairline cracking appreciably.

Most operators feel that a hydrogen content less than 2.0–2.5 cc/100 g in liquid is *safe* to avoid hairline cracking. Hydrogen content can be reduced below this level by vacuum treatment of liquid steel (Chapter 28). In the absence of such a treatment, steel must not be allowed to cool to room temperature unless sufficient time is allowed for hydrogen to diffuse out. In practice, the steel is held at 600°C–654°C to reduce hydrogen content to below the safe limit. The holding time increases with increasing cross section and may even extend to several days or weeks.

Many times the ingot is stripped off the mold very early and is transferred to a soaking pit for prolonged soaking to allow hydrogen to diffuse out. This procedure also minimizes the tendency to form internal fissures.

It has also been shown that if the sulfur content of steel is less, then the safe hydrogen level also decreases. For the usual safe level of hydrogen to be permissible, the steel may even be re-sulfurized to some extent, if otherwise permissible.

25.7. Surface Defects

These are the defects that are apparent on the surface of an ingot after solidification or are visible in some form after mechanical working of an ingot.

25.7.1. Ingot Cracks

The chilling effect of a mold forms a thin solid layer on the side faces and bottom of the ingot immediately after pouring and is known as ingot skin. The thickness of this skin has quite an important bearing on the formation of ingot cracks. The contraction of the ingot on skin formation and the expansion of the mold on heating tend to separate the two, and this separation forms an air gap in between. The ferro-static pressure of the liquid core of the ingot now has to be withstood by the ingot skin alone. If the skin is not thick enough to stand the internal pressure, it ruptures, giving rise to cracks in the surface.

The thickness of the skin formed depends upon the time of contact of steel with the mold and is given by

$$\text{Thickness} = \text{constant } (k) \cdot \sqrt{\text{Time}}$$

where k is a constant varying between 0.9–1.2. The value of k depends upon the mold mass, mold design, temperature of liquid steel, temperature of the mold, pouring technique, and rate of pouring.

The factors that tend to decrease the skin thickness tend to help form cracks. A too high teeming temperature, rapid rate of teeming, and too high mold temperature decrease the skin thickness and thereby increase its tendency to crack. The formation of *hot shot*, that is, the area that cools less rapidly than adjacent areas owing to less close contact with the mold, tends to develop surface cracks. The friction between ingot and mold may also develop sufficient tension to cause surface cracking. If steel is poured more on one side than the other it may decrease skin thickness on the hotter side and thereby help form cracks. One or more of these factors may contribute to develop enough tension to finally rupture the skin. The types of ingot cracks are as follows:

1. **Longitudinal Cracks.** These are more or less parallel to the vertical axis of the ingot and are caused due to the development of lateral tension in the skin. The tendency to form this type of crack increases if the ratio of cross section to height increases. These are

formed more at the bottom position of the ingot. Alloy steels are more prone to form such cracks than are mild steels.

2. **Transverse Cracks.** These are nearly parallel to the base of the ingot and are formed due to longitudinal tension in the skin. The tendency to form this type of crack increases as the ratio of ingot height to cross section increases. This is the most common type of ingot crack.
3. **Restriction Cracks.** These may be longitudinal or transverse in direction and are located at the corners of the ingot. The longitudinal restriction cracks are due to the large corner radius of the ingot. The transverse restriction cracks are due to the friction between the mold and the ingot of a small corner radius.
4. **Subcutaneous Cracks.** These are internal fissures close to the surface and are caused due to thermal shocks. These open up during soaking and/or rolling.

Much of the surface cracks can be eliminated by designing the mold properly. Sufficiently thick mold walls ensure adequate chilling to produce skin strong enough to stand internal ferro-static pressure. The use of corrugated or fluted mold walls in place of plane walls increases the specific surface area so that chilling is improved. It also minimizes the transverse tensile stresses in the skin, developed due to its solidification. The tension in the crest parts is fairly well counter-balanced by the compressive forces in the trough part of the corrugated wave on solidification of the skin.

Smooth corners of the mold and neither a too large nor too small radius of curvature of the corner, minimize the restriction cracks.

The mold walls are tapered to 1%–2% to help strip the mold smoothly. The friction between the mold and the ingot can be minimized by dressing the mold before pouring.

The operational variables such as the teeming temperature of steel, mold temperature, thickness of mold dressing rate of teeming centering of the stream, and so on, should be adjusted carefully to minimize not only ingot crack but many other surface defects as well.

25.7.2. Other Surface Defects

A number of other surface defects are known to occur, mainly due to the faulty teeming practices and the use of faulty molds. These are described below.

Scab. A projection on the side surface of an ingot caused by freezing of steel in a cavity in the mold wall or in a mold with uneven walls. A scab produces seam, that is, grooves or striations on the surface during rolling.

Lappiness. Lap is a fold in the ingot skin caused by freezing of a slowly rising top surface of the metal in the mold before the pouring is over.

The surface gets oxidized along the periphery and separates the ingot portion that is poured from that which is yet to be poured. Several such nearly parallel laps may occur. An ingot tends to be lappy if slowly poured or if the teeming temperature is low.

Double pouring also forms an oxide layer at the point of arrest during teeming and thereby gives rise to lappiness.

Splash. Metal drops are thrown off due to the impact of the metal stream on the mold bottom. If these drops stick to the mold wall they form seams in the rolled products because, being oxidized on the surface, the drops do not weld up to the rest of the ingot during rolling.

Crazing. If a large number of cracks are present in the mold wall, steel may freeze in these cracks and give rise to a network of fins on the ingot face. This is also known as *crocodile skin.* Both formation of an individual fin and crazing, tend to form seams during mechanical working.

Double skin. The skin formed in the lower part of the ingot contracts, and steel flows in the gap between the mold and the ingot, giving rise to a double skin in the lower part of the ingot. This is due to the slow rate of pouring and/or severe chilling effects of the mold.

Spongy top. The viscous top tends to rise due to the late gas evolution and thereby makes the top spongy.

Flash. It is a plate of solidified steel formed due to steel entering crevices in the mold assembly (*e.g.*, at the joining of mold with the bottom plate or hot-top).

Boot leg. It is the sinking of an ingot top below the original level in the mold due to decreased evolution of gas. It calls for extra front crop during rolling.

Skin holes. These are formed due to entrapment of gas evolved from mold dressing in the skin. It often leads to seams in rolled products.

CHAPTER 26

GASES IN STEEL

In This Chapter

- Oxygen in Steel
- Nitrogen in Steel
- Hydrogen in Steel

Even at the end of refining when most of the impurities such as C, Si, Mn, S, P, etc., are eliminated below the required levels of gases such as oxygen, nitrogen, and hydrogen may still remain in solution as deleterious impurities worth decreasing them further down. The solubility of these gases depends upon the composition of steel, and it increases with temperature. On heating, the rise in solubility, in solid state, is slow, but at the melting point it increases very steeply and thereafter, in liquid state, it continues to increase but at a slow rate. The amount of these gases dissolved in steel at the end of refining depends upon the quality of raw materials used and the steel making process itself. For example, the solubility of oxygen in pure iron at 1600°C is 0.23% and that in solid iron at its melting point is 0.003% only. These values are affected by the presence of other elements in iron, both in solid as well as in liquid state. The excess dissolved oxygen and similarly all other gases precipitate out during solidification. These desorbed gases may cause skin or pin holes, blow holes, etc. In the case of oxygen it has a tendency to react with the carbon present in steel and form CO, which comes out as gas and plays the same role as any other gas on solidification. The cavities thus formed because of gas evolution are detrimental to the mechanical properties of steel except when the CO evolution is successfully controlled and trapped, to produce a semi-killed or rimming variety of steel ingots, which was very common practice until recently.

26.1. Oxygen in Steel

Oxygen is supplied for refining iron and hence, a certain fraction is inevitably left over as dissolved oxygen in liquid steel at the end of refining. The detrimental effect of oxygen was recognized way back in 1850 by R. Mushet, who advocated the use of spiegel to deoxidize liquid steel produced by the Bessemer process before casting. Several useful deoxidizers have since been found to effectively deoxidize liquid steel to obtain sound ingots on solidification. The subject has been dealt with earlier in Chapter 4 and will not be repeated again. The use of deoxidizers, unless it forms a gaseous product of deoxidation reaction, tends to decrease the cleanliness of steel. Except where cleanliness is of supreme importance (*e.g.*, ball bearing steels, die steels, tool steels, air and space craft steels, etc.) deoxidation of liquid steel prior to teeming, in the usual manner, is however adequate to eliminate the deleterious effect of excess dissolved oxygen in steel.

Where cleanliness is of supreme importance and where oxygen is to be reduced to a level lower than that attainable by normal deoxidation practice, liquid steel has to be treated under vacuum prior to solidification.

26.2. Nitrogen in Steel

Since the steel making operation is carried out in furnaces open to atmospheric air, a certain amount of nitrogen always finds its way in liquid steel. Nitrogen also comes from the raw materials charged in the furnace and from the nitrogen that is directly brought in contact with the bath along with oxygen during melting and/or refining. Nitrogen content of refined steel very much depends on the steel making process adopted. The problem of nitrogen has been discussed in detail in Chapters 8 and 20, which may be recollected here again.

Nitrogen content varies from 0.002% in the Oxygen-Steam-Thomas process to 0.022% in a Thomas process operated under most favorable conditions. In an open hearth process it is in the range of 0.0035%–0.005% and causes no trouble in the manufacture of the usual products. Nitrogen content in this range is attained in an open hearth practice only if the necessary carbon boil takes place during refining. It is better to have nitrogen below 0.002% in the deep drawing type of steels. This is readily possible in an electric steel making process because nitrogen does not come in contact with the bath in this process.

In oxygen steel making processes, a high nitrogen absorption can take place in the hot zone, where oxygen jet impinges on the metal, if the jet gas contained nitrogen. The purity of oxygen used is thus very important, and it should not be below 99.5% to produce steels with less than 0.002% nitrogen, since these processes are meant to produce deep drawing–type dead soft steels.

Nitrogen is picked up by liquid steel during tapping as it comes in direct contact with atmospheric air. The amount of nitrogen thus picked up depends directly upon the height of the furnace tap hole from the ladle. It also depends upon its own content in steel; the pick-up is more if it is already low (*e.g.*, in the LD process) and is less if it is already high (*e.g.*, in the Bessemer process). The LD process showed increase in nitrogen content during tapping as follows:

Rimming steels	0.0005%–0.0006%	nitrogen
Semi-killed steel	0.0003%–0.0010%	-do-
Killed steels	0.001%–0.0015%	-do-

Sulfur content of steel has often been found to be related to its nitrogen level. Low sulfur content is accompanied by high nitrogen content in the Bessemer process, whereas the reverse is true in the case of an electric arc process. Tapping temperature within the usual range does not indicate any appreciable variation in the nitrogen content of steel. The addition of recarburizers, in particular, does increase the nitrogen content of steel, as nitrogen is always associated with the usual recarburizers.

Nitrogen content is decreased while teeming rimming steels due to the scavenging action of the evolved gases during rimming, whereas it is increased in semi-killed and killed steels.

Summary. Except in a small proportion of steel wherein the hardening influence of extra nitrogen content is made use of in developing, in combination with aluminum, better abrasion resistance, by and large, nitrogen is a deleterious impurity in steel. It makes steel strain brittle so that it cannot be cold worked much without intermediate annealing, and hence in general, and more particularly in deep drawing–type of steels, it must be less than 0.002%.

Fortunately, bulk steels with such low nitrogen content can be readily produced by adopting oxygen or modified processes of steel making to meet the vast requirements of sheets, strips, etc. No further treatment except proper care in steel making as such is, therefore, required to obtain low nitrogen steels.

The vacuum treatment that is adopted to reduce oxygen and hydrogen contents of steel, side by side, does reduce some nitrogen content as well. However, in general, vacuum treatment is not specified for the removal of nitrogen alone.

26.3. Hydrogen in Steel

Hydrogen is formed when water vapor comes in contact with steel or slag. The amount of hydrogen dissolved in steel varies with the partial pressure of hydrogen, composition of steel, and its temperature. Hydrogen content is expressed as cc or ml per 100 g steel or parts per million (ppm). 1.2 cc hydrogen per 100 g equals 1 ppm (0.0001%). The solubility of hydrogen in solid iron at its melting point is nearly 10 cc/100 g. Liquid steel may contain as high as 20 cc/100 g in high alloy steels made under unfavorable conditions. It constitutes the serious problem of hairline cracking in fully killed high-alloy steels with large cross-sectional area. In rimming steels it causes no problem because it is reduced well below the *safe* limit by the scavenging action of gases evolved during cooling. In semi-killed steels some believe that it causes defects that cannot, however, be identified with certainty. The ductility of steel is decreased with increasing hydrogen content, and this effect becomes greater the higher the tensile strength of the steel. A high level of hydrogen content leads to the formation of even blow and pin holes. The deep-seated holes weld up but the shallower ones are punctured and oxidized and hence, need increased surface dressing operation.

The sources of hydrogen in steel making are:

1. Wet and rusty scrap
2. Wet solid charge materials (other than scrap)
3. Steam fed into the furnace (*e.g.*, atomization of oil in an open hearth)
4. Atmospheric humidity, if air blast is used for refining or burning
5. Wet refractory channels, runners, containers, etc., with which steel comes in contact prior to its solidification

In order to decrease the final hydrogen content of steel, rusty and wet solid materials should not be charged in the furnace, as far as possible. Alternatively, these materials should be dried before charging, to minimize hydrogen pickup. Similarly, all refractory linings must be thoroughly dried before metal comes in contact with them prior to solidification. In addition, in every process, refining is carried out in such a way that the hydrogen content of bath is reduced during refining and is not allowed to pick it up subsequently.

26.3.1. Open Hearth

Moisture is present in the gas phase. It dissolves in slag and then after diffusion, it dissociates at the slag-metal interface so that hydrogen is picked up by the metal phase. CO bubbles during the boil scavenge dissolved hydrogen out of the bath. The rate of desorption of hydrogen is much more than that of absorption during the boil. However, if the heat is held inside the furnace for long without the boil it may pick up hydrogen, so much so that more than what has been desorbed during the boil is picked up during the holding period. The heat is refined by long and vigorous boil and is tapped either during the tail end of the boil period or immediately thereafter to obtain steels with low hydrogen contents. This may or may not be a practical proposition. The hydrogen level can be minimized by employing high basicity (as otherwise permissible), less slag volume, low oxidizing potential of slag, and low temperature (as far as possible).

26.3.2. Electric Arc Furnace

Water vapor content of the gas phase is much less in an arc furnace than that in an open hearth. Hydrogen content of electric steel is therefore less than that when the same steel is made in an open hearth. Besides the usual precautions, hydrogen pickup can be minimized by keeping the reducing period, if adopted, to a minimum because in the absence of boil the bath tends to pick up hydrogen via the slag phase in this period.

26.3.3. Bessemer

In this process, hydrogen content of the bath increases with atmospheric humidity, with the rest of the conditions remaining the same. In the modified practice of oxygen-steam process the steam is shut off a minute ahead of the blow to obtain normal level of hydrogen, in refined steel.

26.3.4. BOF Processes

Hydrogen content of steel in these processes is increased mainly due to the increase in the atmospheric humidity transferred to the metal bath via lime that is added as a flux. Freshly burnt lime in lumpy form is therefore used to minimize moisture pickup by the metal. It is added early in the below, as is the case in normal LD blow, so that enough time is available for whatever hydrogen is transferred from lime to the metal to be removed during the vigorous boil that takes place almost all through the blow. This ability of the LD process to produce low hydrogen steel could be advantageously used to make alloy steels.

As against this, fine powdered lime in large proportion is added in the LDAC process almost all through the blow. It increases the hydrogen level in the steel at the tap.

26.3.5. Tapping and Teeming

In general, if steel contains hydrogen less than 5 cc/100 g, it picks up hydrogen from the open atmosphere during tapping and teeming, whereas it slightly desorbs hydrogen to the atmosphere if it is more than this level. If the molds are not rusty, proper grade of mold dressing is used, and the refractories in the mold assembly are thoroughly dried, no significant pickup of hydrogen takes place in the mold. The finishing additions added in the ladle or the mold are preheated to minimize their moisture contents.

26.3.6. Means to Obtain Low Hydrogen Contents

In general, hydrogen content of less than 2 cc/100 g is considered to be a safe limit to avoid the possible defects that arise due to the presence of hydrogen in steel. The typical levels of hydrogen contents in various important steel making processes are shown in Table 26.1.

Table 26.1.
Typical levels of hydrogen contents in various steel making processes.

	Approximate H_2 content (cc/100 g)
LD (low P iron)	2.0–2.5
LDAC	2.0–6.0
BOH (hot metal)	3.0–8.0
BOH (cold metal)	6.0–12.0
Electric arc (oxidizing slag)	2.0–6.0
Electric arc (reducing slag)	4.0–8.0

In the above ranges, the hydrogen level can be kept at the lower level if the following precautions are taken:

1. Avoid use of rusty scrap.
2. Avoid use of moist raw materials.
3. The refractory linings must be thoroughly dried.
4. A vigorous and long carbon boil must take place during refining.
5. Bath should not be left in the furnace for long in *quiet* state (*i.e.*, without boil).

Jet degassing or flushing can be adopted to decrease hydrogen content if it is high. An inert gas such as argon is bubbled through liquid steel so that hydrogen is desorbed in these bubbles. Hydrogen content can be reduced to 3 cc/100 g by this method. High cost of argon is the prohibitive factor in applying this method commercially. Nitrogen has also been used in this way but without much advantage.

Prolonged annealing in the range of 600°C–650°C for a sufficiently long time allows hydrogen to diffuse out. Until recently, this was the only technique by which hydrogen content was reduced to below 2 cc/100 g in commercial practice. The annealing time increases with increase in cross-sectional area and the initial hydrogen content of steel.

It is now possible to reduce the gas content of steel in general and hydrogen content in particular, below the safe limits by vacuum treatment of steel, at any stage after the refining but prior to its solidification in molds. The most deleterious gas impurity (*i.e.*, hydrogen can be reduced to less than 2 cc/100 g by vacuum treatment of steel. It is the subject matter of the next chapter.

CHAPTER 27

VACUUM TREATMENT OF LIQUID STEEL

In This Chapter

- Principles
- General Considerations
- Vacuum Raising Equipment
- Degassing Processes
- Ladle Degassing
- Stream Degassing
- Recirculation Degassing
- Comparison of R-H and Tank Degassing

The gases dissolved in steel, beyond a certain limit, are undesirable because their presence finally leads to several defects in steel products. The vacuum melting processes are meant to produce steels having low gas content and inclusions either in a relatively very small amount at a reasonable cost or in large quantities at considerable expense. These processes are not being described here. The processes that have been developed to treat large tonnages of steel (up to 300 t), produced by the large-scale commercial steel making processes, under reduced pressure, are being described in the present chapter. These are also known as vacuum degassing processes. The first commercial unit of this type—the Bochumer

Verein process—was installed in Germany at the Bochumer Verein plant in 1952–1953. Several different processes have since been developed and put into commercial practice. The total number of vacuum degassing plants now in commercial operation all over the world may be nearing as much as one thousand five hundred.

Vacuum degassing was originally meant to reduce hydrogen from steels that were prone to hairline cracking; hence, the name. It was soon realized that deoxidation of steel, by way of carbon–oxygen reaction, could also be carried out under vacuum to produce steels of much better cleanliness since the product of deoxidation, a gas, escapes out of the melt. The present day degassing plant may involve one or more of the following functions:

1. To remove hydrogen from steel so that the prolonged annealing treatment is not required, particularly in forging quality steels
2. To improve cleanliness by removing oxygen in the form of CO gas
3. To produce steels of a very low carbon content (below 0.03%) or even down to 20 ppm by transferring part of the refining from furnace to the degassing unit
4. To bring about desulfurization of steel by reagents carried along with an inert gas such as argon bubbled through the bath
5. To ensure better control of the chemical composition of steel by adding the requisite amount of additions under vacuum

In view of these purposes, the process may more aptly be called as vacuum treatment of molten steel prior to its solidification, and vacuum degassing is only a part of the activities involved.

27.1. Principles

Vacuum treatment consists of exposing liquid steel to a low-pressure environment. The equilibrium solubility of diatomic gases such as H_2, O_2, and N_2 in steel is given by Sievert's law as:

$$\%\ \text{gas dissolved} = k\sqrt{\begin{matrix}\text{partial pressure of the gas}\\ \text{in ambient atmosphere}\end{matrix}}$$

The excess dissolved gases are desorbed from steel on its exposure to vacuum. The kinetic factors may prevent the system from attaining equilibrium as per Sievert's law. The value of *k* in the above relationship will alter according to the interaction of the gas atoms with those of iron and other constituents present in steel.

Hydrogen does not form any stable hydrides in steel and hence can be readily removed to a level of less than 2 cc/10 g in commercial practice wherein usually the pressure is less than 10 mm of Hg. Almost 80% of the hydrogen present in liquid steel can be readily removed by vacuum treatment in practice.

The carbon–oxygen reaction is very much influenced by the partial pressure of CO in contact with the bath. This reaction takes place in steel on its exposure to vacuum (see Figure 4.1). The product of reaction (*i.e.,* CO escapes out, and deoxidation is achieved with better cleanliness in the resultant product). If the steel is not already deoxidized by strong deoxidizers such as Si, Al, Ti, etc., vacuum treatment can remove a substantial portion of oxygen present in steel. The percentage of oxygen present in steel that is removed by vacuum treatment is approximately in the range given below:

Killed steels	25%–30%
Semi-killed	50%–75%
Rimming	70% –85%

Theoretical considerations indicate that deoxidation by carbon will take place in preference to silicon at a pressure of approximately 10 mm Hg. There is, however, no conclusive evidence to prove that silicates, if present in steel as slag or inclusions, get reduced by carbon, under vacuum of less than 10 mm Hg to further improve its cleanliness.

It is not economical to produce steels containing less than 0.06% C by usual commercial processes. If an undeoxidized steel containing less than 0.04% C is exposed to vacuum, it can be readily reduced to a level of 0.01% C by way of carbon–oxygen reaction. But to attain a 20 ppm level, ingenious strategies are required.

In practice, if the CO evolution is violent, killing agents such as Si and Al are added to achieve a moderate rate of degassing.

Nitrogen present in steel is often stably bonded as nitrides in steel and as such only a small amount, approximately 10%–15% of the total, can be removed during vacuum treatment. In many processes, after vacuum treatment, steel is teemed in ingot molds in open air. In that case the same amount of nitrogen may again be picked up by steel from atmospheric air. For both of these reasons, vacuum treatment *per se* is not employed solely to reduce nitrogen content of steel in practice. Whatever reduction is achieved is just by the bye.

The recovery of alloying elements and the efficiency of deoxidizers will be extremely high under vacuum if added after the degassing is nearly complete. Almost all vacuum treatment processes, therefore, provide facilities for adding these to liquid steel at the appropriate time during degassing. The amount added should not be so large as to quench the bath beyond a certain acceptable limit. These are added towards the end but while enough agitation is still present to homogenize the composition. The chemical composition of steel can thus be much better controlled to obtain closer specifications.

27.2. General Considerations

The desorption of hydrogen and carbon monoxide is a gas-metal interfacial phenomenon. The effectiveness of vacuum treatment, therefore, increases with increase in the surface area of liquid steel exposed to vacuum. The development of vacuum treatment processes has been in the direction of improving the surface area of steel exposed to vacuum. The metal is, therefore, made to flow in the form of a thin stream or even to fall in droplets to accelerate the degassing process. It also eliminates the adverse effect of ferro-static head during degassing of steel.

The temperature of liquid steel is bound to drop during vacuum treatment unless deliberately heated by an auxiliary device. The more the surface area there is of steel and the more prolonged the treatment is, the more the heat loss will be. To offset this, the tapping temperature of steel is generally kept 30°C–50°C higher than the steel (of the same composition) that is not treated under vacuum. The degassing time must be kept to a minimum, consistent with adequate removal of gases from steel. In some processes, a radiation shield and/or a layer of slag cover are used to minimize heat losses from steel. In some other processes, steel is heated during the treatment to counter-balance, at least partially, the heat loss.

The degree of vacuum employed depends upon the degree of degassing required and the cost of generation of the vacuum. The degree of degassing increases with the degree of the vacuum but it means more cost in terms of equipment and its operation. In general, vacuum of the order of

1–10 mm Hg (1 mm Hg ≡ 1 Torr ≡ 100 microns) is employed in practice to obtain optimum conditions. Many operators believe that pressure lower than this value is a shear *waste of money.* The degassing action give a boiling appearance to steel. The vigor depends upon the rate of pumping of gases. The pumping equipment should be of adequate size to give necessary vigor that is beneficial during the process. High vigor can be controlled by adding killing agents, such as aluminum, so that steel does not flow out of the container. Oxygen level can be decreased below that attainable by vacuum treatment by adding strong deoxidizers such as Al and Ti after the degassing is almost over rather than by lowering the pressure.

27.3. Vacuum Raising Equipment

Two types of gas-pumping equipment are in use. Mechanical pumps are employed if small pumping capacity and low pressures (≅ 0.01 mm Hg) are required. Multi-stage steam ejectors are used when larger pumping capacity and relatively higher pressures (1–10 mm Hg) are needed. In general the steam ejector is best suited for large-scale industrial use. The operator has much better control over a steam ejector than over mechanical pumps. The steam ejector is described in brief below.

In principle, high-pressure steam is ejected through a level-shaped nozzle so that on its expansion the pressure energy is converted to kinetic energy. The gas molecules in the chamber are carried along with the steam, which is separated out by condensation, and the gas is let out. This chamber is connected to the degassing chamber assembly. Several such steam ejectors are connected suitably in series, as shown in Figure 27.1, to obtain the desired speed and degree of evacuation. Usually a four-stage or sometimes even up to a six-stage ejector pump is used. Cold water is sprayed in each condenser to condense the steam to reduce the gaseous volume. The water thus formed is collected and removed by a barometric leg that may be up to 10 meters in height. It ensures that the condensing water is not sucked back to the degassing chamber in the event of a vacuum breakdown. Several plants use much shorter legs and employ other safety devices to reduce the headroom requirements.

Pumping sets are designed to meet the particular requirements, which vary with the type of degassing process adopted. For example large pumping capacity is needed in the beginning to evacuate the chamber in ladle degassing. The pumping load decreases as degassing proceeds. In contrast

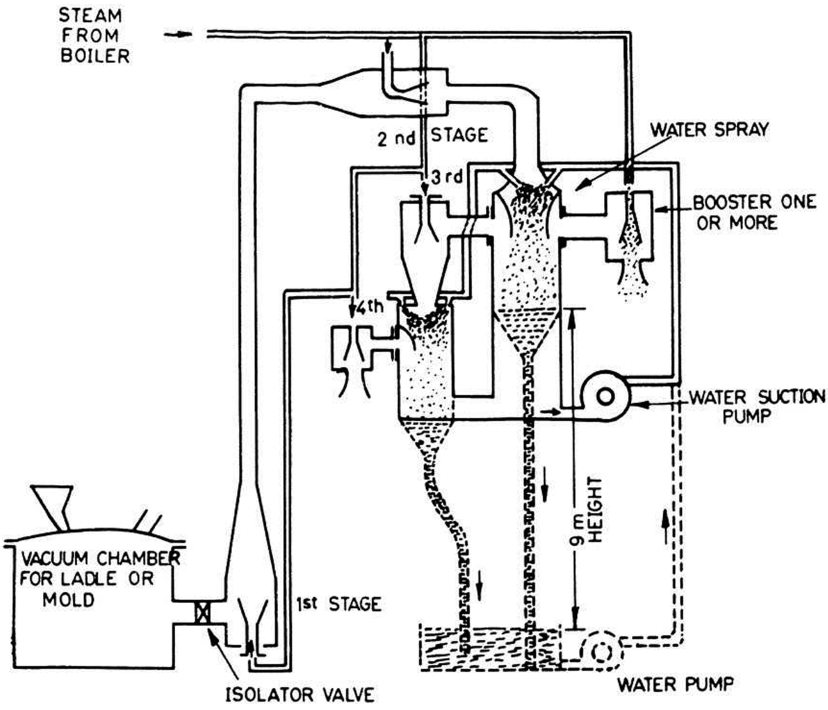


FIGURE 27.1. Multi-stage steam ejector. The dotted portion indicates the design if a water suction pump is not used. In this case, the danger of water suction does not arise even if the vacuum accidentally fails.

to this, uniform pumping speed is required in stream and circulation degassing. In fact, the pumping speed determines the rate of teeming in stream degassing and the rate of circulation of metal in circulation degassing.

Booster ejectors are provided to increase the pumping capacity to speed up evacuation. These are single- or two-stage ejectors without condensers. Their steam consumption is more and they are open to atmosphere. They are connected after the first two stages in a multi-stage ejector.

Adequate provision for steam-raising equipment should be made on the pitside to adopt these pumps for vacuum treatment of steel. A good amount of water supply at constant temperature is also required to run the multi-stage steam ejector pumps satisfactorily.

A steam ejector system is relatively cheap to install and, cheap and easy to maintain because, except for the valves, very few moving parts are involved. It takes less space than equivalent mechanical pumps.

27.4. Degassing Processes

A large number of processes are employed in commercial practice for vacuum treatment of molten steel after tapping. These can be broadly classified into three different groups:

(1) Ladle Degassing. Liquid steel is held in a ladle, which is put inside a vacuum chamber. Steel may be stirred by bubbling an inert gas or by an electro-magnetic stirrer while being exposed to vacuum.

(2) Stream Degassing. Liquid steel flows down in the form of a stream from the furnace or ladle to another ladle or mold during its exposure to vacuum.

(3) Circulation Degassing. Liquid steel is either continuously or intermittently circulated during its exposure to vacuum. The circulation is brought about in various ways (*e.g.*, the *D–H*, and *R–H* continuous degassing circulation systems).

These processes are being described below one-by-one.

27.5. Ladle Degassing

In principle the teeming ladle, after tapping, is kept in a vacuum chamber, which is then evacuated. The ladle is removed after a predetermined time when the correct teeming temperature of steel is reached. Teeming is carried out in open atmosphere.

In a still bath of fully killed steel the effectiveness of degassing decreases from the top to the bottom of the ladle because of the increasing ferrostatic head of steel. The metal bath is therefore stirred by bubbling an inert gas such as argon or by electro-magnetic induction. As the pressure falls the vigor in the ladle increases and the bath appears as if it is boiling. The ladle is not filled completely. A considerably free board, up to approximately 30% of the total volume (*i.e.*, approximate °C; height), should be provided or else the metal may spill over the edges during degassing.

In some processes, steel is heated by an electric arc or induction to compensate for the heat loss during degassing and help dissolve a large amount of deoxidizers and alloying additions. These additions are made when degassing is nearing completion but well ahead of the point when stirring of the bath due to degassing practically stops. Stirring is essential for homogenization of the bath composition.

The pumping capacity should be related to the chamber size and the final level of gas content required in steel. The teeming ladle is kept in the tank, the cover is replaced, and evacuation is started. In a well-designed plant the chamber is evacuated at the earliest possible time, in order to decrease the time of degassing. Pumping continues until the pressure drops to the ultimate value, which may be 1–10 mm Hg and, at this stage the steel is considered to be degassed adequately. Finishing additions are made, and the pressure is slowly built up to atmospheric pressure to remove the ladle for teeming in a normal way. Many operators carry out degassing until the teeming temperature of steel is reached. The temperature of the bath is kept under watch throughout the process. Even television cameras are mounted in the cover of the chamber to observe the process of degassing.

The rate of degassing is controlled by aluminum addition rather than by controlling the pressure in some way.

A minimum of slag should be allowed to flow in the ladle during tapping, as it hinders the gas desorption process. A certain amount of slag is beneficial. The (FeO) content of the slag keeps supplying oxygen to the bath; that is, the reaction

$$(\mathrm{FeO}) + [\mathrm{C}] = [\mathrm{Fe}] + \{\mathrm{CO}\}$$

takes place and the final oxygen level is decided by the (FeO) content of the slag. The carbon content may also be thus decreased.

In general, degassed steels are teemed at almost 20°C less than those at which corresponding untreated steels are teemed. Some of the important ladle degassing processes are described below:

Gas stirring. Argon is bubbled through the bath, as shown in Figure 27.2, to hasten degassing of lower layers of full killed steel in the ladle. Gas is either introduced through a refractory-covered tube dipped in the bath or through a porous refractory plug fitted in the bottom of the ladle. The latter method is better because it allows better control of gas bubbled through the bath. Gas is introduced as soon as evacuation is started. It not only purges the gases but also homogenizes the bath very thoroughly.

This method is quite popular and is sometimes also known as the Finkle or the IRSID process.

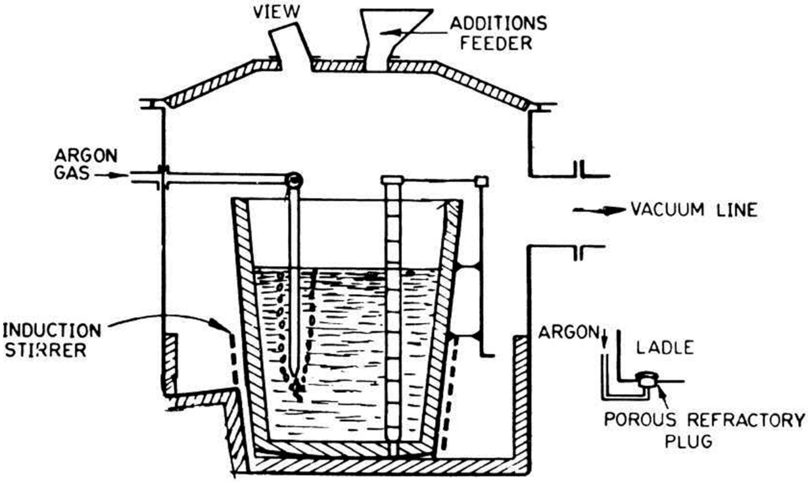


FIGURE 27.2. Ladle degassing showing gas stirring and electromagnetic stirring of the bath. Either of the two is employed in practice. Argon can also be bubbled through a porous plug fitted at the bottom, as shown in (a).

Gas stirring and arc heating. The process is exactly similar to the above except that the metal is heated during degassing by an electric arc. The electrode is inserted from the roof of the vacuum chamber.

Induction stirring. In this process liquid steel is stirred by an electromagnetic (induction) stirrer placed around the ladle during degassing. The ladle shell in this case has to be made of non-magnetic austenitic steel. The arrangement is shown in Figure 27.2. The rest of the operation is similar to what has been described earlier.

Induction stirring and arc heating. This is also known as the ASEA-SKE ladle degassing process and has been described in Chapter 23. In addition to the induction stirring, the bath is heated by an electric arc as in the gas-stirring and arc-heating process. Alloy steels of closer chemical specifications are being made by this process in several countries.

27.5.1. Gas Contents

The final gas level depends upon the pressure and the time of exposing steel to vacuum. Although equilibrium is not attained, hydrogen is generally reduced to below 2 cc/100 g from around 4–6 cc/100 g present in steel at tapping. The process can thus be used for casting even large forging ingots.

Even if the nitrogen content gets reduced during degassing, it attains the same level during teeming in open air.

Oxygen removal very much depends upon the amount of slag and its (FeO) content that goes in the teeming ladle, and hence it varies considerably. The oxygen level usually attained in this process is not comparable to that obtained from a steam degassing plant.

27.5.2. Summary

Ladle degassing is the simplest of all the degassing processes and hence is very widely adopted. The ladles are generally lined with high-alumina bricks at the top level whereas for the lower portion the usual fire bricks are employed. The vacuum chamber cover is lined to protect it from direct radiations. Similarly, the chamber bottom is also lined to protect it against any accidental failure of a ladle stopper rod or the side gates.

27.5.3. Mold Degassing

This is very much akin to the ladle degassing process. After pouring mold(s) in air its mouth is closed and connected to a vacuum line to expose steel to vacuum during solidification. It does not need extra super-heat in the metal at the beginning. At least some steel solidifies before it is exposed to vacuum and hence it is not as effective in degassing as other processes. Now since ingot casting is abandoned, its applicability is over.

27.6. Stream Degassing

Molten steel is exposed to vacuum in the form of a stream of metal flowing from one vessel to another. Sudden exposure to a vacuum results in such a rapid rate of degassing that the stream explodes into thin droplets. The large increase in surface area of steel in the form of falling droplets leads to very efficient degassing and surface reactions. Steel is poured either from a teeming ladle, a pony ladle, or a tundish. It is received either in a ladle or in a mold kept inside a vacuum chamber.

If the metal is received in a mold, it solidifies under vacuum and the resulting ingot has the lowest gas content. If the metal is received in a ladle, it is teemed in a normal way later on in open air. The major amount of degassing takes place during the fall of metal drops and as such the height of the pouring vessel from the receiving vessel is an important parameter in the design of the equipment. A small amount of degassing does take place after the metal is received in the ladle. Only an insignificant amount of degassing takes place when the metal is being solidified in the mold under vacuum because of its narrow cross section and chilling effect.

The following processes are used in commercial practice wherein degassing is accomplished by pouring steel from one vessel to the another:

1. Ladle-to-mold degassing
2. Ladle-to-ladle degassing
3. Tap degassing
4. Gero vacuum casting process
5. Therm-I-Vac process

Whatever process is used, the vacuum pumps and vacuum chambers must be properly chosen to suit the requirements. The vacuum chamber should be big enough to accommodate the ladle or the mold. The top cover of the vacuum chamber should be adequately away from the metal level in the ladle so as not to get heated by radiations beyond a safe limit. The cover is lined by refractories from inside for additional safeguards. The entrance in the top cover of the chamber, through which steel stream enters the chamber, is sealed by an aluminum disc before evacuating the chamber. As the metal steam explodes on entering the vacuum chamber, the metal drops tend to spread out considerably. This is undesirable because it may fall outside the ladle or the mold. In general, a *stream director* is used to confine this spread. It is a refractory tube that is expected to stand the conditions without any significant erosion and is placed, from inside, around the entrance through which steel flows in.

The mold or the ladle is preheated inside the chamber before the chamber is evacuated. The chamber is evacuated by connecting it to the pumps and opening the cut-off valve. An inlet valve is provided to let air or nitrogen in, at the end of degassing, to raise the chamber pressure to atmospheric level before opening the chamber.

The tapping ladle, the pony ladle or the tundish that are used to pour steel are all provided with a nozzle in the bottom and stopper rod assembly. It is placed above the vacuum chamber so that the nozzle is just above the entrance in the chamber cover to let steel flow in. The pouring vessel is sealed to the chamber cover during pouring. The rate of pouring is controlled by the stopper rod assembly provided therein.

If a pony ladle or a tundish is used, the stopper rod is not raised until these are filled to almost two-thirds of their capacities. Once the stopper rod is raised, metal flows down, the aluminum seal is melted, and metal enters the vacuum chamber. Alloy additions can be made if the metal is received in a ladle. If it is vacuum-cast, no additions are possible.

27.6.1. Ladle-to-Mold Degassing

Preheated mold with a ho- top is placed inside the vacuum chamber. A pony ladle is placed above the chamber. Steel is tapped with approximately 30°C superheat and the teeming ladle is transferred to a position just above the pony ladle. Steel is bottom-poured in the pony ladle. Only one ingot may be vacuum-cast from one heat, or several tappings from different furnaces may be poured in the pony ladle only after the other to cast as large as a 400-t ingot. The pony ladle should always be nearly two-thirds full until the end of pouring.

27.6.2. Ladle-to-Ladle Degassing

The scheme is shown in Figure 27.3. The teeming ladle from the furnace is placed on top of the vacuum chamber, and they are sealed to each other. The operation is similar to that of the ladle-to-mold degassing process. Alloy additions are made under vacuum. The metal at tap should have 30°C–50°C superheat since it is to be teemed in open air in a normal way after vacuum degassing.

27.6.3. Gero Process

This is a modification of the ladle-to-ladle degassing process capable of casting multiple ingots from a single heat. The arrangement is shown in Figure 27.4. A gas-tight cover assembly is provided with each mold to expose the metal stream to vacuum during casting. Molds are provided with hot-tops. A mobile mechanical pumping unit moves on a trolley from mold to mold and is connected to the exhaust opening of the mold-top assembly. After evacuating the mold, teeming is commenced. An aluminum disc ensures the sealing of the entrance for a steel stream. After the teeming is over the pumps are cut off and air is let in to raise the pressure to atmospheric level before disconnecting the mold from the pumping line. This is primarily used for hydrogen removal.

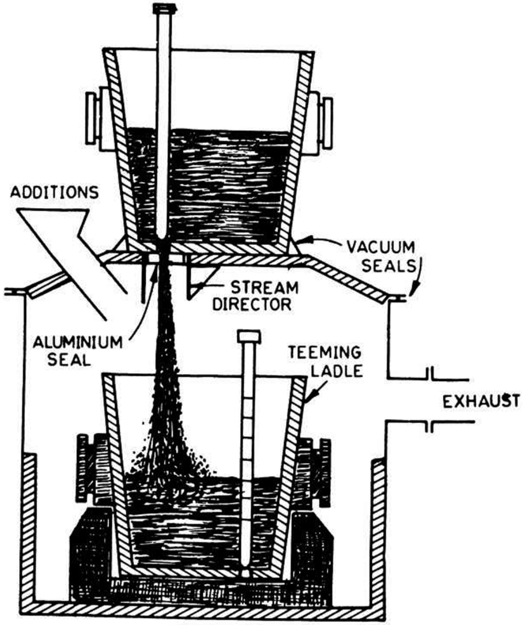


FIGURE 27.3. Ladle-to ladle degassing.

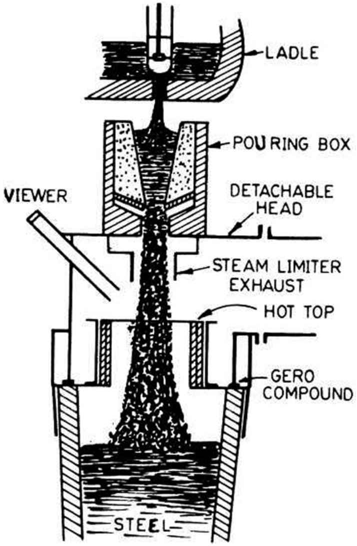


FIGURE. 27.4. Gero process of degassings.

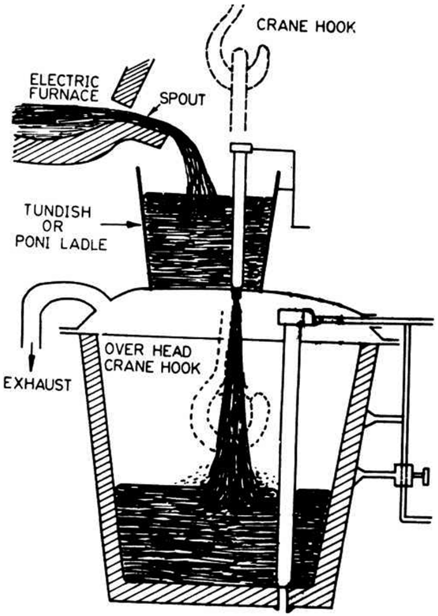


FIGURE 27.5. Tap degassing.

27.6.4. Tap Degassing

The arrangement is shown in Figure 27.5. The furnace is tapped in a ladle via a tundish placed on the ladle. The ladle acts as a vacuum chamber. It is, therefore, closed from the top with a special cover containing openings for exhaust, steel entrance, and stopper rod insertion. O-ring seals are used to ensure airtight joints. Steel with a superheat of 25°C is tapped from the furnace into the tundish. When it is two-thirds full, steel is allowed to flow into the teeming ladle under vacuum. An aluminium disc is used to seal the entrance before evacuating the ladle.

Later, steel is teemed in open air in a normal way.

27.6.5. The Therm-I-Vac Process

The equipment in this process is designed to suit varied requirements. It has an induction furnace placed inside a vacuum chamber that also houses a series of molds beneath the furnace. It can be used to melt and refine steel scrap in the induction furnace under vacuum. After careful adjustment of its composition it can be cast under vacuum. Alternatively, liquid steel can be stream degassed by pouring it in the induction furnace. It may also be subsequently cast under vacuum. In this case furnace acts as a ladle wherein the composition and the temperature can be better controlled. The scheme is shown in Figure 27.6.

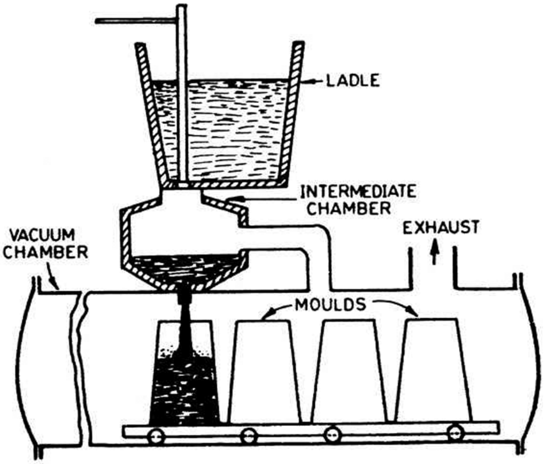


FIGURE 27.6. Therm-I-Vac process of degassing.

27.6.6. Summary

In stream degassing, steel is exposed, drop by drop, to vacuum. The size of the drop is important as it determines the efficiency of degassing. Although it hastens degassing, it also increases temperature losses during the process. The geometry of the system is therefore important and should be such as to keep the heat-losses to a minimum.

The rate of pouring and the level of vacuum are yet other parameters of interest to investigate the economics of these processes for degassing.

Unlike the other stream degassing processes, the Gero process is not meant to accomplish a significant amount of carbon–oxygen reaction in addition to removal of hydrogen. The final hydrogen content attainable by these processes is in the range of 1–2 cc/100 g and is well below the acceptable limit for most steels.

27.7. Recirculation Degassing

Molten steel is exposed to vacuum in the form of either a circulating stream or a small portion temporarily lifted from the rest of the metal pool. In commercial practice the former system is known as Ruhrstahl Heraeus or for short, the R-H process, and the latter system is known as Dortmound-Horder or for short, the D-H process. These are described below.

27.7.1. R-H Degassing Process

The process was developed by Rheinstahl Heinrich Shutte at Hattingen, Germany in 1957. It has since been improved, particularly by the Hirohito Works of the Fuji Iron and Steel Co., Japan, and as a result it is being adopted with considerable interest. The later modifications have increased its applicability greatly.

The degassing chamber is shown in Figure 27.7. The chamber is a cylindrical steel shell with two legs, often called snorkels, and openings at the top side are provided for exhaust, alloy additions, observation, and control. It is lined from inside with fire bricks in the upper portion and high alumina bricks in the lower portion that directly comes in contact with steel during degassing. The snorkels are also lined from inside with high-alumina refractory. During degassing, metal enters the cylindrical vacuum chamber through one snorkel and flows back under gravity through the other. The inlet snorkel is designed to a laval shape using refractory. The outlet snorkel is just plain tubular. A refractory tube is attached to each snorkel since it is to be dipped in liquid steel for degassing. These are made of a commercially pure grade of alumina. Lifter gas such as argon is introduced in the inlet snorkel at the point where the alumina tubular tip is attached to it.

Before the actual use, the chamber is variously heated to 900°C–1500°C at different plants by firing fuel up in the snorkels. The chamber is lifted and lowered to an appropriate level in the ladle containing molten steel. The chamber is evacuated and liquid steel just rises in the chamber. The sp. wt. of liquid steel is 6.94 t/m^3 at 1600°C. The atmospheric pressure causes the liquid steel to rise to 1.45 m, above the still-bulk level, in the snorkel, under deep vacuum. The lifter gas is then introduced in the inlet snorkel.

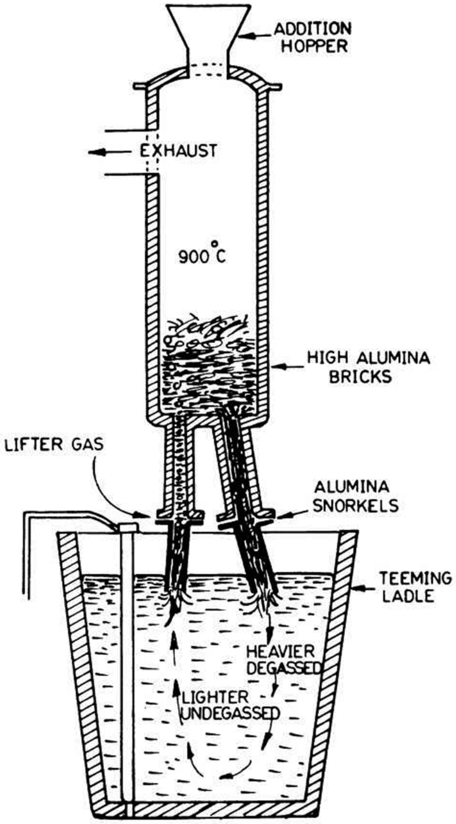


FIGURE 27.7. R-H process of degassing.

The gas expands and rises up, thereby raising the velocity of the steel in the inlet snorkel. The lifter gas bursts in the chamber and likewise steel also explodes on meeting vacuum as in a stream degassing process. The net result of all this is that degassing takes place very efficiently. Gravity causes the steel to flow back in the ladle via the other snorkel. Degassed steel is slightly cooler and denser than that in the ladle and hence it forces the lighter, undegassed steel upwards, thereby ensuring adequate mixing and homogeneity of the bath.

The rate of circulation of metal is controlled by adjusting the vacuum and the rate of flow of lifter gas. The level-shape design is quite critical in this respect. The average rate of circulation is nearly 12 t/min. and nearly 20 min are required to treat 100 t of steel to bring down the gas contents to the usual desired levels. Degassing is quite efficient, and more than 90% of the gas is removed by exposing the metal twice only to vacuum.

At the end of degassing, alloy additions may be made, depending upon the superheat available in steel. Circulation ensures homogeneous composition of the bath. The metal cools down by approximately 25°C–50°C, depending upon the ladle size.

The amount of argon required is 0.015–0.075 m^3/t of steel treated. Every care should be taken to keep slag out of the snorkels and the chamber to avoid slag-metal reactions and formation of inclusions.

At the end of the treatment, the vacuum is broken as described earlier, the chamber lifted out, and the steel is teemed in a normal way in open air.

The process has several advantages:

1. The pumping capacity that is required is small because of the small vacuum chamber as compared to any other process of degassing.
2. Heat losses are relatively low.
3. It can produce steels with less than 0.002% C, to meet EDD requirements.
4. Alloy additions can be made to adjust the specifications more closely.

The possibility of using this system directly in an arc furnace is being investigated at some plants. It could thus be used for final stages of decarburization, and the correct teeming temperature could be readily adjusted in the furnace. The steel, however, may pick up gases later on during tapping and teeming. The furnace availability is also reduced. The financial advantages of such an improvement over the use of ladle are yet to be conclusively proved. It is doubted that even the cleanliness of the product may not be improved.

The process is also being modified in such a way that the steel is exposed to vacuum only once. This makes the process continuous and may prove to be useful, particularly in conjunction with subsequent continuous casting. BISRA has reported some success in this respect. Many problems regarding the delivery of steel of uniform chemical composition and correct temperature in such a continuous process are yet to be solved satisfactorily for its commercial application. It is expected that metal with less of a superheat would be required for continuous degassing at a low capital cost.

One of the chief operational problems faced by vacuum treatment units, when the chamber is very small, such as that of an R-H unit, is the sudden exposure of steel to a very high degree of vacuum. This happens because of the connection of an already-evacuated line to these units. This connection must be made slowly. Any sudden connection of the vacuum line with the R-H vessel may make the steel bath explode and even jam the vacuum inlet connection. It is a very troublesome breakdown problem.

27.7.2. D-H Degassing Process

In this process a small portion, approximately 10%–15% of the total steel in the ladle, is treated at a time under vacuum. The process is repeated until required degassing is achieved. It was developed by Dortmund-Horder-Huttenunion in Germany. It is also known as a *lifter* degassing process.

The arrangement is shown is Figure 27.8. The vacuum chamber has a long leg to be dipped in the steel pool in a ladle.

The chamber is lined from inside with fire bricks in the upper portion and high-alumina bricks in the lower portion. The leg, or snorkel, is lined from both sides since it is dipped in liquid steel. The chamber is provided with an exhaust, a hopper for alloy additions, and a heating device. The length of the snorkel must be more than 14 m because atmospheric pressure would force steel to rise to such a height when the chamber is under vacuum.

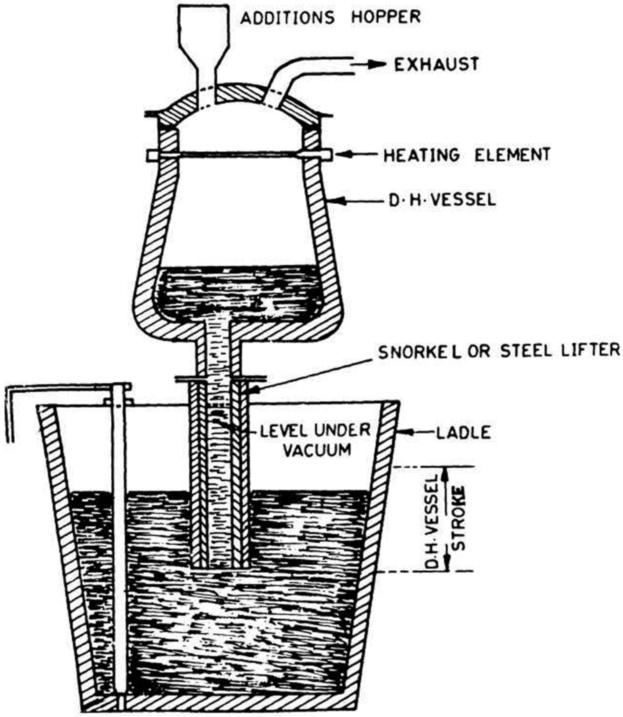


FIGURE 27.8. D-H process of degassing.

A preheated chamber is lowered in a tapping ladle so that the snorkel tip is dipped below the surface of liquid steel. The evacuated chamber is moved up and down so that steel enters the chamber and is exposed to vacuum. Mixing takes place as in the R-H process. The chamber is moved through 50–60 cm with a cycle time of approximately 20 seconds. Since 10% –15% steel is exposed at a time, some 7–10 cycles are required to expose theoretically the entire steel once. Adequate degassing is possible in approximately 20–30 cycles, and an additional 5–10 cycles may be required, depending upon the alloy additions to ensure compositional homogenization. Therefore, nearly 15–20 minutes are required for degassing a ladle of almost any size.

A layer of slag is deliberately kept on the steel bath in the ladle to minimize heat losses. Slag, however, must not enter the snorkel. A steel cone is, therefore, fixed at the tip of the snorkel while lowering the chamber in the ladle. The tip of this cone penetrates the slag layer without allowing it to enter the snorkel. The snorkel should be dipped in steel for more than 60–70 cm so that the tip does not come above the surface of steel during its up stroke.

Initially the pressure in the chamber is high, but as degassing proceeds it decreases and settles to some value that is usually taken as an indication of completion of degassing. The alloying, vacuum breaking, et cetera, are carried out as described earlier, and the steel is teemed in a normal way in air.

Now-a-days, magnesia bricks are used to line upper part of the chamber. The lower part is lined with fusion-cast magnesia and the snorkel is lined from both sides by a high-alumina refractory. The snorkel lining lasts for 90–100 heats, the lower part of the chamber for nearly 400 heats, and the upper part lasts for some 800–1000 heats.

The D-H process possesses most of the advantages of the R-H process, and in addition it is possible to work with much less superheat because a heating device such as a graphite resistor rod is provided in this process to compensate for part of the heat losses during degassing.

The largest unit in operation works on a 400-t ladle. However, several 100-t plants are in operation in various parts of the world.

27.8. Comparison of R-H and Tank Degassing

Over the years, only two dominant processes of vacuum treatment seem to be left in the field for adoption, viz. the tank or ladle degassing and the R-H recirculation process; the others have been left aside. The ladle degassing is adopted when end gas contents and carbon are not too low. This can give comparable hydrogen content and around 50 ppm carbon even after adopting vacuum decarburization. It is also much cheaper to install and operate as compared to the R-H process. On the contrary, R-H is not only costly to install but it is also costly to operate. However, it is far more efficient as compared to the ladle process. It is quite capable of achieving the 20 ppm carbon level required for making IF grade steels because every bit of steel is exposed to a vacuum turn by turn.

The ladle process, even after adopting bottom purging, is not that much efficient for vacuum decarburization. The reasons can be seen in its mechanism of decarburization. In this, the still bath is exposed to vacuum and in spite of bottom stirring, the vacuum-bath interface is practically limited only to the apparent bath surface area at the top, that is, equivalent to ladle diameter. In R-H all the steel is exposed directly to a vacuum turn by turn due to its circulation. Since almost all of the steel is exposed to a vacuum, that is, it forms a metal-gas interface with the vacuum, the C-O reaction is very efficiently carried out.

In the ladle, as the purging neutral gas enters the bath, carbon and oxygen diffuse to the neutral gas-metal interface and react to form CO, which desorbs into the bubble and, since it does not contain any CO, is carried with it and desorbed out. This reaction can continue in proportion to the gas-metal interfacial area provided; that is, in proportion to the volume of gas purged and the number of plugs adopted.

The bubbles that are formed are very small at the tip of the plug-bath interface and hence the rate of decarburization is also small at the bottom. The bubble size increases as it rises because of the decreasing ferro-static pressure of the bath. The C-O reaction does occur all through the rise of the bubble in the bath. The reaction rate decreases with lowering of the contents of C and O. The decarburization rate therefore inversely decreases with decreasing concentrations of C and O. This rate may be of the order of 5–10 C ppm /min, under the usual practice and when the carbon is above 50 ppm.

The rate drops considerably as the carbon approaches a level of nearly 50 ppm. This drop is far more at the bottom and may be perceptible only when the bubble size achieves fairly large size, and this is possible only at the top portion of the ladle. The rate can be increased further only if the purging rate is considerably increased. This rate, however, cannot be increased because of the danger of spillage.

A ladle can be used for further decarburization only if gas bubbles are provided at the top layers where further decarburization reaction may still occur. If this could be achieved by injecting neutral gas near the top, it would still lead to produce 20–30 ppm carbon, as is possible in the R-H process.

This is the reason why R-H is readily preferred over the ladle process because of its very high efficiency for vacuum treatment.

CHAPTER 28

CONTINUOUS CASTING OF STEEL

In This Chapter

- Introduction
- Principle
- Types of Continuous Casting Machines
- Essential Details of a Continuous Casting Machine
- Operation
- Output
- Metallurgical Aspects
- Recent Trends in Continuous Casting
- Prospects of Continuous Casting

28.1. Introduction

Since the production of liquid steel for the first time (nearly more than 200 years ago), it has almost been cast into more regular shapes, known as "ingots", from which the desired semi-finished long or flat product was obtained by mechanical working such as rolling. From the initial small size of ingot, with the developments in steel making technology, it became a common practice to make 5–20 t individual ingots. These ingots were rolled in primary mills to produce blooms or slabs, which were subsequently further rolled to make structural shapes (such as angles, channels, I-sections and so on), rails, merchant products (round, hexagonal, or square cross sections), flats in the form of plates, sheets, strips, etc. The method of producing blooms, slabs, etc. by primary mills had the following disadvantages:

1. A large amount of capital had to be invested in molds, bottom plates, transporting equipment, rails, cranes, shops for stripping and reconditioning of molds, soaking pits, primary mills, etc.
2. In spite of all care defects did occur in ingots (*e.g.*, segregation).
3. A certain fraction, particularly the ends of the ingots, had to be discarded in rolling it into blooms and slabs, which decreased the yield of useful product considerably.
4. If billets are needed, additional rolling mills had to be installed.

All these disadvantages led to the development of continuous casting of steel into blooms, slabs, or even billets, so that much of the above problems are eliminated and the operation, on the whole, became economical. The aim was also to improve upon the quality of the product. A part of the investment thus saved, however, had to be invested in installing continuous casting equipment. The developments gradually led to the development of producing continuously cast billets, thin slabs, and very recently casting almost the steel sheet/plate itself.

The very notion of continuous casting of steel dates back to Bessemer—the inventor of the first process of processing liquid steel—himself, that is, the 1860s, who could think of casting steel plate directly by pouring liquid steel, made by his Bessemer Process, in between two water cooled rolls. He could not however carry out his idea of casting a steel plate in practice. The interest could not revive again until 1930s. The high melting point, high specific heat and low thermal conductivity of steel, made most of the early attempts in 1930s abortive or only partially successful. It, however, proved to be quite practicable for non-ferrous metals and alloys by 1937.

Persistent efforts finally led to a successful installation of a pilot plant in 1943 in Germany for continuous casting of steel billets. By 1953 only a few commercial installations were working successfully. By 1963 more than 60 such machines with 130 strands were in use the world over. The advantages in replacing the conventional ingot casting became very attractive in the 1960s and 1970s. during which period the technology established itself beyond any doubt, in spite of the teething problems that persisted then. By 1980, nearly 30% of crude steel production the world over was continuously cast. It increased to 65% by 1990 with South Korea attaining the level of 94% and Japan 93% share of continuous casting. The turn of twentieth century saw near-complete replacement of traditional ingot mold route of casting, at least in large integrated steel plants. If the old ingot mold casting persists anywhere in any large plant, then it has very special plant-specific reasons for continuing the same.

The technology has been progressing rapidly in the direction of eliminating the rolling part, to the extent possible, or even altogether to produce the near-net shape required in the form of a semi-finished product. The earlier thick slab casting is now increasingly being replaced by the thin slab (approximately 50–60 mm thick) casting.

These trends in continuous casting are going to transform the steel industry, once again, completely. The overall, high-volume–low-profit-margin steel industry looks very much for improved efficiency and quality of steel production. By the year 2000 continuous casting would replace the ingot casting in almost all the major steel plants.

The continuous casting process has been independently developed by several groups in various countries such as Russia, Germany, the UK, the USA, Austria, etc. The principal designs now available the world over are by Concast, Mannesmann, Demag, Olsson, Hitachi, Voest, Koppers, etc.

28.2. Principle

Continuous casting may be defined as teeming of liquid metal in a short mold with a false bottom through which partially solidified ingot is continuously withdrawn at the same rate at which metal is poured in the mold. The equipment for continuous casting of steel consists of:

1. The ladle to hold steel for teeming.
2. The tundish to closely regulate the flow of steel in the mold.
3. The mold to allow adequate solidification of the product.
4. The withdrawal rolls to pull out the ingot continuously from the mold.
5. The cooling sprays to solidify the ingot completely.
6. The bending and/or cutting devices to obtain handlable lengths of the product.
7. The auxiliary electrical and/or mechanical gears to help run the machine smoothly.

The mold is open at both ends and is water cooled. The operation is started by fixing a dummy plug-bar to temporarily close the bottom of the mold. Steel is slowly poured in the mold *via* a tundish and as soon as the mold is full to a certain level withdrawal of the plug begins. The rate of withdrawal must exactly match with the rate of pouring for smooth operation of the machine. Uninterrupted pouring and simultaneous withdrawal gives rise to the whole cast being poured in the form of one piece which may be cut into smaller pieces as per the requirement.

In order to expedite the process ingot does not completely solidify in the mold. As soon as a sufficiently thick skin, which would be able to stand the pressure of liquid core is formed, the withdrawing from the mold commences. It is then cooled by secondary cooling. The situation in the mold is somewhat as shown in Figure 28.1.

Small area of the ingot, where the liquid core is able to press the solid skin against the mold walls, maintains a sort of *seal* to prevent liquid steel from leaking from the mold. This acts as a moving seal if the bar is withdrawn slowly from the mold and an equivalent amount of liquid steel is poured in.

If the bar is withdrawn rapidly, this seal may fracture and may produce cracks in the ingot or even *leak-outs.* Both of these eventualities can be eliminated and the casting speed can be increased if a moving mold is adopted rather than a stationary mold.

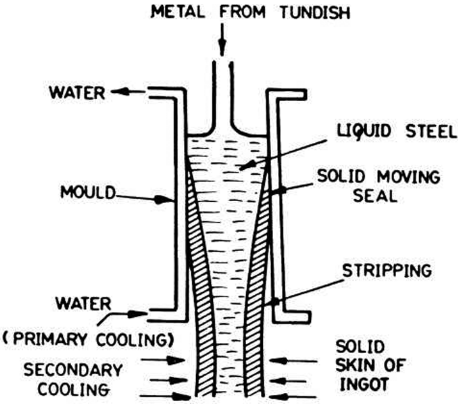


FIGURE 28.1. Solidification characteristics of continuously cast ingot.

The principle of moving the mold is known as *Junghan's Principle* so named after the investigator. In this mold is moved up and down variously, through a stroke of 3–10 mm, depending upon the characteristics of steel being cast. The speed of the downward to upward stroke is generally dissimilar, and the ratio may vary in the range of 25%–50%. In the earlier machines this was 1:3. The downward speed is more than the speed of withdrawal which leads to "negative stripping" of the ingot from the mold. If the downward speed is even slightly less than that of the rate of withdrawal major surface cracks are formed. The negative stripping is beneficial in the following ways:

1. The initially crystallized skin of the ingot is further compacted.
2. Formation of tensile stresses is prevented and even compressive stresses may be developed in the initially solidified skin.
3. It practically eliminates the possibility of transverse cracking of the ingot skin.
4. Transverse cracks that may be formed earlier are liable to be welded again.
5. It allows maximum rate of withdrawal, i.e. maximum production from a given machine.

Continuous casting operation is adjusted with respect to the following variables:

1. Chemistry of the steel being cast
2. Temperature of the steel at teeming (all through from beginning till the end, degree of super heat)
3. Number of strands cast in parallel and simultaneously
4. Mold size (volume) and dimensions
5. Mold stroke
6. Casting speed (*i.e.*, rate of withdrawal (determines the productivity))
7. Frequency of mold stroke (per min)
8. Negative strip time
9. Positive strip time

10. Total cycle time
11. Negative strip ratio
12. Casting powder viscosity
13. Powder consumption
14. Thermal conductivity of the steel being cast

All these factors have to be adjusted to produce relatively defect-free product and without any break-outs.

Except in small capacity plants, steel is continuously cast sequentially, wherein one after the other ladleful of liquid steel is cast continuously to improve upon the yield as well as the efficiency. The above factors are taken into account to carry out this continuous-continuous casting or "sequential casting".

28.3. Types of Continuous Casting Machines

Three main types of machines are in use in practice, namely (*i*) the vertical-type, (*ii*) the vertical-mold and horizontal-discharge-type, and (*iii*) the curved mold (S-type).

28.3.1. Vertical Type

It is the first continuous casting system wherein the mold and the discharge are both vertical. The general scheme is shown in Figure 28.2. Liquid steel is brought to the machine in a stopper controlled ladle and is teemed in a stopper controlled tundish which regulates the flow of steel to the mold. Below the mold is the secondary cooling zone in which rollers are set to make close contact with the ingot. The water spray nozzles are interspersed in between those rolls. The number of sprays, pressure of water, etc., are adjusted to control the degree of cooling. It is also known as a "roller apron." These hold the product fairly tightly to support it. The main withdrawal rolls are situated just below the roller apron. The cut-off torch travels at the same speed as that of the withdrawal by clamping the product. After cutting, the torch goes back to its position quickly. The product is then laid horizontal and is hoisted to the normal floor level.

If a large heat is to be cast into relatively small sections, it is essential to have more than one such strand to expedite casting. As many as 8-strand

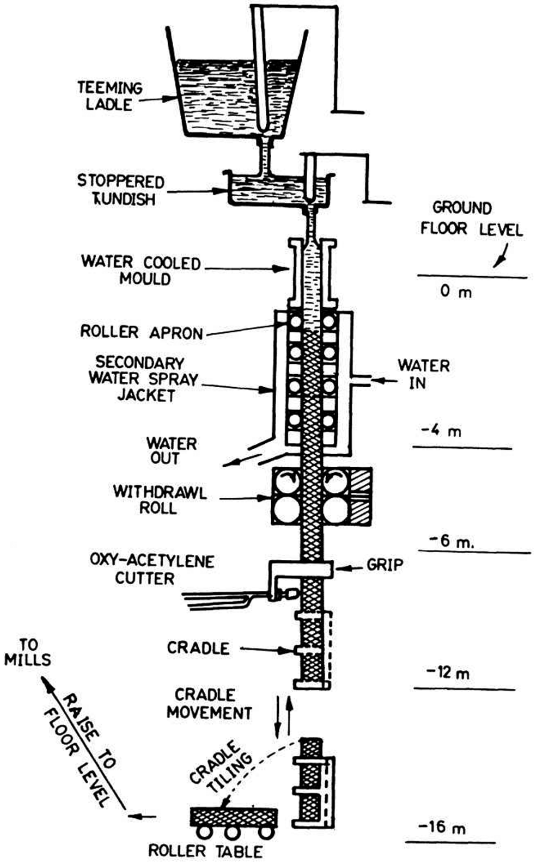


FIGURE 28.2. Vertical type continuous casting machine.

machines are in use. The tundish in this case divides the metal stream in that many equal parts.

This type of plant is very tall and hence needs either a tall shop or a large pit to accommodate the equipment. The problem is acute if high casting speeds are employed and, as a consequence, longer cooling zone is required. This type of plant is, therefore, used for large and medium sections. It is good for slabs wherein bending is avoided for its adverse metallurgical effects. In the event of a breakdown, it is easy to repair and restart the machine. It is most simple in construction and most reliable to operate. All steel qualities can be cast at high speeds without fear of damage to the stand by bending and straightening.

28.3.2. The Vertical-Mold Horizontal-Discharge Type

This is a modification over the earlier vertical design to reduce the overall height of the machine as shown in Figure 28.3. The mold, the roller apron design, and the pinch rolls are similar to those in a vertical machine. After the product emerges from the pinch rolls it is bent to obtain the discharge horizontal. The cutting torch moves horizontally. A horizontal set of straightening rolls becomes necessary. A savings of 30% in height is thus possible by this design. The floor space requirement is, however, more. Heavy sections being difficult to bend cannot be cast by

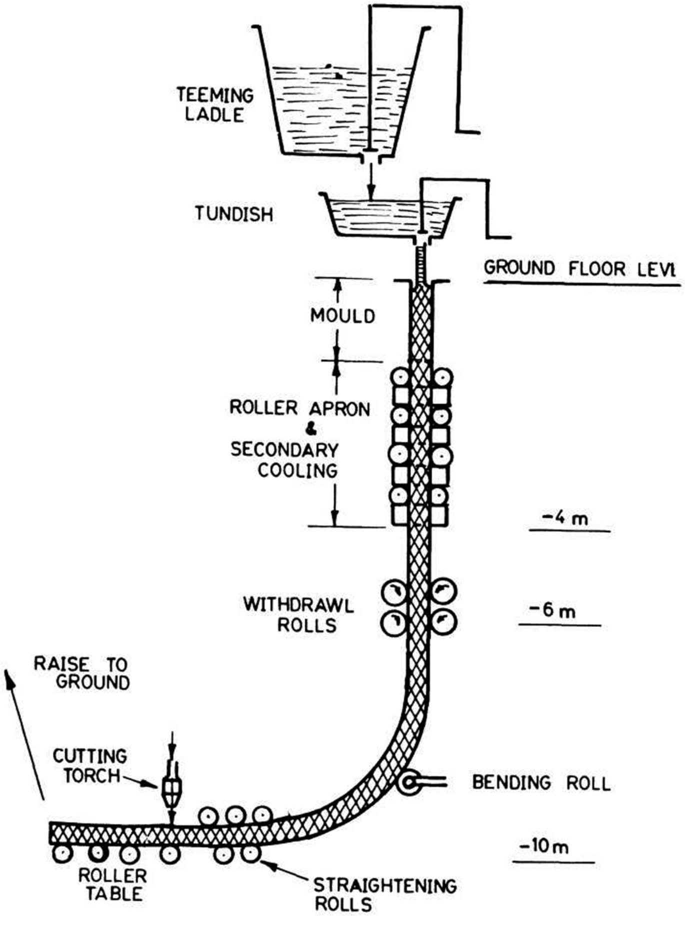


FIGURE 28.3. Vertical-mold-bend-discharge type continuous casting machine.

this machine. In the event of a breakdown, it is more difficult to repair and restart than the vertical machine. This was popular for small and medium size cross sections.

28.3.3. The Curved Mold (S-type)

This is the latest design now almost universally adopted for continuous casting of almost any section, such as, billets, blooms and slabs. The characteristics of this machine are:

(*i*) The mold is itself curved mold rather than straight one employed in the earlier two designs.

(*ii*) The strands come out of the mold in curvilinear fashion with a fixed radius.

(*iii*) It is bent before the entire cross-section is solidified.

(*iv*) The curved strand is in fact straightened after it is fully solidified and cooled to the designed extent.

The cross section of the machine in a simplified way is shown in Figure 28.4 to describe the details of the various parts of the machine along with their functions. The withdrawal rolls carry out bending as well and hence should be of adequate strength. The height of the shop in this case is minimum as compared to that of the other two designs and hence it is also called "low head" – type machine. The S-type machine is however more popular name. The radius of curvature of the strand should be as high as possible to help smooth operation. The modern designs are fairly safe to be run without any breakouts but in case it does then this design gives more problems than are faced in the other two types. All possible safety arrangements have been provided in modern machines, as will be obvious from Figure 28.4.

28.4. Essential Details of a Continuous Casting Machine

The basic requirements of any continuous casting machine are as follows:

(*i*) Hot metal handling system as a source of molten finished steel.

(*ii*) Tundish for supply and distribution of liquid steel to the mold.

(*iii*) Mold to freeze the skin of the casting.

(*iv*) Water sprays to complete solidification and required cooling.

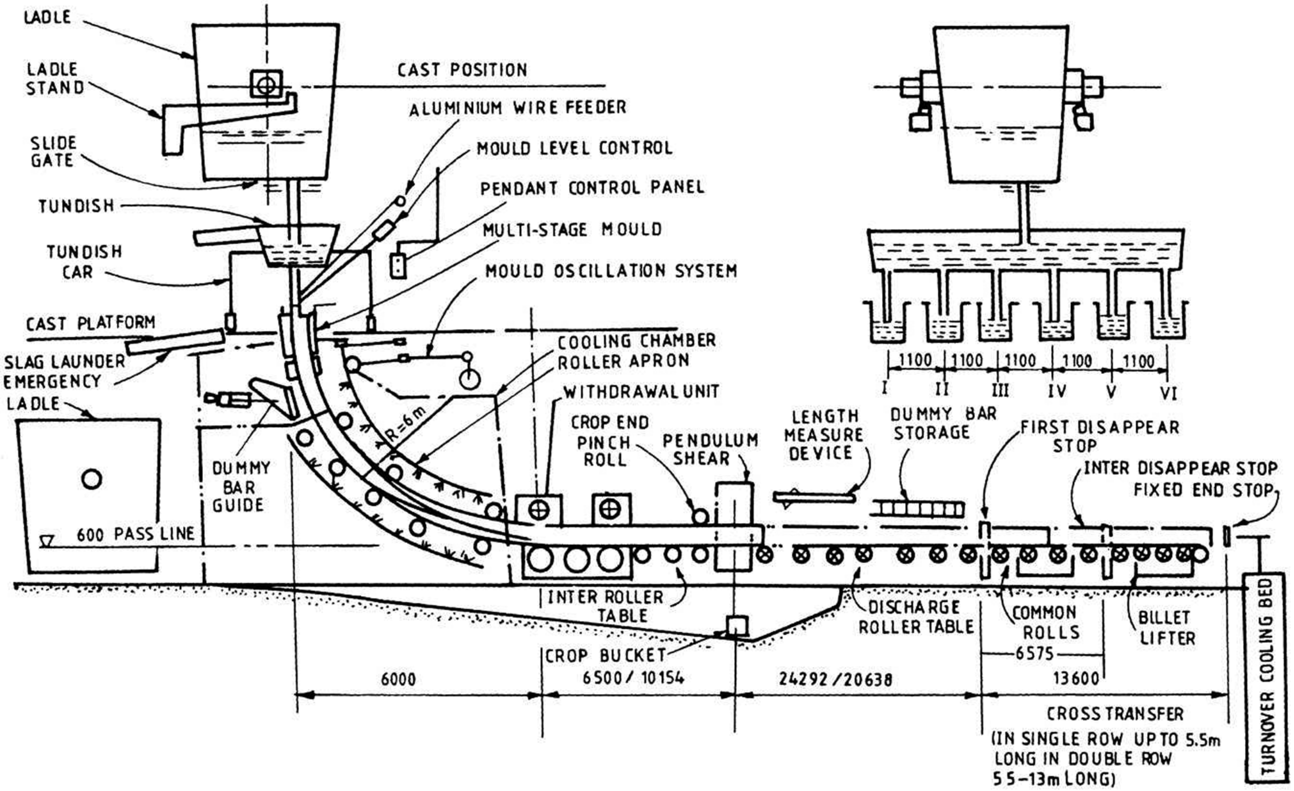


FIGURE 28.4. Schematic view of curved mold (S-type) continuous casting machine as adopted at Tata Steel[42].

(*v*) Drive system to withdraw the strand continuously at a predetermined rate.

(*vi*) Cut off machine to cut the continuously solidified piece into required lengths.

28.4.1. Molten Steel Handling System

Earlier the lip-poured or the stoppered teeming ladles were used for supplying liquid steel to the machine but now it is universally that bottom poured teeming ladles with slide gate system are used. The earlier machines were small and hence needed small capacity ladles but now with the use of multi-strand machines large capacity ladles are accepted without any difficulty. The earlier firebrick-lined ladles are being replaced by dolomite lined ladles to effectively control the dissolved oxygen content of the steel. The studies at Tata Steel have shown that the oxygen levels found were as follows:

dolomite lined ladles ----- 2–6 ppm oxygen
bauxite lined ladles ----- 2–6 ppm oxygen
silica lined ladles ----- 10–20 ppm oxygen

Usually steel is treated with calcium for deoxidation and desulfurization and alumina ladles in this case give problem of higher oxygen.

The ladle may be held from an overhead crane or may be suspended in a simple stand that is provided at the top of the machine. The latter one is preferred to free the crane for other work.

It is common practice these days to use LF for secondary refining. The same ladle with bottom gas purging acts as the teeming ladle, eliminating the need for unnecessary transfer of liquid steel from one to the other ladle. This LF is normally lined with carbonaceous magnesia bricks as are used for lining the main modern refining furnace. But if deep drawing steels with extra-low carbon contents are to be produced, then VOD is often adopted for the last leg of decarburization. In that case such low levels of carbon cannot be produced while using C-magnesia brick-lining of the ladle. In that case dolomite lining will have to be used for the ladle while processing under vacuum.

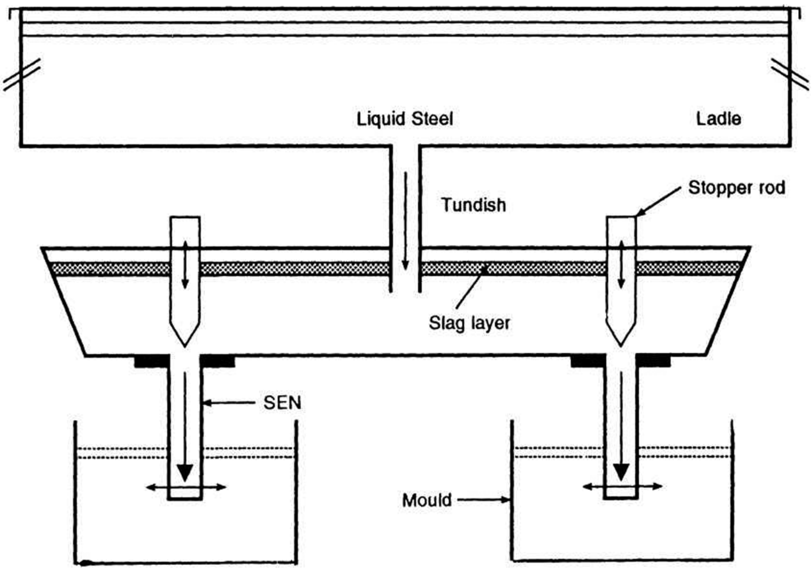


FIGURE 28.5. Modern tundish showing its location and use.

28.4.2. Tundish

It is necessary to teem steel from the ladle into the mold *via* tundish. It is not merely a distributor of steel into multiple streams or a buffer in between the mold and the ladle. It is beneficial in many ways.

Tundish is a must for sequential casting to maintain casting continuity. This is now a commonly adopted technology for further improving the yield.

Tundish is usually preheated before use to minimize heat losses from the liquid steel during the teeming period and to prevent steel from freezing in the nozzle area at the beginning.. Tundish covers are employed to reduce heat losses. A modern tundish is shown in Figure 28.5.

(*i*) The steel flow is controlled more accurately by maintaining the metal level the same during the entire casting period. This is not possible by using only a ladle, as the ferro-static head in ladle decreases with the progress of pouring and thereby alters the flow rate.

(*ii*) It can sub-divide the flow into several streams to simultaneously cast multiple strands and thereby finish casting in an acceptable time period.

(*iii*) A single, large mold can be teemed with a number of streams from the same tundish.

(*iv*) It can deflect the metal streams into an empty receptacle in the event of machine breakdown.

(*v*) Even if the slide gate of tundish fails, spare tundish can be used to complete the teeming.

(*vi*) The metal level in the tundish can be altered readily to effect adjustments in the rate of teeming.

(*vii*) The most important function that modern-day tundish is expected to perform is effecting cleaning of the steel in the tundish, that is, to produce better cleanliness. This also helps in removing the slag, if any, that has come along with the steel in the ladle. To effect removal of inclusions, the residence time of steel in tundish is increased by the following:

(*a*) Improving flow pattern

(*b*) Using suitable fluxes to absorb inclusions

(*c*) Using ceramic filters to trap inclusions

(*d*) Minimizing vortexing

(*viii*) Use of tundish allows final deoxidation adjustments, trimming additions, desulfurization, inclusion modification, and temperature control.

Single strand, multiple strand, "T"–type, Delta-type, V-shape, etc. are some of the commercially adopted tundish designs, each offering its specific characteristics. A proper tundish design, its lining, its maintenance, and so on, need to be carefully assessed to obtain the best possible metal flows to give a defect-free concast product.

There are two methods of tundish lining, viz., the conventional lining, which requires preheating, and the cold castable lining, which does not need preheating but needs insulating powder cover on the steel bath. A carefully designed and approximately lined tundish should last nearly 40–50 heats.

There are always ways and means to control the steel bath in tundish to obtain best possible results of the products. Additions such as aluminum wire can also be made in tundish for final control of steel chemistry.

28.4.3. Mold

The developments in molds have been the key factor in the development of continuous casting process. The mold is made out of drawn copper tube or machined out of a solid block or is a welded plate construction of high conductivity electrolytic grade copper as shown in Figure 28.6. The mold is water cooled. It has open bottom and hence is closed by a dummy plug bar in the beginning. The strand withdrawal begins by withdrawing this dummy bar. The length of the mold should be such that under the conditions of its cooling adequate skin must be formed before the ingot emerges out of it. It is nearly 75–140 cm in length. The mold is expected to extract 10% of the total extractable heat in the machine. To avoid transverse cracking it is oscillated at the rate of 30–60 oscillations per minute with a negative strip.

The primary functions of the mold are to contain and start the steel solidification, all through the teeming period, with the following aims:

1. To obtain enough shell thickness of solidified steel to stand bending stresses and ferro-static pressure of liquid steel from inside

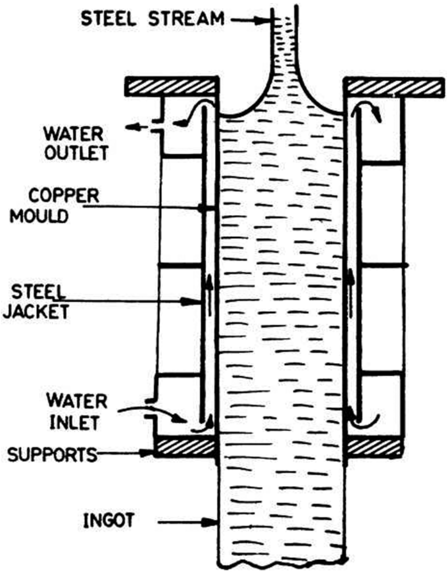


FIGURE 28.6. Mold of continuous casting machine.

2. To equalize temperature all through liquid steel mass

3. To ensure internal and surface quality of the product

Early vertical molds were simple to fabricate and use. The modern curved molds are more complicated to make and the correct cooling conditions are not easily obtained therein.

The modern molds are tapered to narrow down towards the bottom. This is to accommodate the shrinkage effect on solidification and yet remain as far as possible in contact with the steel surface and thereby effectively extract heat. This taper is not straight. It increases downwards because the steel temperature decreases downwards in the mold and the thermal coefficient of expansion does not vary linearly with temperature.

Molds are invariably lubricated to assist stripping. Moisture-free rapeseed oil is almost exclusively used because of its minimum smoke and flame. It also leaves no hard residue on the surface of the mold. For smooth operation a film of approximately 0.025 mm in thickness over the mold walls is essential. The oil is supplied continuously from a ring similar in shape to that of the cross-section of the mold, during casting. The oil consumption varies considerably with the practice in the range of 50–250 cc per metric ton of steel cast.

In the beginning, the mold cross sections were almost the same as those of the ingot molds used for slabs and the blooms. However, the sizes continuously decreased, and now very small cross sections are cast without any difficulties. This eliminates, to that extent, the role of primary mills in reducing the cross section and thereby improves the productivity and economy. This is how thin slab casting and billet casting have been developed and put into practice.

The chief problem put forward by the reduced cross sections is metallurgical in nature. Large cross sections undergo larger reduction by additional hot mechanical working and thereby there is certainty of breaking the original cast dendritic and the coarse grain structure developed earlier in the casting operation. The extra reduction is metallurgically very essential in many varieties of steels. The thin slab casting does not provide this much mill reduction and hence poses problems of inadequate breaking of the cast structure and thereby results in poor metallurgical properties in the end product. Therefore, thin slab casting cannot produce some quality steel products, although chemistry-wise there is no such limitation. Mold specifications for billet caster at Tata Steel are given below:

	Billet Caster
Mold size	801 mm long curved
Taper	0.4%/m for 100 mm^2 and 0.6%/m for 125 mm^2
Casting radius	6 m
Metallurgical length	16.2 m

28.4.4. Submerged Entry Nozzle (SEN)

In the early machines liquid steel was teemed in the concast mold in open air. It was found to give rise to various defects arising out of oxidation of a metal stream in the open atmosphere. This was avoided by enclosing the teeming area and introducing a neutral or reducing gas to shroud the stream to avoid contact with normal atmosphere. Further developments led to the use of "submerged entry nozzle," SEN for short. It is now commonly used. This is a ceramic tube which has a closed bottom and is connected to the tundish at one end, while the other end is dipped in the molten pool in the concast mold. It has openings on sideways, near the bottom, to let liquid steel flow out in the mold laterally. It is kept dipped in the liquid steel pool, all along, and the steel flows out more in horizontal direction as shown in Figure 27.7. It also shows the relative positioning of the tundish vis-à-vis the mold and their interconnection through SEN.

In slab casting the SEN has two openings in opposite direction towards the smaller opposite walls, and in blooms it may have four openings to distribute the steel as uniformly as possible over the entire cross section of the mold. Argon gas is introduced inside the SEN to prevent sticking of steel to the SEN surface from inside and to prevent aspiration of air through the tube pores and joints. Currently, mainly Al_2O_3-C is used as the base material for making SEN. But some do use fused silica with ZrO_2-C to make SEN.

The main problem of using SEN is that it gets clogged due to accumulation of alumina from inside, possibly due to a reaction such as SiO_2

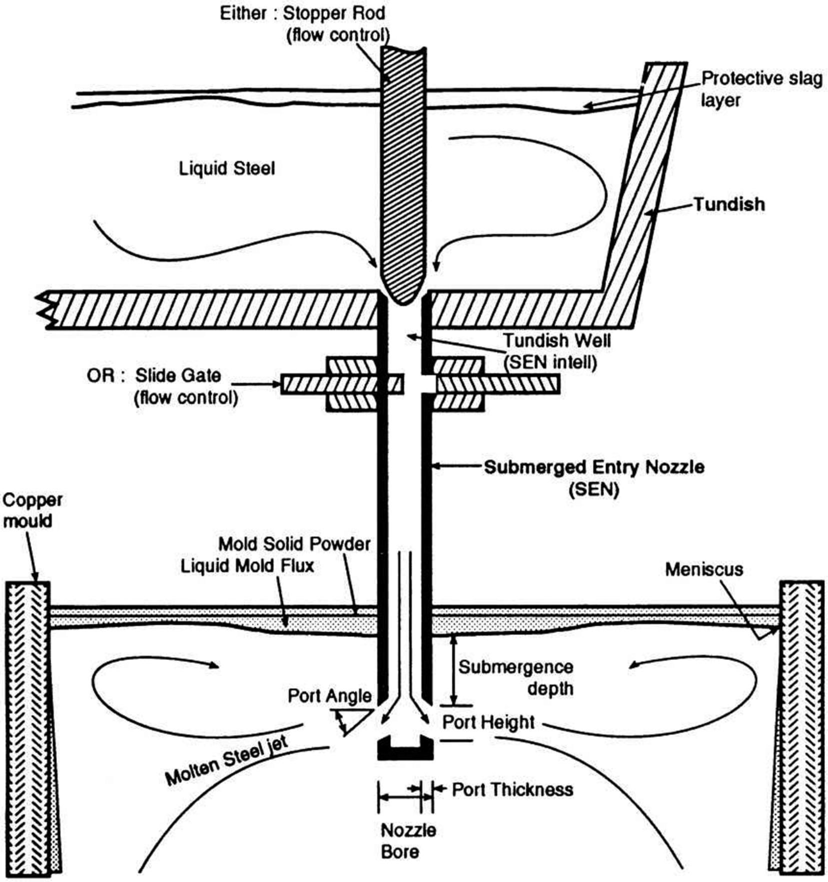


FIGURE 28.7 Schematic representation of the SEN while in use.[44]

(refractory) + 3C (refractory) + 4Al = $2Al_2O_3$ (deposit) + 3Si + 3C. This involves reduction of silica and oxidation of dissolved Al to alumina, which gets precipitated on the inner walls of the SEN. This is an ideal substrate for deposition of alumina inclusions invariably present in the melt due to prior Al-deoxidation. Once deposition occurs, it is easy for it to get sintered and thereby reduce the cross section of the SEN and consequently decrease the flow rate, causing problems in solidification of steel in concast mold.

28.4.5. Mold Lubrication

In the early days of continuous casting of steel, moisture-free rape-seed oil was almost exclusively used as the lubricant. It produced a minimum amount of smoke and flame. It also did not leave much hard residue on the surface of the mold. For smooth operation an oil-film of approximately 0.025 mm in thickness on the mold wall was considered essential. The oil was continuously supplied from a circular ring similar in shape to that of the cross section of the mold. The oil consumption was in the range of 50–250 cc/t of steel cast.

Now mold fluxes have been developed and used to have smooth casting operation. Mold fluxes are mixtures of different components. It is added as powder in the mold over the liquid steel surface. It melts in contact with the liquid steel surface in the mold and provides a safety cover that also prevents atmospheric oxidation. The functions of mold flux are:

1. Lubrication of the strand
2. Transfer heat from strand to the mold wall
3. Thermally insulate the top steel surface to minimize heat losses
4. Protect liquid steel from atmospheric oxidation
5. Absorb non-metallic inclusions rising to the surface

The mold fluxes used may have the following composition:

CaO = 25%–45%, Na_2O = 1%–20%, BaO = 0%–10%, SiO_2 = 20%–50%, K_2O = 0%–5%, Li_2O = 0%–4%,

Al_2O_3 = 0%–10%, FeO = 0%–5%, B_2O_3 = 0%–10%, TiO_2 = 0%–5%, MgO = 0%–10%, F = 4%–10%,

C = 1%–25%, MnO = 0%–10%

It only means that a wide variety of chemical compositions of fluxes are used. Carbon is added to control the melting rate of the powder only.

The flux powder is supplied in various forms as:

1. Fly-ash based powders
2. Synthetic mixtures
3. Pre-fused and ground powders
4. Pre-fused and granulated powders

The granulated powders are specifically suitable for automatic feeding. It is also dust-free.

The most important property of interest of powders is mainly its viscosity when it melts on the steel surface in the mold. It must have a corresponding liquidus temperature. Its properties should not substantially alter even on absorbing the alumina inclusions.

It is fed in the mold in powdered form continuously and automatically. It should therefore melt at the point of contact with liquid steel surface and may remain in powdered form at the top level. The presence of carbon controls its melting rate, and accordingly the feed rate is adjusted. The molten layer of flux gets solidified at the mold wall in the form of a ring. The flux gets carried over with the meniscus as it moves down and forms a thin protective coating on the strand surface as it moves down. This layer acts as the lubricant between the strand and the mold surface. The powder consumption should be in the range of approximately 0.3–0.6 kg/m^2 of cast surface of steel. The efficiency of the caster is assessed in terms of this consumption. It does vary with casting speed.

28.4.6. Secondary Cooling

Having accomplished the initial cooling (primary) in the mold the solidification of the remaining core and further cooling (secondary) of the solid product is accomplished by quenching using high pressure water sprays. The sprays are directed from all sides on the casting. The total length of the secondary cooling system is generally of the order of 50 times the minimum strand dimension (*e.g.*, for 100 mm^2 billet, it will be nearly 6 m). If the minimum dimension is large, say in slabs, the ratio may be less than fifty. Nearly 90% of the total heat to be extracted in casting is extracted in the secondary zone. The secondary cooling has to be precise to obtain a defect free product.

As the ingot emerges from the mold it is positioned with the help of several guide rolls held horizontally from all sides of the ingot. This is known as “roller apron” and the spray-nozzles are situated suitably in between these rollers. The roller apron supports the ingot and maintains its shape, prevents bulging and helps bending without causing cracking. The spray design in curved-mold is far too critical because the ingot has to be bent continuously along the curvature of the machine. Although the design varies, in general flat sprays are preferred to any other shape. The water sprays are controlled independently to effect the required cooling at that level of the spray. The quenching severity increases downwards, and as a result the liquid core forms a conical shape with its apex downwards.

Dirty water can cause blockage in the nozzle and thereby uneven cooling. The secondary cooling efficiency depends on:

(*i*) Cooling water velocity

(*ii*) Inlet temperature of cooling water

(*iii*) Direction of spray

(*iv*) Water quality and scale formation

The cooling water specifications are also laid strictly and followed.

The product must be fully solid, be at correct temperature, and be defect-free when it emerges from the secondary cooling zone.

The rate of withdrawal from the mold decides the production rate of the machine and which in turn is decided by the rate of heat extraction particularly in the secondary cooling zone. The efficiency of secondary cooling is therefore of paramount importance. In the early machine, direct water sprays were used. Now these are not only direct sprays but instead of using only water, a mist of water and air in suitable proportion is used, as it has been found to be more efficient in extracting the heat.

28.4.7. Withdrawal Rolls

These are one or two pairs of rolls meant to finally grip the ingot and pull it out at a pre-fixed rate without deforming the product. The pressure exerted by the rolls on the product should neither be excessive to cause its deformation nor less to allow slipping. It must be precise to effect the correct rate of withdrawal and thereby obtain the correct temperature profile in the product in the direction of withdrawal.

28.4.8. The Dummy Bar

To begin with, the bottom of the mold is temporarily closed by what is known as a dummy bar. It is meant to lead the product through the roller apron and withdrawal rolls. In the early designs it was similar in cross-section to that of the product but now for big sections it is much smaller than the cross-section of the product. The head of the dummy bar is supposed to nearly close the bottom of the mold and whatever gap is left is packed with asbestos to obtain a temporarily fully closed bottom of the mold. A certain amount of solid scrap is put at the bottom in the beginning to hasten solidification and form a solid moving seal against leakouts. The head of the bar has a bolt or a loop around which the metal solidifies and thereby ensures a good grip for pulling the cast bar. The head has to be prepared anew for each pour.

The length of the dummy bar in a vertical machine is equal to the vertical distance from the mold bottom to more than the cut-off-point. In a bending machine it extends vertically below the bending rolls. In a curved mold machine the plug bar is flexible.

During withdrawal as the head of the dummy bar crosses the cut-off point in a vertical machine, bending rolls in a bending machine and cut-off point in a curved mold machine it is disengaged and kept ready for the next heat.

28.4.9. Solidification

Solidification of steel, no matter where it occurs, takes the same course. However the cross sectional shape and the cooling severity results in differential resulting structures. The steel in a continuous casting machine is only partially solidified in the mold. In this condition it is pulled out and hence the casting has solid rim with liquid steel core. In some machines it is bent in this condition before complete solidification. It solidifies fully in the secondary zone of cooling. This process of solidification is highly dynamic in nature[45]. This still leads to the formation of initial fine grained-rim-structure and subsequently, though small in size, dendrites as well. It is therefore prone to segregation as well[46].

The micro-segregation in concast product gets eliminated significantly due to subsequent hot working. Macro-segregation however persists, resulting in quality problems. Micro-segregation is a consequence of rejection of solutes by the solid into the inter-dendritic liquid whereas macro-segregation results from movements of micro-segregated regions over macroscopic distances due to the motion of liquid and free crystals. In general, the more the cross section of the concast is decreased, the more serious this problem will be. This can be obviated to a large extent by hot-working. This is the reason why large cross sections are still preferred to small sections, in spite of the obvious better economics of producing small cross sections.

28.5. Operation

The machine is kept ready by fixing the dummy bar in position. The teeming ladle is brought into position and the temperature of the metal is measured. The teeming ladle must be preheated to the required temperature before tapping steel in it. Teeming commences when the temperature of steel falls to the required level. The tundish must be pre-heated to the required level by this time. The metal is allowed to collect to a certain level in the tundish before commencing pouring in the mold. As the level of steel in the mold rises to a certain prefixed level the reciprocation of mold begins and immediately thereafter the dummy bar is withdrawn slowly. The secondary cooling sprays are put on as the dummy bar moves down. The lubricating oil is turned on immediately after the teeming commences.

The speed of withdrawal should be quickly adjusted to the predetermined speed for the particular steel being cast. Pouring should continue steadily until the ladle is empty.

28.6. Output

The output of casting varies with the cross section being cast and the number of strands being used simultaneously. Present-day machines can cast a product as small as 50 mm^2 billet or as high as 2 m × 0.3 m size slab. The following data illustrate the rate of production from a single strand:

Product Size (meters)	*Casting Speed (meters/ min)*	*Product-metric ton/hr (approx.)*
0.050 × 0.050	5–6	7
0.100 × 0.100	2.5–3	12
0.150 × 0.150	1.5–2	16
0.200 × 0.200	0.9–1.2	21
1.250 × 1.150	0.8–1.2	80
1.500 × 0.200	0.7–0.8	105
1.750 × 0.250	0.5–0.7	130
2.000 × 0.300	0.4–0.5	140

The values in the last column can be treated as the ladle size to be adopted for single strand machines.

Assuming the above speed of casting, usual pouring duration (one hour), usual time of preparing the machine (one hour) and other delays the annual production from a single strand would be approximately:

Size in Meters	*Metric Tons/Year*
0.05 × 0.05	30,000
0.02 × 0.2	90,000
2.0 × 0.25	5,00,000

It can also be seen that for a eight strand machine casting billets of 0.05 m2 should have at best a ladle size of 50 metric tons and no more. Similarly for a 0.15 m_2 bloom a ladle of 80-t capacity will be used on a four strand machine and so on. The size of the product and the number of strands on the machine will have to be worked out on the basis of furnace size employed for steel making. Longer casting times than approximately an hour require much higher initial temperature of steel that has all the attendant disadvantages during steel making as well as casting.

28.6.1. Yield

The slab, bloom, or billet yields from liquid steel is much higher in continuous casting than in conventional method of casting. The two have been compared below:

	Conventional Practice (% approx.)	*Continuous Casting (% approx.)*	*Gain (%)*
Liquid steel	100	100	–
Ingots	96	–	4
Slabs	85	95	10
Blooms	83	96	13
Billets	81	96	15

In the case of killed steels this advantage is even more evident because the loss from hot-top discard is absent in continuous casting. The data are:

	Conventional Practice (% approx.)	*Continuous Casting (% approx.)*	*Gain (%)*
Liquid steel	100	100	–
Ingots	95	–	5
Slabs	84	95	11
Blooms	76	95	19
Billets	71	96	25

There should, therefore, be no hesitation in adopting continuous casting for killed steels if it is otherwise justifiable in terms of investment. Production of billets of ordinary-quality steels on a small scale is also quite economical by coupling an electric arc furnace with a continuous casting machine. For small-scale production of slab and blooms, the gains are not very much to warrant installation of a continuous casting machine. It may be adopted if very narrow range of slab and bloom sizes are to be made.

28.6.2. Process Control

The process variables in concast operation are:

1. Mold stroke (linear movement of mold), in the range 5–10 mm
2. Frequency of the stroke, which is approximately 150–200/min
3. Casting speed, normally in the range of 1–7 m/min
4. Negative strip time, in the range of 0.1–0.2 sec
5. Total cycle time of the order of 0.3–0.4 sec
6. Positive strip time of approximately 0.15–0.3% of cycle time
7. Oscillation depth of nearly 0.11–0.18 mm
8. Mold oscillation of 0.015 to max 1.0
9. Casting powder viscosity of approximately 0.10–0.20 Pas
10. Powder consumption in the range of 0.17–0.40 kg/m^2 of strand area
11. Negative strip ratio (*i.e.*, % negative strip-time of the total cycle time)
12. ARCCOS Number in the range of -1 to $+1$

The casting speed is a function of heat extraction efficiency, which in turn depends on the size of the cross section as well. All of the above variables are dictated by the steel chemistry, its thermal conductivity, yield strength, quality requirements, and so on.

28.7. Metallurgical Aspects

The continuous casting machine is supposed to operate under designed conditions of operation. Although there are innumerable variables, some of the important variables are as follows:

There is a definite tapping temperature of steel for each of the variety to be cast continuously in a given machine of given number of strands. This temperature is fixed with respect to the chemistry of steel, rate of casting, number of strands, and so on. Every attempt is made to stick to this in operation.

The rate of cooling in the mold and in the secondary sprays is to be determined for each variety of steel in a given machine and is to be strictly adhered to.

A continuously cast product has similar internal structure to that of usual conventionally cast ingot. Any difference is due to the different sizes of the products only and due to the faster rate of cooling in concast product. The concast product has a fine grained shell at the surface followed by equiaxed grains in the center. All the possible defects that can arise in conventionally cast ingots can also occur in concast product and are discussed earlier in Chapter 25.

28.7.1. Defects in Concast Products

The common defects in concast product are listed below:

1. Alumina streaks
2. Edge cracks
3. Longitudinal cracks
4. Primary scales
5. Roll marks, roll peel-off, rolled in scale, roll wear
6. Slivers
7. Random surface defects arising due to fluctuations of the surface level "telescopeity" and stagger
8. Variation in width (curly edges)
9. Non-uniform thickness and flatness (ripples in both direction)
10. Non-uniform micro-structure in either direction
11. Segregation
12. Oscillation marks—the slag gets entrapped in this, its consumption increases, and the mark depth decreases with casting speed
13. Slag particles sticking the surface—entrapment of lubricant slag particles at the surface

28.7.2. Various Types of Cracks

By far formation of cracks is one of the most common and yet avoidable defect in concast product. There are wide varieties of such cracks that may occur as shown in Figure 28.8.

Formation of cracks is due to the pressure of tensile stresses exceeding the yield stress at that spot. Such stresses may arise due to a variety of reasons related to cooling, stripping, bending, withdrawal rate and tension, surface finish of the roller apron and withdrawal rolls, mold taper, mold wall surface finish, mold lubrication, reoxidation of steel in open air, and so on. More importantly they are related to the hot-ductile behavior of the steel being cast. The operation must therefore be carried out more and more rigorously under controlled conditions to ensure relatively defect-free product.

Concast product is more susceptible to hydrogen pick-up because of use of lubricating oil in the mold. This can cause formation of pin holes readily. The presence of excess aluminum can prevent this to a large extent. Prior vacuum treatment may also be beneficial. Cleanliness can be improved by carrying out metallurgical operations even in tundish and preventing reoxidation by shrouding the metal stream in the mold.

The quality of a concast product is generally better than that of the conventionally cast ingots. It is reported to have favorable grain size, less of segregation and more homogeneous chemical composition perhaps due to

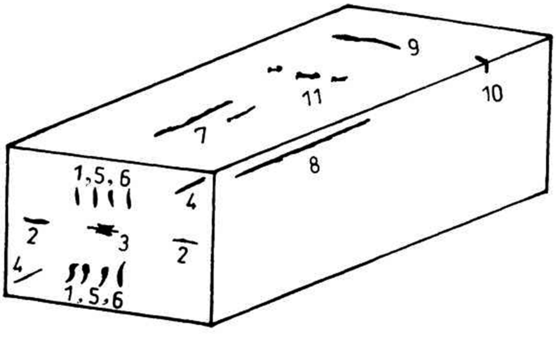


FIGURE 28.8. Scheme of different types of cracks that are likely to form in a continuously cast product. 1–6 are internal cracks:
1 – Midway, 2 – Triple-point, 3 – Centerline, 4 – Diagonal, 5 – Straightening/bending, 6 – pinch roll.
Nos. 7–11 are surface cracks: 7 – longitudinal mid-face, 8 – longitudinal corner, 9 – Transverse mid-face, 10 – transverse corner, 11 – star.

rapid cooling. This has further been improved by some modern techniques. The use of electromagnetic stirring is one such technique.

28.7.3. Defects Arising from Lubrication

A thin slag cover is always maintained on the open surface of steel in the mold to prevent its oxidation, which otherwise lead to additional defects. In the early days, lubricating oil was used as a lubricant for smooth withdrawal of the ingot. It often led to hydrogen pick up from the hydrocarbon of the oil leading possibly to formation of pin holes. This was somewhat offset by using aluminum to a large extent. This, however, led to the development and use of flux-powder to replace the use of lubricant oil. This melts and forms a slag cover on the surface of the liquid steel and lubricates the mold wall as shown in Figure 28.9.

The process of continuous casting is quite complex and is shown is Figure 28.9. The incoming liquid steel comes in the direction as shown by the jet, and it affects the fluid flow characteristics of solidifying steel in a very complex way. This turbulence, caused by the incoming jet of liquid steel, can float inclusions to join the slag or create fresh inclusions, even by shearing the slag cover at the top, depending upon the precise conditions prevailing in the mold. Quality problems are therefore directly associated with the flow patterns in the mold. The mold stroke creates transverse ripples on the cast surface and wherein this slag gets entrapped, causing other defects. On the whole, surface defects, such as longitudinal cracks and star cracks, have been associated with variations in slag-lubrication.

The lubricating slag flows into the gap between the mold wall and the steel strand surface because of the contraction of ingot cross section on solidification inside the mold. The slag gets solidified because of the chilling effect of the mold and forms a ring which sticks more to the mold wall. A gap is created between the mold wall and the strand (face of solidified steel) because of contraction of steel on solidification. The rate of heat extraction, which is the most vital parameter for cooling, is decided by this gap. The rate of withdrawal has to match the heat extraction so decided. The roller pressure also has to be in tune with this cooling rate. Any excessive pressure can lead to breakouts.

Variations in practice with respect to sudden changes in casting speed, SEN gate position, argon flow, non-uniform flow due to clogging, etc., can lead to defects in the final product.

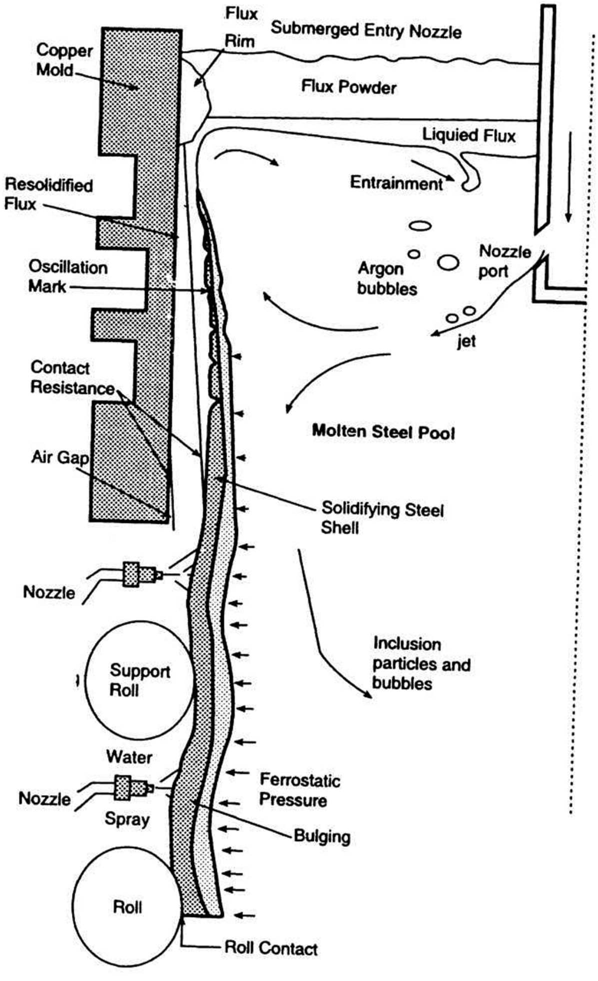


FIGURE 28.9. Schematic of phenomena in the mold region during casting[47].

The stroke often leads to horizontal striations, if not actual cracks, on the surface of the solidified steel, and slag gets in to these gaps and solidifies.

Cracks may form in relation to temperature, stress, dendrite and grain micro-structure, phase transformations, segregation, etc.

High-meniscus heat transfer and variation in meniscus heat transfer mechanisms correlate with increased sliver defects, but the reasons are not understood. Hence, improved understanding of slag layer behavior is important to improve product quality.

During each oscillation stroke, liquid slag is pumped from the meniscus into the gap between the strand surface and the mold wall, where it acts as a lubricant, so long as it is liquid. A solid slag layer forms against the mold wall. It is much thicker there and is called slag-rim. The slag in due course does solidify and causes friction and drag. The solidified slag also interferes with the heat transfer problem.

Friction experienced in the operation may be due mold/slag contraction resulting in the moving slag layer, excessive taper of the mold, misalignment, or a combination of the three.

28.7.4. Defects Arising from Deoxidation Practice

Early concast operation posed many serious problems in casting very low carbon steels. These were cast as rimming or semi-killed steels in traditional ingot molds. It was not possible to cast them in the same way in concast machine purely because of an inability of concast machine to control the rimming action. Concast required fully killed steels. Many earlier steels were killed with Si and Mn and were considered adequate for an ingot mold-casting route. Concast generally requires Al-killed steels so that there is no problem of any C and O reacting during cooling to generate CO gas and its attendant problems. Now Al is therefore generally used for deoxidation before using a concast machine. However, it poses several problems.

Al deoxidation produces solid Al_2O_3 as the product of deoxidation. These particles do not coagulate readily and remains suspended in the bath. These particles stick to the SEN during the teeming in concast machine and interfere with the smooth teeming operation by altering the nozzle cross section. The automatic concast operation becomes erratic. At times it has to be discontinued. The SEN replacement becomes necessary. On the whole the cost goes up. Besides it also creates alumina streaks as defects in the final product. Any excess Al addition aggravates the problem considerably. The process of deoxidation must therefore be carried out under very controlled conditions, and that is often rather difficult. It has been shown that if residual Al dissolved in steel is kept less then this problem can be controlled[46] to a large extent. It, however, still cannot eliminate the dirtiness of steel because of alumina inclusions. This problem has been solved by calcium treatment of the bath just before casting, wherein the alumina inclusions get modified and most of these problems are minimized.

28.7.5. Calcium Treatment

Calcium has been found to obviate most the problems caused by aluminum deoxidation. After deoxidation is through, sufficient time is allowed for the deoxidation product to rise and join the slag at the top of the bath. Then calcium is injected in the form of either Ca metal or an alloy of Ca-Fe-Al with low or high Si content and again some time is allowed for inclusions to rise. It is generally introduced in the form of a cored steel wire. This treatment modifies the morphology and even the chemistry of the inclusions such that they are more prone to coagulate and thereby rise and join the slag layer at the top of the melt. The details of this treatment are given under Section 28.10 and in Figure 28.3.

28.7.6. Electromagnetic Stirring

The electromagnetic stirring (EMS) is generally installed around the mold or the sub-mold regions approximately 1.3–3.7 m below the meniscus level. The structure gets modified due to the altered flow patterns in the liquid metal pool in the mold or below it in the mold core as long as it is molten. The structure becomes less dependent on variables such as superheat, casting speed, carbon content, size, and so on. It decreases the length of columnar grains, increases the area of equiaxed grains and reduces the center line segregation.

The induced motion in liquid metal sweeps away the inclusions from the solidified shell, improves sub-surface cleanliness, and prevents pin hole and blow hole formation. On the whole it gives a more uniform shell and lower sensitivity to cracking.

28.7.7. Prevention of Oxidation During Casting

Steel is always prone to reoxidation if it comes in contact with oxidizing media during teeming from ladle to tundish and from tundish to mold or even in ladle or in tundish itself.

The carryover of refining slag from steel making furnace into the teeming ladle is a potential source of oxygen, and it must be minimized.

These methods to minimize the carryover slag are discussed in Chapter 23.

Even after minimizing the carryover slag, the liquid steel in the teeming ladle and tundish is to be protected against oxygen pick-up. Similarly, the metal stream from ladle to tundish and tundish to mold has to be protected from atmospheric oxygen. This is avoided in various ways as follows:

1. Use of protective cover on liquid metal in tundish and in ladle
2. Use of protective atmosphere when it flows from tundish to mold
3. Protective cover to metal stream

The tundish nozzle is often extended to dip in the usual metal level in tundish so that the stream does not come in contact with atmospheric oxygen at all.

Alternatively, the stream is shrouded by an inert gas by enclosing the flowing stream in a bellow-like enclosure which is provided with an inlet and an outlet for the inert gas.

A metal pool in tundish is covered with a powder mixture containing essentially CaO, SiO_2, Al_2O_3, and CaF_2 such that it softens and covers the surface of liquid steel and thereby prevents atmospheric contact.

Powder mixtures are also used in molds to cover the meniscus and provide a thin lubricating film for facilitating withdrawal from the mold. This also helps in collecting the deoxidation products and thereby prevent steel from being dirty.

As and when steel is exposed to atmospheric air, it picks up oxygen as well as nitrogen from the air. This is minimized by the use of shrouding inert gas and in modern times by using SEN and flux cover. This is already explained above.

28.7.8. Watercooling

Cooling is effected in concast operation in two broad stages, viz. in the mold to the extent of approximately 10% and the remaining 90% in the spray cooling zone. In concast nearly 0.8 × 106 kcal/m^2/hr heat is extracted to cool the liquid steel, and this figure can be as high as 40 × 106 kcal/m^2/hr at the meniscus level. A very thin oxide formation on the mold wall can be disastrous. An equivalent amount of water needs to be circulated through the apron cooler.

28.7.9. Control Sensors

It is impossible to produce sound casting unless the rate of feed of liquid steel is absolutely correctly matched by the rate of withdrawal. The meniscus level therefore has to be correctly controlled. This is done either by controlling the tundish flow by a stopper rod or slide gate valve or, by withdrawal of speed control. This needs continuous level sensing in the mold. These are:

1. Sensor using radioactive radiation
2. Electromagnetic sensor
3. Eddy current mold level indicator

28.7.10. Cut-Off Device

The strand comes out of the mold and the secondary cooling system in the form of a long continuous piece. It is cut into desired lengths either by an oxy-acetylene torch or by mechanical shear of adequate capacity. The cutter travels with the same speed as that of the withdrawal so that the actual cutting operation does not come in the way of continuous production. As soon as the cut is through, the cutter returns back to the original position for the next cut.

28.7.11. Air Cooling of Castings

The individual cut pieces of the castings are transferred to the roller-bed normalizing yard, one by one, by pushers, and are air-cooled on these beds. The pieces can also be transferred to a soaking furnace in red-hot condition for further rolling without cooling to room temperature.

In the early designs the cut pieces of concast ingot were transferred on to roller-bed normalizing yard, one by one, by pushers, and were air cooled on these beds. They were sold as-is. Alternatively, these pieces were transferred in red-hot condition to soaking furnaces and from where they were rolled, one by one. In a continuous strip plant, the pieces pass through a hearth furnace and get rolled, one by one, without the temperature coming below the hot-rolling temperature.

28.8. Recent Trends in Continuous Casting

The already-existing old machines are being modernized by providing the facilities not existing in the machines, (*e.g.*, extended tundish nozzle or shrouding the stream with inert gas, electromagnetic stirring if possible, meniscus level sensor, last minute tundish metallurgy such as aluminum wire addition, other automatic control devices, and so on).

28.8.1. Continuous-Continuous Casting

The productivity and yield of a continuous casting process can further be improved if casting is made in sequence of more number of ladles without a break in between two ladles. This is often carried up to nearly 7–8 ladles of casting in a row, now popularly known as “continuous-continuous casting”. This requires synchronizing of that many number of heats in the steel making furnace in such a way that the next ladleful of steel is ready for teeming before the earlier ladleful steel is fully teemed and is empty. The operational and maintenance problems may not allow it to proceed too far beyond this 7–8 number of ladles in one go.

28.8.2. Combination Caster

It is a bit difficult to continuously cast rounds that are needed for the production of seamless tubes, but it is uneconomical to install a continuous caster solely for producing rounds because the demand for rounds is not that high in proportion. There are plants where this demand goes up to 40% of the total concast product. In that case a combination-caster has been designed to produce round sections as well as square or rectangular cross sections from the same machine. This involves considerable changes in the machine to change over one from the other. If the proportion of rounds is much less than 40%, then it is not worthwhile to go in for a combination caster. The changeover require changing of molds, roller apron, cooling sprays, withdrawal rolls, etc. The narrower the size ranges in square and rounds, the easier to make the changeover.

28.8.3. Near-Net Shape Casting

Having developed the concast technology to fair perfection, it is logical to expect that the technology would be further developed to produce the net or the near-net shape, finally required, continuously, starting from liquid steel without any intermediate break or discontinuity. The trends in continuous casting in the last couple of decades are shown in Figure 28.10. Two such technologies have emerged as practicable. The first is the thin slab casting in which the slab size has been reduced to approximately 60 mm thick. This needs only further reduction down 60 mm to obtain the flat product. Similarly, billets of the size 100 × 100 mm cross section have been successfully continuously cast on a commercial level.

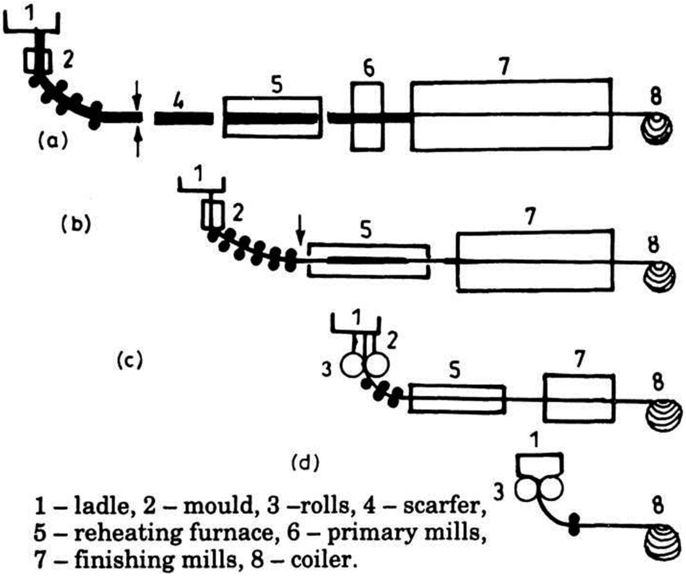


FIGURE 28.10. Progressive changes that are occurring in a continuous casting machine design (only schematic):
(*a*) Is a standard curved mold curvilinear withdrawal
(*b*) Relatively thinner slab caster
(*c*) Future near-net shape caster

This development has led to the development of CSP technology. This "compact strip production" technology has been fully exploited in designing steel plants wherein the continuous caster is synchronized with a rolling mill to produce continuously the desired flat product, without intermediate heating in soaking furnace or the like. This process is ideally suited for narrow range of flat products. This is best suited for finally producing 1.6-mm-thick sheets after completion of rolling. Further reduction has to be carried out by cold rolling.

The latest continuous casting technology is to cast final thickness of sheet or plate required directly from liquid steel. In this process, liquid steel is poured between two horizontal rolls separated at the appropriate distance and rotating in the opposite direction to produce the final net shape of the product. This eliminates the need of any rolling and size reduction, except perhaps the final cold roll-pass. This is bound to improve economy considerably. The structure of the product is claimed to be very fine grained and very uniform all through (*i.e.*, quite isotropic). The products can be compared as shown in Table 28.1. This has been commercialized as

Table 28.1
Comparison of the processes of producing thin steel strip[48]

Parameter	*Caststrip*	*Thin Slab*	*Thick Slab*
Cast thickness (mm)	1.6	50	220
Casting speed (m/min)	80	6	2
Ave Mold Flux (m.wt/m^2)	14	2.5	10
Total solidification time (sec)	0.15	45	1070
Ave Shell cooling rate (°C/sec)	1700	50	12

Cast-Strip-Technology and the first such commercial plant has been successfully put up by NUCOR in the USA in 2002. Additional plants are under construction.

This technology is useful to cast 0.7–2.0-mm-thick sheets. However, the productivity is on the lower side as compared with the thin slab technology.

28.9. Prospects of Continuous Casting

In 1980, approximately 30% of the crude steel was processed by continuous casting all over the world. This figure rose to 65% in 1990. In South Korea, more than 95% of steel is continuously cast. This figure is equally as large in Japan. By the year 2000, 98% of the crude steel in major integrated steel plants would be continuously cast in India.

Now liquid steel is continuously cast in all the major steel making plants. If it is not, it is an exception due to some very special and specific reasons or due to far-too-low production rates.

Since its inception, a major breakthrough occurred in the seventies, and since then its adoption has been on the ascendancy. The whole process of liquid steel processing up to the final semis is bound to be a non-stop continuous operation in the beginning of the next millennia.

CHAPTER 29

COMMENTARY ON PRACTICAL STEEL MAKING

In This Chapter

29.1. Introduction

Steel in large bulk quantities is basically produced by two broadly different processes. In one a BOF of some variety is used for primary conversion of molten iron to steel and in the other a suitable mixture of scrap, DRI, and even hot metal is incorporated to convert it into steel in an electric furnace. This difference is necessary in view of the difference in the charge quality in these two cases. These are the primary processes of steel making. The difference exists only up to this level of processing. Hereafter, further processing as secondary steel making may even be exactly the same in both the cases. The teeming and casting has nothing to do with how and by which route the liquid steel is produced. The steel making routes already have been shown schematically in Figure 20.1.

If a blast furnace is adopted for ironmaking and hot metal is to be predominantly converted to steel, then adoption of any of the primary BOF process is the only alternative because of its relatively better economics as compared to the electric furnace process. When a solid charge such as DRI is to be treated, predominantly as charge, then the electric furnace is the only alternative, as the primary process of steel making, in spite of the fact that blast furnace may also be used to make up part of the charge.

The choice of secondary processing of steel is a very wide one, and it depends on the type, the quality, and the chemistry of the product to be produced. Here again the choice is getting more and more narrow with the passage of time. In a modern steel plant using either BOF or the EAF, both are invariably provided with bottom purging of argon to stir the slag and metal system to achieve more closely the equilibrium conditions of refining. This has almost become a must now. Similarly, use of a transfer ladle as LF, preferably with an arc heating facility, is the most minimum secondary steel making facility that is adopted almost universally in integrated steel plants. Only small-size steel plants using arc or induction furnace as a primary steel making unit, and producing only the ordinary constructional variety of steels, may manage steel making only with primary units, for producing very cheap steels. For any quality steel product or even for achieving reasonably good productivity levels, in relation to the capacity of the primary unit, a secondary set-up is a must. A very common secondary facility is the LF being used as a transfer ladle for steel right up to the caster for teeming.

Many times, a vacuum treatment plant is provided to cater to obtain desired lower gas contents, better alloying, and cleanliness. This facility is often provided to cater even for some specific non-routine heats, which are asked by some customers occasionally. Here again the choice is getting down, more and more to the adoption of R-H process of degassing for its better performance, efficiency, and working conditions. It is relatively cheaper to install and adopt tank degassing for occasional use. Many plants have, in fact, provided tank degassing from this point of view only.

Whatever are the facilities provided for steel making for every new grade of heat to be made, for the first time, it should be designed beforehand. There is a science, technology, and engineering involved in designing such a heat of different grade of steel not made earlier in the plant. For example, a medium- or high-carbon grade is to be produced for the first time the heat must be designed on a paper before actually processing it in practice.

29.2. Designing Steel Making Heat

29.2.1. Product Classes

From the point of view of steel making, the classification of steels has already been given on the first page of this book. On the shop floor for practical classification of various grades of steels made, they are often referred to as:

1. Constructional steels—which are usually cast as blooms, and qualitywise they have moderate cleanliness levels.
2. Ultra-low-carbon steels—which usually have carbon less than 30–40 ppm and are usually cast as slabs (sometimes thicker are better and otherwise thin slabs are preferred), and the cleanliness requirement is very stringent.
3. Line-pipe steels—which have high strength and high fracture toughness and have low impurity levels and inclusions.
4. Engineering steels—which are heat-treatable and usually low-alloy grades and are often cast as billets at high casting speeds. The plate and sheet forms are cast as slabs.

29.2.2. Heat Design Considerations (Operating Practices)

For making any steel grade for the first time, the steel making process that is necessary needs to be designed beforehand. With the available knowledge of steel making it is now possible to design such an operation in advance, based on experience and taking into account the theoretical principles underlying steel making. The basic approach involved in this exercise is discussed here.

The design involves fixation of the following major variables:

(*a*) The charge variety, individual weights of charges, and their chemistry

(*b*) Type and extent of impurities to be removed during primary refining

(*c*) Assessment of basicity of the slag needed if P is to be removed

(*d*) The amount of flux addition, its rate of addition, and time of addition

(*e*) Additions during the heat, to produce the desired slag

(*f*) The liquidus temperature of the slag expected to be formed or to choose the slag type as dictated by the furnace operating temperature

(*g*) The oxidizing/reducing ability of slag to be aimed at in terms of the iron oxide content

(*h*) Operating temperature or its range considered useful for refining

(*i*) Oxidizing medium and its rate of feed, that is, the pressure and rate of oxygen flow and its mode of introduction

(*j*) Rate of oxidation of impurities expected under the operating conditions

"Take care of the slag and the steel shall take care of itself," goes the saying. This must be remembered while designing the slag for the heat. The slag aimed at should be thin enough at the furnace working temperature. For this, the knowledge of the liquidus temperature range is necessary. This can be assessed from relevant oxide equilibrium diagrams. Normally the slag has more than half a dozen important constituents. Equilibrium diagrams of oxides for a five-or-more-component system are not available. Ternary systems are readily available. The proposed slag composition with more than three important components then needs to be reduced to a pseudo-ternary oxide system. For this the basic oxides are added together to term it as base content. For example, CaO, MgO, MnO, etc., are put together, either as summation of their weight percentages or by summating their equivalent weights to work out their effective percentage. Similarly, the acid oxides are added together likewise. The amphoteric oxides such as Al_2O_3 or Cr_2O_3 may not be added with either of these. The third dominant constituent shall be either Al_2O_3 or FeO or such oxide as the case may be. Such pseudo-ternary diagram can give the liquidus temperature of interest.

Since oxygen gas is invariably used for refining, the rate and pressure of oxygen blowing need to be ascertained, taking into account its efficiency of oxidation and the total impurities to be removed, in a given steel making setup. As indicated in the previous section, the rate of decarburization is to be worked out to control the refining, to achieve the desired end chemistry.

The initial flux addition and subsequent rate of its addition has to be worked out to finally get the total required addition to make the desired slag. The slag must be close to the type aimed for at the end, all through the heat. This is the reason why the flux is added as per requirement during the heat.

The proportion of refining to be carried out in the primary refining vessel and the remaining in one or more of the secondary steel making processes that are adopted has to be designed beforehand.

Similarly, the time and stage of desulfurization, if required, has to be suitably fixed. The slag composition required for this needs to be worked out. How it is to be exactly achieved must be worked out beforehand in terms of its economy and effectiveness.

The deoxidation practice necessary, in view of the final restrictions on the contents of Si, Mn, and Al, has to be worked out to get the right chemistry at the end. Even vacuum decarburization or deoxidation has to be considered and adopted, if necessary, in this context. The efficiency of individual deoxidizers must be known to evaluate their actual additions for effective deoxidation to the required extent.

The cleanliness of steel required has to be juxtaposed against the available techniques such as Ca injection, vacuum deoxidation, bottom purging, and so on, to finally get the desired cleanliness.

The alloy additions must be worked out, taking into account the efficiency of alloying, in each individual case of alloy additions. Since these generally contain carbon, the problem of carbon increase, on account of ferro-alloy additions must be borne in mind while refining carbon earlier. Any failure on this front is bound to lead to the use of costly low-carbon ferro-alloys to meet out the carbon specifications.

Finally, the tapping temperature of steel needs to be adjusted, to suit the subsequent process of casting. Final reheating in LF with arc or chemical heating by aluminum may be necessary in this connection. Any reheating of finished steel under arc, particularly after vacuum treatment, can lead to carbon pick-up of 20–30 ppm.

It is necessary to take into account the quality of raw materials in working out the above-mentioned details to produce the desired quality of steel from the given raw materials by the most economical route.

29.3 Primary Refining

Primary refining is designed with respect to the total impurities to be oxidized, that is, the percentages of C, Si, P, and Mn in the charge-mix to be removed. If DRI is used, then the gangue content of the DRI is to be assimilated in slag while maintaining the basicity. It will also need to fix the end chemistry so that remaining refining could be carried out in the secondary treatments. In BOFs all the impurities are aimed to be removed down to the lowest value, in one go, except perhaps the sulfur content. In modern EAF practice using hot metal in the charge, primary refining is carried out in two stages. The hot metal is refined as in BOF to the extent possible to remove P, Mn, and Si and carbon is partially left to be removed during the DRI melting period to take advantage of using it as a reducing agent for iron oxide in the DRI. Deoxidation is carried out mainly during tapping and never inside the BOF vessel, to obviate any chance of P-reversal from slag. In EAF practice, partial deoxidation is possible inside the arc furnace, if required.

The important point in this design comes when making high-alloy steels such as stainless steel, when it is necessary to decide as to when, what type, and how much of Fe-Cr is to be added to achieve the desired alloying. This becomes critical because the bath is to be decarburized in the presence of Cr. The refining media used is generally oxygen, but it is necessary to use an inert gas and oxygen mixture for stainless steel making, and it becomes necessary to design the composition of this mixture for the entire refining period, to achieve the desired decarburization rate in preference to the oxidation rate of chromium.

29.4 Deoxidation and Secondary Refining

At the end of primary refining, the steel bath generally contains 400–1000 ppm as dissolved oxygen. It may even be higher on account of faulty practice. The value of 600–800 ppm as dissolved oxygen is indicative of good primary refining. Getting the sulfur specification level is the worry of secondary refining. If the sulfur levels are rather high in the hot metal, then it may be worthwhile to adopt external desulfurization of hot metal before processing it in the primary steel making process. If this level is not high, that is, in the range of 0.02% –0.05%, some 20% of this may be removed in the primary process as a result of highly basic slag employed over there and this may be considered adequate for some varieties of steels. But if further desulfurization is required, then it is to be carried out in the secondary unit for some specified steels. This then becomes the chief activity of LF processing for such steel production. For this, the bath has to be deoxidized to a level of 2–4 ppm dissolved oxygen. Al is the only useful and cheaper deoxidizer that can give such a low level of dissolved oxygen. Deoxidation design must also prescribe the end Al content in the bath. The relevant equilibrium is shown in Fig 29.1.

RDCIS, Ranchi has developed economical practices of deoxidation for different steels. Use of coke powder as preliminary deoxidizer helped in significant saving of costly deoxidizers such as Al to the extent of 0.5–1.0 kg/t.[48]

Desulfurization of Si-killed, Al-free (Al<0.01%) steel is difficult and only limited extent of desulfurization is possible. A modified desulfurization slag such as 54% CaO, 26% Al_2O_3, and 20% SiO_2 has been shown to achieve 45%–80% desulfurization[50]. Hence, Al-killing followed by desulfurization using synthetic slag is the usual practice that is routinely adopted.

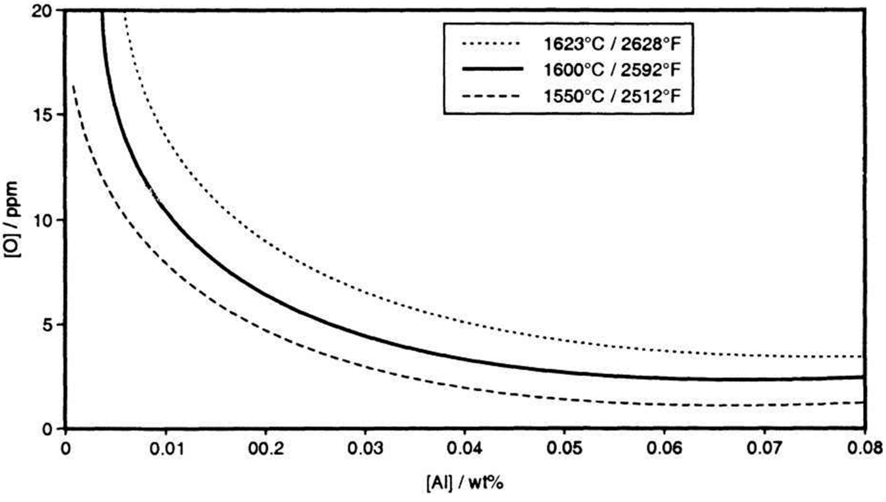


FIGURE 29.1. Al-O equilibrium at unit activity of Al_2O_3 at various temperatures.

How to work out the Al requirement for such a process:

Let us assume the heat wt as 100 t

Final Al content desired is 0.04%

Initial oxygen level is assumed to be 1000 ppm

Hence, Al required for reaction with oxygen is

$$= (54/48) \times \%O$$
$$= (54/48) \times 0.1000\ \% = 0.1125\%$$

Al required for end Al in steel = 0.0400%

Total Al needs is = 0.1425%

For a 100 t heat total Al required is = 142.5 kg

Assuming efficiency of Al utilization as 50%, Al required

= 285 kg

Wt of Al_2O_3 generated is (285 − 40) kg Al × (102/54)

= 846 kg

The LF is supposed to contain less than 2% of (FeO+MnO) at this stage under the reducing conditions achieved by Al deoxidation.

How and in what form is this total Al to be added?

A major part is usually added during tapping in the form of blocks and cubes, and a minor part may be added in the form of Al wire later in the LF before desulfurization.

29.5. Problem of Carry-Over of Slag

In spite of all care, a certain amount of oxidizing slag from the primary steel making furnace does get entrained and carried over, along with the liquid steel, into the LF, during tapping. There are “floats” designed to prevent slag being carried over, but they are not fool-proof to stop slag getting carried over along with the steel during tapping. This slag always contains some SiO_2, besides some FeO and MnO. Aluminum is consumed unnecessarily in reducing the FeO and MnO thus present, during deoxidation. Their content reduces to 2% total, during the deoxidation. The consumption of Al for this purpose is a waste and a drain on cost of deoxidation. If the refining slag gets carried over into the LF, there is no alternative but to stand this loss. Similarly, when deep deoxidation is carried out by Al, the SiO_2 in the LF slag, whether from the carry-over slag or from any other source, tends to get reduced as

$$3(SiO_2) + 4\underline{Al} = 2(Al_2O_3) + 3\underline{Si}$$

because of the shift in equilibrium of Si/O in the direction of more silicon dissolved in the bath. When silicon specification is very low, this creates problems in steel making. The residual Al in steel therefore needs to be controlled carefully to minimize silicon reversion in this way. The lower the carry-over slag, the better it is in this regard.

The slag carry-over problem is also serious when the blast furnace slag gets carried over along with the hot metal and enters the primary steel making furnace. This B.F. slag is primarily accounted for as hot metal when it is weighed. It is bound to create error in hot metal weight charged. Secondly, it contains sufficient SiO_2 that would require additional lime in order to maintain the required basicity during steel making. Thirdly, since it contains sulfur there is a danger of sulfur reversion from this slag to steel, under the oxidizing conditions of steel making.

29.5.1 Slag Weights

Slag that is generated in any process of iron and steel making is never weighed on a routine basis the world over. The weight of slag generated in any such operation can however be calculated. For this the slag analysis and the known added weight of some slag-forming constituent such as CaO or MgO must be known accurately. In steel making, the charge generally does not contain any CaO or MgO. This is added in a known amount during the steel making operation as flux to carry out the process smoothly. The analysis of flux is known. It means the actual amount of CaO or MgO added in a given operation is known. This weight is responsible to give the %CaO or %MgO in the slag as observed in the analysis. Material balance can be established as follows:

$$\text{Slag Wt(t)} \times \%\text{CaO}/\%\text{MgO} = \text{wt of flux added(t)} \times \%\text{CaO}/\%\text{MgO}$$

Out of the above, three variables are known and hence the fourth, that is the slag weight, can be calculated.

29.6. Problem of Desulfurization

The problem of sulfur in steel making is related to the excess sulfur, over the specification level aimed at, considering its level in iron from which steel is to be made. There are alternative routes available to tackle this problem, which has been discussed earlier in Chapter 4. If the input sulfur level in primary charge is above the final specification level, then it becomes necessary to remove it in LF, which then becomes its main function. Synthetic slag is used to achieve necessary desulfurization in LF.

If LF is adopted, desulfurization may be carried out, partially or fully, in LF using synthetic slag. To achieve desulfurization in LF, the steel needs to be fully deoxidized either during tapping or in LF or by a combination practice to achieve dissolved oxygen levels below 5 ppm. This level of deoxidation can only be achieved cheaply by using Al as a deoxidizer. Si and or Mn are incapable of achieving this low level, since the equilibrium dissolved oxygen level in that case is only be 30–60 ppm. The standard practice for effective desulfurization in LF therefore requires prior deoxidation by Al only.

29.7. Tapping Temperature

Refined and finished liquid steel is invariably cast, these days, in a continuous casting machine. The prime requirement for an efficient continuous casting operation is the correct teeming temperature, which in turn depends on the correct tapping temperature. This is decided by the liquidus temperature of the steel, which primarily depends on its chemistry. Since there are more than half a dozen different elements present in steel, the actual liquidus temperature is empirically calculated from its chemistry as follows:

Liquidus temperature

= melting point of pure iron (1535) – effect of those which decrease this + effect of those which increase it

= $1535° - (65\ \%\ C) - (8\ \%Si) - (5\ \%\ Mn) - (30\%\ P) - (25\ \%\ S) - (1.7\ \%Al) - (5\ \%\ Cu) - (4\%\ Ni) - (1.5\ \%\ Cr) - (2\ \%\ V)$ (in degrees celcius)

University of Liverpool in their "Annual Virtual Steel Making" competition[14], gave the following formula:

Liquidus Temp (for <0.5%C) = $1537 - 73.1\%C - 4\%Mn - 14\%Si - 45\%\ S - 30\%P - 1.5\%\ Cr - 2.5\%\ Al - 3.5\%Ni - 4\%V - 5\%Mo$

and **Liquidus Temp** (for >0.5%C) = 1531 − 61.5%C − 4%Mn − 14%Si − 45% S − 30%P − 1.5% Cr − 2.5% Al − 3.5%Ni − 4%V − 5%Mo

A suitable formula can be evolved for one's own practice on these lines.

Lifting temperature from LF (°C) = Min Temp in Tundish + Temp Loss from LF to Tundish (with time) + Temp Loss during teeming

Usually, temperature is lost at the rate of nearly 0.5°–1.0°/min from a ladleful of liquid steel, with the passage of time. Similarly, temperature is lost from tundish during the actual teeming operation at the rate of 0.15°–0.20°/ min, with time.

Therefore, usual calculations assume that a drop of 30°–40°C may occur during the period of transfer of steel from LF to tundish, before the teeming actually begins, and a drop 10°–20°C may occur in tundish from the beginning of teeming to the end of teeming of a ladle.

29.7.1. Heating

Quite often steel is refined in LF followed by vacuum treatment for reducing dissolved gases or for lowering carbon content to very low levels. In this process, steel often loses temperature, as a result of heat losses to the surroundings. It then becomes essential to reheat the bath to the desired tapping/teeming temperature. This can be done by the following alternative routes:

1. Either an induction heated vacuum treatment process is adopted. But this is costly in view of the establishment cost involved. Some do use this route.
2. The second alternative that is commonly adopted is to take back the heat to the LF station for arc heating. This is problematic, particularly when deep drawing grades of very low carbon contents are processed since the heat is liable to pick up some 20–30 ppm carbon from the graphite electrode during the arc heating. However, it is possible to heat the bath at the rate of nearly 3°C per minute by this method of heating. It does involve ladle movement and loss of time, resulting in increasing the tap-to-tap time.
3. The third alternative is to adopt chemical heating using Al as the fuel, which has been described below separately.

29.8. Desulfurization in LF

The sulfur problem has been discussed in Chapter 4. Sulfur causes hot shortness in steel, that is, failure at high temperature. Sulfur is generally present in solid steel as MnS inclusions. During deformation, MnS results in cracks and zones of weakness. Mn addition avoids the formation of FeS, which is even more detrimental to hot workability and leads to severe cracking during hot rolling. Sulfur is found to reduce particularly the toughness, although it does reduce ductility, formability, weldability, and corrosion resistance as well. However, it improves machinability and hence steel-grades for better machinability are produced by resulfurization of the steel bath.

In LF, desulfurization can be effectively carried out under highly basic and reducing slag, made by the addition of synthetic slag as discussed above. In the presence of Al, the desulfurization can be considered to take place as

$$3(\mathrm{CaO}) + 3\underline{\mathrm{S}} + 2\,\underline{\mathrm{Al}} = 3(\mathrm{CaS}) + (\mathrm{Al_2O_3})$$

It is advisable to work out the extent of desulfurization and sulfur partition ratio D_S in a given practice.

$$\text{Desulphurisation required (\%)} = \{[\underline{\mathrm{S}}(\text{initial}) - \underline{\mathrm{S}}(\text{final})]/\underline{\mathrm{S}}(\text{initial})\} \times 100$$

If initial sulphur is 0.035% and final required S is 0.005 % then % Desulphurisation required is = 87.5 %

and Desulphurisation partition ratio = (% S)/%$\underline{\mathrm{S}}$ will be:

If the slag is to contain, say, 2.0% S at the end, then the partition ratio = (2.0/0.005) = 400

In a 100 t heat to desulfurize the bath from initial 0035% to 0005% and to have slag not to contain more than 2.0% S, the weight of slag will have to be minimum

$$= (0.035 - 0.005)/0.02 = 1.5\ \mathrm{t}$$

Hence, the synthetic slag to be charged in LF will be in the range of 1.5–2.0 t for effective desulfurization of steel heat of 100 t, from 0.035% to 0.005%. Calcined lime may also be used to react with Al_2O_3 formed by Al deoxidation to make up the composition similar to that of the synthetic slag. This economizes the operation to some extent. The low melting composition region, with 50% −55% CaO, 38% −40 % Al_2O_3, and 3% −5%

SiO_2 has a liquidus of less than 1400°C. It has a very high sulfide capacity. The slag composition in LF must be controlled within this range carefully. The relevant ternary diagram already has been shown, in Figure 3.2. The sulfide capacity of any composition of slag in this range is very high and hence the desulfurizing ratio, (%S)/%S of the order of anything between 500–1000 is possible to be achieved. Extra-low sulfur of the order of 0.0010% can also be achieved by this treatment. Sometimes a second similar slag may be required.

Thin liquid slag is a must for effective desulfurization or else the process shall be kinetically slow and, within the refining time available in LF, sufficient desulfurization may not take place. Alternatively, longer refining time will have to be allowed, to achieve the desired level of desulfurization, which is again uneconomical.

The use of synthetic slag helps to prevent nitrogen pick-up.

Use of synthetic slag needs stirring for effective slag-metal contact. This is achieved by using bottom purging of argon invariably. In the absence of this, the desulfurization efficiency of synthetic slag will go down considerably.

29.9. Alloying Additions

Ferro-alloys of different grades and carbon contents are used for deoxidation and alloying to obtain the necessary steel chemistry at the best possible economy. This has been covered in Section 5.5. Here their % recovery, place where they are best added, and so on, are shown in Table 29.1

29.9.1. Quality of Ferro-Alloys

The ferro-alloys, besides containing the useful alloying element, do contain some undesirable elements as well. These contaminate the steel and create problems when present at unacceptable levels. Some examples can be cited to illustrate the point. In ball-bearing steel, Ti must be below 30 ppm. This brings restrictions on Ti content of Fe-Cr used for making this steel. Similarly, only 0.5% Al in Fe-Si is permissible to achieve <0.007% Al in a steel bath to avoid clogging during casting. For line-pipe steel with a max P of 0.012%, costly low-phos Fe-Mn is required. When nitrogen max-level is prescribed, the ferro-alloys need to be low in nitrogen. The undesirable elements in ferro-alloys are shown in Table 29.2.

Table 29.1.
Type, grade, use, and recovery of various ferro-alloys in steel making

Type of Ferro-alloys	*Grade*	*% Alloying Content*		*% Recovery*		*% C*	*Place of Addition*
				Tap/LF	*VD*		
LC-FeMn (Gr 1,2)	80–85	80	Mn	98	100	0.5	Tap/LF/VD
LC-FeMn (Gr 3)	70–75	70	Mn	98	100	0.5	Tap/LF/VD
MC-FeMn	70–75	70	Mn	98	100	1.5	Tap/LF
HC-FeMn	70–75	70	Mn	98	100	7.0	Tap/LF
SiMn	60–65	60	Mn	98	100	2.0	Tap/LF/VD
FeSi(Ord)	70–75	70	Si	62	90 min	0.2	Tap/LF/VD
FeSi(low Al)	70–75	70	Si	62	90 min	0.2	Tap/LF/VD
LC-FeCr	70–75	70	Cr	98	98	0.1	Tap/LF/VD
HC-FeCr	60–65	60	Cr	98	98	7.0	Tap/LF/VD
FeNb	62–65	62	Nb	100	100	0.2	Tap/LF/VD
Fe-V(Gr 1)	50–55	50	V	100	100	0.2	Before Ca treatment
Fe-V(Gr 2)	80–85	80	V	100	100	0.2	Before Ca treatment
Fe-Ti	35–40	35	Ti	70	75	0.1	Before Ca treatment
Fe-P	22–66	23	P	100	100	0.1	Tap/LF/VD
Fe-B	14–18	14	B	65–70	70–75	0.5	Before Ca treatment
Fe-Mo	62–65	62	Mo	100	100	0.1	Tap/LF
CPC	99.5 min	99.5	C	80	-	99.5	Tap/LF/VD
Cu	99.0 min	99	Cu	100	100	-	Tap/LF
Ni	99.0 min	99	Ni	100	100	-	Tap/LF/VD

Table 29.2.
The level of undesirable elements in ferro-alloys[51]

Undesirable Elements	*Ferro-alloy Sources*
Ti	MC-Fe-Cr ~ 0.09% –0.12% HC–Fe-Cr ~ 0.06%
N	HC FeMn ~ 100 ppm, LC FeMn ~ 495 ppm Si-Mn ~ 1 ppm
P	Si-Mn ~ 0.25%–0.35%, HC FeMn ~ 0.25%–0.35% Extra Low P-LC-FeMn ~ 0.16% –0.20%
C	Si-Mn ~ 0.30%, HC FeMn ~ 6.0% –8.0% Low P-LC-FeMn ~ 0.10%
Al	FeSi ~ 1.0% –1.5%, Low-Al-FeSi ~ 0.50%

29.9.2. Alloy Addition Calculation

Elemental addition

In the simplest case when pure element is added the amount of the element added, M_{add} is given by:

$$M_{add} = \{(\%M_{aim} - \%M_{present}) \times \text{bath wt}\}/(100\%)$$

Suppose a 100-t bath heat contains 0.1% Mn and the aim Mn content is 0.7%, then the amount of Mn added is

$$Mn_{added} = \{(0.7 - 0.1) \times 100 \times 1000 \text{ kg}\}/100\% = 600 \text{ kg of Mn}$$

Alloy addition

If High C FeMn of 75% Mn is added in this case to achieve the final 0.7% Mn then, assuming typical Mn recovery of 90%, the alloy addition will be

$$\begin{aligned}\text{HC-FeMn added} &= (100\% \times (0.7 - 0.1) \times 100 \times 1000 \text{ kg})/[75\% \times 90\%] \\ &= 888.8 \text{ kg}\end{aligned}$$

Impurity contribution

If this alloy contains some undesirable elements as impurities, the bath will be contaminated by this element(s).

Assuming that in the above example the Fe-Mn contains 500 ppm of nitrogen, then the bath shall pick up nitrogen as follows, assuming 70% assimilation of nitrogen:

$$\begin{aligned}\text{ppmN} &= \{\text{kgWt of alloy} \times 0.70 \times 500\ \text{ppm}\}/[100 \times 100 \times 1000\ \text{kg}] \\ &= 3.11\ \text{ppm}\end{aligned}$$

29.9.3 Cooling Effect of Ferro-Alloys

The deoxidizers and the alloying additions are invariably made in cold conditions. They have a cooling effect on the liquid steel bath, in spite of the fact that some additions such as Si, by virtue of its strong interaction with iron or with dissolved oxygen do generate some heat. As a rule of thumb, every kilogram of solid ferro-alloy added per ton of liquid steel would normally result in a drop of temperature by approximately 0.5°C–1.5°C. If the additions are much more, this cooling effect can be somewhat obviated by pre-heating the additions beforehand.

At times, classified steel scrap of known chemistry can also be used as an alloying addition to obtain the desired alloying during steel making.

29.9.4. Mode of Alloy Additions

Any alloy addition to molten steel, whether for deoxidation or for alloying, can be made either in the form of virgin metal or, simple or complex ferro-alloys. The tendency of the added element to oxidation in the given situation, the impurity contamination of the bath it may lead to, and re-carburization that may result because of its carbon content, to a large extent, decide the form in which it is to be added.

Similarly, the way or the mode of its addition also has to be decided. Alloying addition can be added as blocks, cubes, lumps, granules, powder, etc. At the same time, it can be added directly to the melt or via inert carrier gas or be fed in the form of cored steel-wire. Out of these, the use of carrier gas is only exceptional. Direct addition in the form of blocks or lumps is a commonly adopted mode of addition. The actual proportion of the added element used, for the purpose, indicates its recovery in the operation. For economic reasons, recovery must be high. Atmospheric oxidation of the added element decreases its recovery. The size of the lump is decided in terms of kinetics of its heating, melting, and dissolution. It is advisable to use that size that readily gets assimilated in the melt and gives high recovery.

Steel-cored-wire is nothing but a small, flexible steel-pipe, filled with the powdered or granular addition. It behaves like wire and hence the name.

It is available in the form of wire reels, filled with the required material of the right size and, using electric motor drive, can be fed with sufficient speed, usually vertically downwards into the melt. The steel wire-pipe melts inside the bath and releases the alloy for assimilation in the bath. It helps to inject the material well below the surface of the molten bath, whereby minimizing the atmospheric oxidation and helping better distribution and assimilation in the bath. This results in much better recovery. This is a very effective mode of alloy addition, but it is more costly than direct addition.

In cored mode, the alloy addition is in the form of powder or at best in fine granular form. The finer the material, the more is its surface area per unit weight. This increased area helps the kinetics of its dissolution in the bath. If the alloy addition is refractory in nature (such as Ti) it may have more surface area covered with its refractory oxide. This can be a hindrance if the addition is on a micro-level. Therefore, when a small addition of a refractory material is to be made, it is better made in lump form rather than in cored form.

Considering the mechanism and kinetics of alloy reaction in the melt, the form and mode of alloy addition has to be decided. Although cored wire addition is costly it may be preferred for other metallurgical advantages. When cost is no consideration, cored wire is generally preferred for its better recovery and effectiveness.

29.10. Production of Clean Steels/Inclusion Control

Except the very ordinary commercial grades, most other steels do insist on quality, particularly with respect to the cleanliness of steel (*i.e.*, steels with low non-metallic constituents). Inclusion formation is in a way unavoidable. But attempts should be made to minimize them. However, it is possible to remove inclusions in a cost-effective manner, only to a limited extent. The alternative strategy is to modify the inclusions, in terms of its chemistry or configuration, such that they become less harmful. These days, this process of inclusion modification is being called "inclusion engineering."

The general approach is to convert the inclusions into soft, deformable, and for this, preferably in globular form, so that they would not act as stress raisers during subsequent hot or cold deformation in rolling. The aim is also to get them in liquid form to help them coagulate and thereby get eliminated to finally produce better clean steel.

Deoxidation by Al, Mg, Ti, and Cr gives rise to solid oxide products, which are hard and un-deformable inclusions, which jeopardize a rolling operation. These also result in clogging of the nozzle (SEN) in concast machine and hence needs modification to minimize this trouble.

Somewhat cleaner steels are produced by a simple method such as bottom-stirring by argon. The bubbles of argon rise from the bottom of the ladle and pick up the floating non-metallic particles and are expected to be assimilated by the slag available at the top of the melt. This can achieve some cleanliness. The rate of bottom purging must be commensurate with the lifting of the inclusions[52]. Gentle bubbling helps decantation of inclusions. Excessive flow can lead to unnecessary oxidation of bath at the eye. In a 200-t ladle, up to 400L/min may be used for mixing and only 50–60 L/min may be used for decantation of inclusions.

Much better and more stringent requirements of cleanliness are met by inclusion modification obtainable through calcium injection[53,54]. Calcium is introduced in the form of a cored wire. It is either pure Ca metal or as Ca-Fe-Al alloy with 32% Ca and 40 %A1 in powder form, with high or low silicon contents, introduced as cored steel tube-wire. Calcium is very volatile at usual steel making temperatures. The solubility of calcium is approximately 165 ppm at one atmosphere pressure of calcium. The solubility is approximately only 0.025% at 1600°C. It must be injected as deep as possible in the steel melt to improve its efficiency. Its consumption is usually approximately 0.150 kg per t of steel treated, depending upon its form. Calcium recovery is generally approximately 15% –20%. Calcium changes the morphology of the inclusions, in particular, of sulfide and the alumina and alumino-silicate inclusions in general, produced by earlier Al deoxidation and desulfurization. Sulfides such as MnS are usually present in the form of streaks, before Ca treatment. Alumina as a deoxidation product is present in the form of fine un-coagulated, dispersed, and suspended particles. The silicates of Al_2O_3, MnO, and CaO are generally present as clusters. These are shown in Figure 28.2.

When calcium is introduced, it first reacts with sulfides and silicate inclusions to form complex sulfide-oxide, which is molten at steel making temperature.

The general chemical reaction describing modification of inclusions is:

$$x\mathrm{Ca} + (1 - 2/3\,x)\,\mathrm{Al_2O_3} = (\mathrm{CaO})_x\,(\mathrm{Al_2O_3})_{1-x} + 2/3\,x\,\mathrm{Al}$$

where x has a value between 0 and 1 and depends on the amount of Ca participating in the reaction. Full modification of inclusions needs

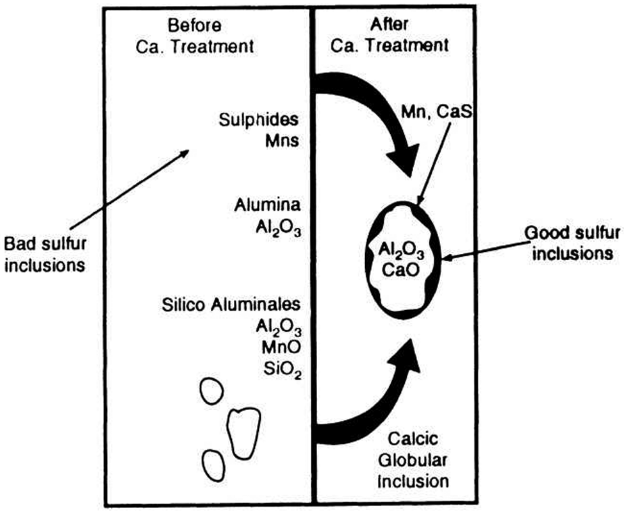


FIGURE 29.2. Depicts the physical and chemical nature of inclusions after aluminum deoxidation and before and after Ca treatment.

sufficient calcium. The inclusions are modified step-by-step and by assimilation of CaO in Al_2O_3 particles, progressively, as shown below[52]:

$$Al_2O_3 \text{ -> } CA_6 \text{ -> } CA_2 \text{ -> } CA \text{ -> } CA_{x(\text{liquid})}$$

This reaction, however, need not go to its logical completion in converting all the solid alumina into liquid alumina particles. This will take place only partially. Some will be transformed into liquid and the others will be semi-solid or even solid state. Calcium also reacts with the existing Mn-sulfide inclusions to form complex liquid sulfide inclusions. Silicon introduced along with the calcium can also combine with oxygen and can join the other oxide inclusions to form liquid phase inclusions. These liquid phase inclusions then surround the solid alumina and form inclusions with solid core and liquid periphery. On the whole, therefore, calcium modifies their morphology and transforms the solid inclusions of alumina into actual or apparent liquid particles. This mechanism is shown in Figure 29.3. In this process all or part of the suspended alumina particles are thus encircled by this liquid, and no pure alumina inclusions, as such, are usually left.

The size of the original alumina inclusion increases slightly when it gets enveloped by the liquid slag generated by calcium addition. It helps to move the particles upwards as per Stoke's Law. However, the important

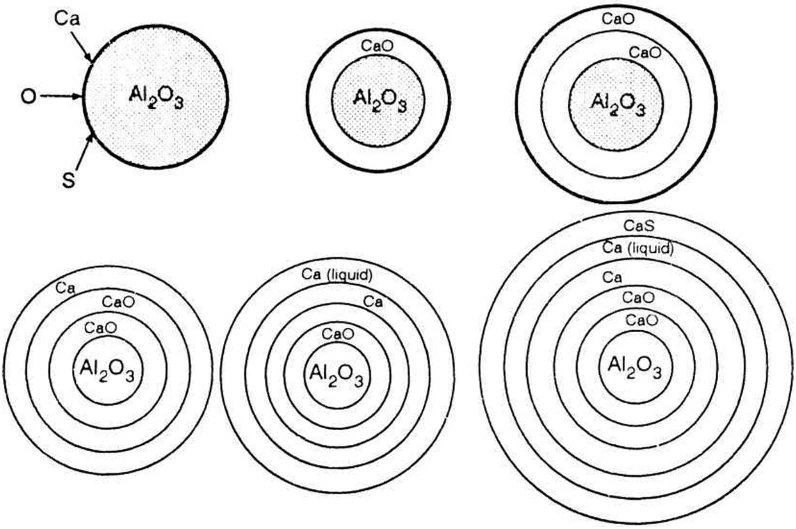


FIGURE 29.3. Shows the mechanism of reaction of CaO with Al_2O_3.

difference is that, due this morphological change, with change to liquid phase or with the liquid cover, the solid particles now behave as liquid globular particles and are, therefore, more prone to easy rise, to join the slag at the surface, than the solid dispersed alumina particles. Calcium injection thus changes the morphology of sulfide and oxide inclusions, in particular of the solid fine pure alumina inclusions and help them rise through the melt depth to join the slag at the surface and thereby produce relatively much cleaner steels. Such treatment obviates many of the problems encountered while casting Al-killed steels in a concast operation.

The requirement of calcium for this purpose is rather critical. It should be enough to convert solid alumina into liquid and combine with the sulfide and silicate particles. Calcium is a costly material and should be used sparingly. Any excess addition is sheer waste and adds to the cost of steel making unnecessarily.

29.11. Vacuum Treatment

No matter what actual steel making operations are adopted for steel making in a given plant, vacuum treatment is the same for a given quality steel production.

Table 29.3.[55]
Comparison and possibilities in various vacuum treatments

	Processes				
	RH Technologies		*Tank and Ladle Technologies*		
	RH-OB *RH-TOP*	*RH*	*VOD* *VD-OB*	*Tank*	*Ladle*
Carbon level	<20	<20	<50	30–40	30–40
Decarb Rate	very high	sufficiently high		less compared to RH	
Decar time	<13 min	<15 min	variable on %C	15–20 min	<20 min
Desulf	seldom Injection Needed	no efficient	highly efficient	highly efficient	highly
Chemical heating	possible	no	possible	no	no
Relative capital cost	1	0.8–0.9	0.4–0.6	0.4–0.5	0.3–0.4
Operating costs	decreasing trend				

The vacuum treatment process that has proved to be very effective on a commercial level is the R-H process as such or its various forms such as RHO, RH-OB, RH-KTB, RH-MESID, and MBF. Since its inception, the R-H process has undergone various modifications to make it more and more efficient. It has several advantages over the others. All vacuum treatment systems perform nearly equally with respect to hydrogen levels achieved, cleanliness improved, and so on. The difference lies in their ability to efficiently decrease carbon and the rate at which it is achieved. The difference is also in terms of their cost of installation and running. This is shown in the form of a comparative statement in Table 29.3.

Ultra-low-carbon levels such as <20–30 ppm are specified for EDD- or IF-grade steels. These can be produced ultimately only by vacuum treatment and nothing else but along with stabilization by Ti, V, Nb, etc. Vacuum treatment can be resorted to in order to remove only the last leg of carbon (*i.e.*, below 500 ppm). It is possible to remove higher than this amount very cheaply in primary steel making itself. Vacuum treatment may also be resorted to when carbon is to be reduced from as high as 0.7%–0.8% to produce steels of approximately 0.02%–0.03% C. This is specifically adopted for stainless steel making. This practice is basically adopted to ensure oxidation of carbon in preference to chromium and which is readily possible under vacuum.

For refining carbon from below a 500 ppm level under vacuum, the bath is allowed to retain nearly 600–800 ppm oxygen dissolved in steel along with. Both the dissolved carbon and oxygen mutually react to form CO gas which gets desorbed from the melt, thereby reducing both of them. Any extra oxygen required is fed from outside or any excess still left over is deoxidized later by aluminum. This can produce steels with less than 30 ppm carbon and finally oxygen to less than 5 ppm by deoxidation. For producing such steels, carbon is removed in the primary process of refining to below 0.05%.

In stainless steel making, using vacuum decarburization, primary refining is carried out until 0.7% –0.8% C and the remaining carbon is removed under vacuum by top oxygen blowing. It takes a longer amount of time to reduce carbon from 0.7% to 0.01% under vacuum. This mode of refining is not adopted to produce EDD- or IF-grade steels. What is practiced for making EDD/IF grades is not economical to produce stainless steels either.

29.12. Residual Contents or Tramp Elements

It has already been discussed in Chapter 5 how the residual elements such as Cu, Sn, Ni, Mo, and Cr present in the charge materials finally end up in the final steel and thereby affect the properties adversely. The maximum amount of these total content as tolerated in various grades of steels are shown in Figure 29.4 in relation to the proportion of various charge materials.

The actual individual levels permissible for various grades are shown in Table 29.4:

Typical chemical composition of residual elements in various charge materials is shown in Table 29.5.

29.13. Metal as Refining Agent

Primary steel making is capable of producing steels up to a carbon level of 0.02% –0.04% only, using pure oxygen gas as refining media.

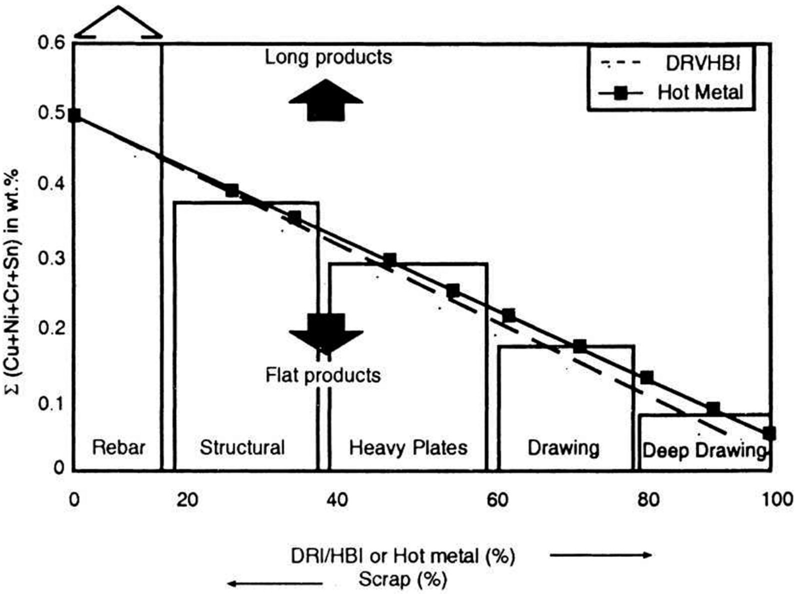


FIGURE 29.4. The level of tramp elements tolerated in various grades of steels as they originate from the variety of charge materials.[56]

Table 29.4.[56]
Shows the permissible levels of impurities in various important grades of steel

Element	*Steel Grades*						
	IF	*DD*	*Drawing*	*Commercial*	*Structural*	*Fine-Wire*	*Rebar*
Cu	0.030	0.040	0.060	0.100	0.120	0.070	0.400
Ni	0.030	0.040	0.060	0.080	0.100	0.070	0.350
Mo	0.008	0.010	0.015	0.015	0.040	0.015	0.080
Cr	0.030	0.040	0.060	0.080	0.100	0.080	0.150
Sn	0.008	0.015	0.015	0.020	0.030	0.015	0.080
Total Residuals	0.106	0.145	0.210	0.295	0.390	0.250	1.060

The ultra-low-carbon IF-grade steels containing 10–20 ppm C and which are required for automotive applications have to be produced by vacuum-oxygen decarburization alone. For such applications, the lower the carbon, the better. Carbon is an interstitial element, and it is very difficult to remove the last traces of carbon from within the iron lattice by using oxygen as the

Table 29.5.[55]
Shows the levels of impurities in various types of steel scraps

Scrap Materials	*Cu*	*Ni*	*Cr*	*Sn*	*Total*
Car shredded	0.230	0.069	0.123	0.052	0.474
Heavy	0.234	0.070	0.130	0.017	0.45
Can	0.050	0.032	0.061	0.128	0.271
Home	0.0221	0.050	0.030	0.030	0.131
H.M/P.I	0.010	0.020	0.020	0.020	0.070
DRI/HBI	0.001	0.008	0.002	0.005	0.016

refining media, even under vacuum. The specification level of below 30–50 ppm is therefore achieved by adding any one individually or a combination of titanium, niobium, and vanadium in small but very controlled amount. These are very strong carbide-forming elements, that is, their interaction with carbon is much stronger than with iron. The strong affinity of these elements for carbon, in particular, fixes the last traces of carbon in the form of these carbides, leaving the iron lattice practically free of interstitial carbon. That is how carbon level of 10–30 ppm is obtained while making the IF grade steels. The Nb, V, or Ti[57] in this case acts in a way as a refining media for removal of the last traces of carbon from steel to produce the desired grade of ultra-low carbon.

29.14. Problem of Nitrogen–Metal Refining

On the whole, nitrogen is generally treated as an impurity, and every attempt is made to remove it during refining. The exception is some stainless steel grades (200 series) wherein nitrogen is used as an alloying element to substitute costly Ni and Mn to ensure austenitic structure even at room temperature. In most other steels, since nitrogen strain hardens the steel and makes further deformation more difficult, it is kept at the minimum achievable level. It is an interstitial element in iron lattice. It is removed to some extent by vacuum treatment. Since most steels even after vacuum treatment are teemed in open air, it picks up nitrogen from air and in a way undoes what was achieved during vacuum treatment. Therefore, vacuum treatment is not generally adopted just for the removal of dissolved nitrogen. There are some special DD-, EDD-, or IF-grade steels which ask for nitrogen level approximately 30 ppm or even less. Special efforts are required to achieve such low levels. Nitrogen is often picked up during

Table 29.6. (a)
Nitrogen at different stages of steel processing

	Nitrogen, ppm				
Stage	*EAF out*	*LF in*	*LF out*	*Tundish*	*Coil*
Mean	32	48	63	72	75
Std Dev	8	10	9	12	8

Table 29.6. (b)
Nitrogen pick-up at various stages of steel processing

	Nitrogen Pick-up, ppm			
Between stages	*EAF-out/ LF-in*	*LF-in/ LF-out*	*LF-out/ Tundi*	*Tundi/ Coil*
Mean1	6	15	9	3
Std Dev	2	−1	3	−4

tapping, teeming, alloying, et cetera. A typical level of such pick-up is shown in Table 29.6 (a) and (b)

Nitrogen pick-up of nearly 16 ppm between EAF-out and LF-in is mainly due to the nitrogen contained in the ferro-alloys added at this stage (*e.g.*, LC-Fe-Cr contains 600 ppm, LC-Fe-Mn contains 400 ppm nitrogen). The nitrogen pick up in LF is due to the time of processing when the bath picks up nitrogen from air. If desulfurization is carried out, it does take time and aggravates the problem. After LF but before tundish, nitrogen is picked up during the holding period, again from atmospheric air. During casting it is picked up from air by the falling steel stream unless special efforts are made to minimize it. The DRI may also contain nitrogen, which is transferred to the bath during its melting.

During arcing in an EAF, the atmospheric nitrogen gets dissociated into nascent nitrogen, which dissolves very readily in molten steel. A good foamy practice can minimize such pick-up. Hot metal contains relatively less nitrogen and is therefore a better charge to minimize nitrogen pick-up. Actions suggested for minimizing nitrogen pick-up are:

1. Careful charge mix, with higher proportion of hot metal
2. Good foamy slag practice in EAF

3. Use of low-nitrogen carburizers and ferro-alloys
4. Reduction of arcing time
5. Reduction of processing time
6. Minimizing desulfurization process needs
7. Minimum holding of refined steel in ladle
8. Use of shrouding media during teeming

As with the fixing of the last traces of carbon (stabilization) by elements such as V, Ti, or Nb having very high affinity for carbon, the last traces of nitrogen are also fixed or stabilized by adding elements such as Ti, Nb, Al, etc., which have a very high affinity for nitrogen. Strong interactions of these elements with nitrogen lead to fix the interstitial nitrogen and thereby reduce its free contents to low levels such as 20 ppm. The Ti, Nb, or Al act here as refining agents.

29.15. Chemical Heating (CAS-OB)

During the processing of a steel heat, at times, it so happens that the temperature of liquid steel drops down to a level lower than the required value. In that case it needs to be heated to the required level before commencing the teeming, to ensure the right quality of subsequent casting operation. This is generally done either by using the arc or induction heat, generally provided in the LF or by chemical heating. Induction-heated LF is avoided because of its high capital and running cost. Arc heating is generally provided in LF. The arc heating, however, suffers from the problem of carbon pick-up from the graphite electrode during the heating operation. The rate of heating in the case of arc heating is generally 3°C per minute. Arc heating is rather problematic when ultra-low-carbon grades such as DD, EDD, or IF, are made. Carbon pick-up of even 20–30 ppm is not tolerated at that stage during the making of heats of these grades.

The problem of temperature fall aggravates when alloying is to be carried out at that stage of refining. Ordinarily, a ton of alloy additions may result in a drop of approximately 6°C of the bath temperature.

Aluminum oxidation is highly exothermic in nature, and it is used as a fuel in such cases to re-heat the steel bath to the required degree. Al-wire is fed along with oxygen jet to burn it to produce the heat. One kilogram of

Al per ton when burnt with oxygen generally leads to heating of the bath by approximately 1°C to 2°C. This process is known as "CAS-OB" (*i.e.*, "composition adjustment by sealed argon and oxygen bubbling") in which Al is burnt in a sealed condition, for heating liquid steel. This is used as a routine practice in many plants. It is also adopted as an emergency operation to avoid spoiling the heat for want of only right temperature. It is recommended that it be used when investment in arc heating is to be avoided consciously.

29.16. Iron Yield

The importance of yield in steel making and how it is evaluated has been dealt with in Chapter 6, while deliberating on efficiency of steel making. In practice, the iron yield is always a matter of concern because of its direct impact on the economics of steel making. During steel making, iron from the charge is lost in the following manner and to that extent the iron yield will be proportionately reduced:

(*a*) As iron oxide in the slag

(*b*) As mechanically entrapped or entrained iron in the slag

(*c*) As vapor of iron due to vaporization at very high temperature

(*d*) As slopping due to dynamic nature of the process

(*e*) As skull formation at the mouth or door of the furnace/vessel

(*f*) As ladle skull

Out of the above, the losses on account of items (*d*), (*e*), and (*f*) are not lost forever. The iron units contained therein are re-circulated as home-scrap in some subsequent heats. The loss due to factor (*c*) as vapor of iron is generally recovered as iron oxide in the gas-cleaning plant in the form of sludge. It amounts to a net loss of iron. This is often due to the excessively high temperature in the arc area and/or at the area of impact of oxygen jet. This is usually in the range of 0.5%–1.0% of the charge weight.

Steel making slag should generally contain 15% –20% iron oxide to effect good dephosphorization during steel making. This is generated by oxidation of iron in preference to oxidation of impurities. This slag is invariably thrown away as a waste product and hence the iron oxide contained therein is a net loss of iron during steel making. The addition of iron-oxide ore to make up this requirement is welcome, and its possibility should always be investigated. In that case, Fe_2O_3 will have to be broken into 3FeO, before it forms part of the slag and which is a net endothermic process. Kinetically, it may also slow down the process of slag formation.

Every attempt should therefore be made to keep the iron oxide within this limit. Any excess of iron oxide in the slag beyond 20% indicates, definitely, a faulty practice, which must be rectified to improve the iron yield.

It should be understood that iron oxide is basic in nature, albeit of weak power as compared to lime or magnesia. Silicon is oxidized invariably during steel making. If adequate lime is not present around, then silica is fluxed in slag by iron oxide, thereby raising the iron oxide content of the slag. It has been found in practice that during oxygen steel making the basicity has direct bearing on the iron-oxide content of slag. The situation is shown typically in Figure 29.5. In fact, the slag exhibits the following equilibrium amongst its own constituents:

$$2FeO.SiO_2 + CaO = 2CaO.SiO_2 + FeO$$

It suggests that when CaO is less, the activity coefficient of FeO is low. But it increases with CaO content of the slag, and as a result the equilibrium iron-oxide content of the slag decreases. This is obvious from actual practical data, as shown in Figure 29.5.

29.16.1. Practical Yield

Steel making charge contains impurities that are oxidized and removed nearly fully via the slag or gas phase. Charge such as DRI contains gangue oxides, which on melting join the slag. Steel scrap may contain dirt as well as iron oxide as rust, which again join the slag. Therefore, the charge actually contains much less weight of iron units than the gross weight measured. The yield is therefore reduced to the extent these are present in the charge.

Iron Yield = Total metallic charge weight – impurities oxidized – gangue present in (DRI) – dirt present in scrap – iron loss as iron oxide in slag – iron loss as mechanical entrapment in slag – iron evaporation loss – iron loss as skull in furnace and ladle expressed as % of the total charge.

The iron-oxide content of slag during oxygen steel making can be minimized to the ideal level by raising the basicity of the slag, that is, by addition of lime/dolomite. Since lime dissolution is not complete in faster oxygen steel making, apparent basicity of the order of 3–3.5 is commonly adopted to bring the iron-oxide content of the slag to the ideal level. This was not

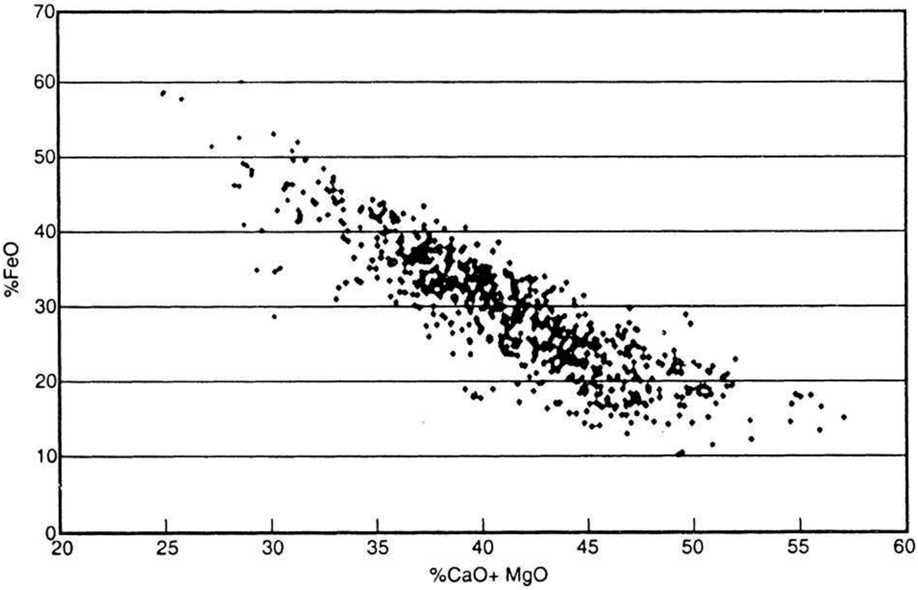


FIGURE 29.5. Shows how the iron-oxide content in slag as it varies with base content.

required to be that high in earlier slower open hearth processes wherein basicity of the order of 2–2.5 was considered adequate. It might have been due to complete dissolution of lime or longer time duration available for desulfurization in the open hearth process.

In oxygen refining, with basicity in the range of 3–3.5, it is expected to give 15% –20% iron oxide in the slag and thereby minimize the loss of iron on this account. If the basicity falls, iron oxide content is bound to rise. In steel making, the amount of slag generally produced is of the order of 15% of the charge weight. When this slag contains 15% iron as oxide, it amounts to a loss of yield of 2.25% of the charge weight.

In a 100-t heat, ideally 2.25 t of iron will be lost in the slag, and that is accepted as inevitable. But by faulty practice, if the basicity falls to 1.5–2.0 (which might even be on account of non-availability of lime), then the loss of iron may be in the range of 4–5 t, which is a net drain on the economy. This is obvious from Figure 29.5.

CHAPTER 30

MODERN STEEL PLANT MANAGEMENT PRACTICES

In This Chapter

- Introduction
- ISO-9000 Quality System
- Total Quality Management (TQM)
- 3-M Approach
- 5-S Approach
- Statistical Process Control (SPC)
- Six Sigma (6σ)
- Total Productive Maintenance (TPM)
- Economic Value Addition (EVA)
- Balanced Score Card (BSC)

30.1. Introduction

Modern management practices of steel plants have to deal with the following parameters to effect maximum productivity, quality of steel products, and overall economy:

(*a*) Product-mix with respect to steel grades and their marketability

(*b*) Optimum combination of steel processing units to produce a specific grade at best possible economy

(*c*) Optimum process operation for maximum productivity and economy

(*d*) Process control and operational details for obtaining best possible product quality (*e*) Avoidance of waste, in particular iron in slag, gas, et cetera, for better economy

(*f*) Synchronization of production, on the whole and, of different grades to meet market demands

(*g*) Forecasting of market to be ready to meet any eventuality

This strategy is meant to meet challenges of making steel products within the plant and of marketing them successfully against similar products from rival competitors. The evolution of efficient and economical practices is essential in view of the varieties of products, their complexity, their quality, equally complex process operation, and the volume of its production.

Even a small steel plant involves coordination of at least a few hundred workers inside and outside the plant. This figure grows to several thousand for an integrated steel plant, thereby increasing the complexity of its management. Sustainability of a steel plant requires continuous improvement in value addition by reducing the cost of production and also by value creation. Cost of production is to be continuously reduced by eliminating waste, by better control over the production process for minimizing the defects, and also by adopting efficient process methods. This requires continuous and sustaining review of the production and marketing of the products in view of the rival scenario. The bottlenecks need to be removed to increase value creation by way of value-added products. R and D efforts in the entire process of steel making become inevitable to remain competitive against the external rivals.

The quality system is just a part of this overall but very important and dominant strategy. The productive operations are described in all possible details so that when it is followed, the required quality is achieved. This is known as “Standard Operating Practices” (SOP for short). This also requires “Standard Maintenance Practices” (SMP for short) of the equipment. It is always open to further scrutiny, as and when needed, in light of experience and requirements. The quality policy also insists on assuring the product quality via a “Technical Delivery Certificate” (TDC for short) of the product to the customers.

In order to handle the above issues, almost every steel plant tries to implement one or several of the available management systems. Some of them are being discussed below in brief to provide an introductory overview of the same.

30.2. ISO-9000 Quality System

This is an international standard certifying the enterprise as following standard practices to ensure quality of the product. Initially, an ISO-9000–certified company was issued three kinds of certification—ISO-9001, ISO-9002, or ISO-9003—depending upon the functions they performed and the certification they applied for looking into their capability. This kind of differentiation in certification existed in the versions of the systems of 1987 and 1994. It was improved by its revised version in the year 2000.

The quality assurance requires creation of a management system that is designed for its effective implementation. In brief, the system requires a quality policy of the steel plant. This is to eliminate the problems of inconsistency and absence of documented procedures for carrying out the steel plant activities. The details of processes depend on its complexity. Variations are bound to come because of varying psyche, competence level, approach, and attitude of the people involved in the processing. Therefore, a great deal of emphasis is laid on the preparation of detailed documented procedures for handling various operations covering the entire business and then creating an audit system to ensure that these processes are carried out, as far as possible, the way they are defined in the relevant idealized document. The ISO-certified establishments are supposed to follow these procedures meticulously, to automatically assure designed quality of the products.

It is expected that such a system shall bring in more consistency in operation and will allow for identification of grey areas, for its possible improvement. Periodic external auditing is also carried out to ensure that the system is in vogue and, is placed properly to consciously effect continuous quality improvement and does not lose its strength, to die a natural death.

30.3. Total Quality Management (TQM)

The ultimate goal of a customer's requirements is often missed since many of the important business processes are not covered under ISO-9000 certification. The customer requires quality product and after-sales service, if needed. It is in the interest of the supplier to retain every customer, in spite of the challenges put forth by the rivals. Retaining them as loyal customers is not only a big challenge but is important for the very survival of the company. TQM practice has been evolved to keep customers more delighted by meeting more than their expectations by supplying them with the products and services that have ever-increasing quality.

Therefore, it becomes necessary to orient all the steel making processes ultimately towards the customer's delight with quality being considered as the most important strategy. TQM has now been recognized as a powerful general business tool that developed through major contributions of Edwards Deming, Juran, Philips Crosby, Genichi Taguchi, and many others. They focus on bringing about change in the organizational culture wherein the decisions are taken based on only facts and not on gut feelings and, without exception, to always please the customers, whether internal or external to the enterprise. To promote this healthy practice, quality awards such as the Juran Award and the Malcolm Baldrige Award have been instituted. Earning such an award is viewed as a measure of enforcement of such quality standards in any steel plant. It also indicates the credibility and strength of the steel plant in terms of quality standards. This can happen only if leadership of the steel plant is committed to quality production.

30.4. 3-M Approach

These three Ms stand for the following three Japanese words: (i) *Muri*, (ii) *Muda*, and (iii) *Mura.* The parallel words in English are *Unnatural, Waste*, and *Unevenness.* Basically, the 3M approach looks for improvement in all the production and other business processes, recognizing them as falling into three major segments, for the single achievement of improvement in productivity.

The focus on *Muri* is to review whether the methodology, posture, working, and practice are unnatural and if so, to what extent and to improve upon them. For example, an unnatural and extra-posture lays a great deal of strain on the worker, which may lead to early fatigue and thus low productivity.

Muda-focus targets to minimize all kinds of wastes to improve economics and efficiency of production and sales. It also helps to identify any activity that is not helping to add value. It is therefore identified and sorted out. This also helps to identify defects generated in any product or process and endeavors to minimize the same to improve process efficiency. This requires laying adequate emphasis on Statistical Process Control (SPC), which is explained later in detail.

Mura is aimed to find the difficulty(ies) coming in the way of assimilating the existing or the proposed practices in improving the productivity. It is mainly related to the psyche of the workforce involved in the operation. Subjective variations, in general, lead to problems related to quality, planning, waste, process control, and so on, and which in turn affect productivity adversely. Therefore, it is necessary to reduce the non-uniformities in the form of subjective variations in the operations for producing the required quality product with expected efficiency.

30.5. 5-S Approach

5-S encompasses all the features of 3-M but is mostly individual-centric. It is where an individual has to continuously focus on making improvements in his working methodology so that the issues of quality and productivity are better taken care of. These five Ss stand for five different Japanese words starting with "S". Their meaning, concern, and scope are described below.

30.5.1. Seiri

It is very common to find in practice that many of the articles, lying even in safe lock-up, or around the workplace, are not going to be used in the near future. These are not necessarily scrap; they may have some functional value, but such items, on accumulation, create avoidable problems, such as:

(*a*) Consuming excessive time in locating them when required

(*b*) Wasting valuable space, causing an increase in overhead costs

(*c*) Blocking of capital on such items

(*d*) Wasting the space/area used to store them

The above factors are clearly representative of inefficient and uneconomical working. For this, the *Seiri* function suggests that the operators, being the best judges, should evaluate their requirements for carrying out the necessary operations. They should themselves sort out unnecessary items from their workplace and keep only those that are routinely required. It also says that they should try to avoid maintaining an unreasonably high level of spares-inventory. Apparently, it appears equivalent to the *Muda* function of 3-M.

30.5.2. Seiton

Even if unnecessary items have been sorted out but are not kept properly arranged at a proper place, time will be wasted in locating any item when really required. This may result in decreased productivity. Therefore, this function desires that the individuals keep the required items and their quantities in an order that facilitates their use with the least possible search and organizing times.

30.5.3. Seiso

This function is related to maintaining the equipment in a steel plant in proper state and shape to keep productivity at its maximum. It helps the operator to take utmost care of the machines, equipment, and other measuring and testing instruments that are being used for day-to-day operations. It requires the operator to develop maintenance schedules of all such items by their periodical inspection and monitoring. Settings of machines and instruments may change with time and thus they need to be adjusted and calibrated regularly so as not to affect process control adversely.

30.5.4. Seiketsu

Here, the objective is to keep high standards of overall working conditions. In addition, the workers are expected to look at their surroundings and think of exercising all possible control, so that their workplace is pleasant enough for work. This, in turn, shall help in maintaining good hygienic conditions at the workplace.

30.5.5. Shitsuke

This function lays emphasis on the training component. It is a common experience that only a particular level of achievement is possible from an operator, commensurate with the current level of his/her knowledge and learning. For further improvement, more insight is required, and this naturally calls for improvement of their knowledge base. This is possible only through participation of the workforce, at all levels, in training programs and workshops. This suggests that the operators should in the first place be capable enough to identify their training needs and mentally be ready for further learning. This activity is possible only if the workforce is considered as an asset and capable of further improvement through training.

30.6. Statistical Process Control (SPC)

SPC is basically a technique for analysis of complex data of a production process wherein more than a few variables are involved. The variables of any process or operation can be quite large. Steel making and its marketing, in fact, typically represent such a complex process. It is necessary to appreciate the exact role played by any variable in such a complex process as steel making. It requires that the productivity, quality, and economics of steel production and its marketing are related to all those variables of this complex system. Statistical processing is best suited for analysis of such a complex system. Such correlations throw light on the effect of any one or more parameters on the productivity, quality of the product, and economics of the operation.

This analysis indicates the extent of influence of variable parameters of the process on performance. It also indicates the deficiency in adjusting the process parameters in the existing operation for achieving the best possible performance. The SPC thus involves data analysis, the evaluation of individual process variables, and their influence on the overall process, and by adjusting them to a required value to ultimately ensure that the operations do not suffer from any deficiency in terms of process control. This exercise should be carried out periodically as a routine matter of process control. It also confirms whether the process has deviated from the standard prescribed practice and to what extent. In case of deviation, it reveals the cause of deviation, and on rectification, ensures that this deficiency is eliminated. Vast variation is an indication of the process being out of control and requires more detailed analysis for effective process control.

30.7. Six Sigma (6σ)

The basic approach of SPC is to simply control the unusual large variations in process parameter values. However, if the permissible variations are not small enough, they may cause a serious concern for quality in those cases where the variations may go beyond the spread defined by tolerance limits. In such cases, even SPC will not suffice in achieving zero-deficiency situations. Therefore, it becomes natural to evolve strategies for reducing the natural variability of the operation and to bring it under control.

Most of the world measurements are seen to follow “Normal Distribution” that has a bell-shaped curve, with mean, median, and mode having the same value. This distribution has a special characteristic of capturing 99.73% cases within ±3σ distance from the mean, where ό represents the standard deviation value and is a measure of variability in the process parameter values. In case a process parameter, set at a mean value, is allowed to have variation such that a ±3σ value matches with the tolerance spread, then 0.27% values (27 out of 10,000) can be noticed beyond the permissible zone of variation and thereby become a concern for quality. The six sigma approach requires this variation to be much smaller and to the extent where the tolerance spread is covered minimally by $\pm 6\sigma$ values. In this case, the process is expected to result on an average only 3.4 deficiencies over 1 million, with practically no chance and worry for bad quality.

Six-sigma philosophy does not limit itself to what has been said in the preceding paragraph. It calls for a change in the mindset of the workforce and thus the culture of the organization wherein the decisions are based on facts and data, and not on whims and gut feelings. Because of the strength of the six-sigma approach, in doing away with the deficiencies and increasing productivity, organizations are using this technique on a increasing scale. Persons are being formally trained and, depending upon their learning, they are certified as Yellow-belt, Green-belt, Black-belt, or Master Black-belt, indicating their ability for effectively using this tool.

30.8. Total Productive Maintenance (TPM)

In short, TPM is all about running a maintenance program for the plant and equipment in a most productive manner wherein maintenance is properly scheduled and is considered as essential as manufacturing, and the down time is not regarded as ill. Benefits from the TPM are seen in the form of better condition of machines and workplace, and very low occurrence of breakdown maintenance. In terms of its basic ingredients as employee-empowerment, benchmarking, documentation, etc., it seems to closely resemble TQM. This is true, but it mainly focuses on maintenance of equipment which, if taken up seriously and sincerely, is bound to keep the processing facilities healthy and, thereby, the production.

The core of TPM is "Autonomous Maintenance" (*Jishu Hozen*), which can be routinely carried out by the operators themselves on the equipment they use. Heavy work content in maintenance shall, of course, demand the help of a maintenance crew.

5-S forms the basis of TPM, with autonomous maintenance being one of the eight important pillars of TPM, as are described in the accompanying figure. The Kobetsu Kaizen approach desires continuous improvement by eliminating waste. Planned maintenance is basically to be carried out for achieving zero breakdowns. Quality maintenance (*Hinshitsu Hozen*) basically makes use of the "*poka-yoke*" concept to come up with a foolproof method to avoid wrong setup and assembly in order to achieve zero loss. Training is important for sustaining ever-increasing productivity. Office TPM processes are aimed at improvement in a manner similar to that for the plant and equipment. Safety, hygiene, and environment are imparted greater importance in a TPM module so as to facilitate zero work-related accidents and a better environment for working. Equipment/product management focus is adopted for reducing waste of all kinds during either the installation of a new machine or the launch of production for a new product.

30.9. Economic Value Addition (EVA)

EVA stands for "Economic Value Added (or Addition)." It is a novel method for judging the financial health of any company. For the employees, it becomes very important to achieve a higher EVA level, as their variable pay may get linked to it. EVA tries to measure the gap between so-called expected return from equity as well as long-term debt capital and the profit that the company has earned before accounting for long-term interest and income tax. The earning takes into account all the expenses, including depreciation, interest on short-term borrowings, and amortization. The expected return is arrived at, based on a desired rate of return. This rate is more than a weighted average cost of the debt capital. The difference between the two is company- and industry-specific. For example, for steel companies it is anywhere between 11 to 15. For the Tata Iron & Steel Company (TISCO), it is approximately 11% and for Ispat Industries Limited it is approximately 14%. Figure 30.1 is House of TPM.

30.10. Balanced Score Card (BSC)

Balanced score card is a technique for properly designing the strategies that shall help in achieving the stated vision and mission statements of any enterprise. Realizing that the traditional business performance measures focus on short-term achievements and do not address the long-term sustainability, Robert S. Kaplan and David P. Norton had proposed this methodology in 1992. The scoring mechanism tries to balance short-term business performance against the long-term ones and, therefore, decisions and actions pertaining to long-term growth and sustainability of the company are not ignored. It also tries to seek balance between quantifiable measures with qualitative ones, and financial outcomes with the human aspects, including customers and employees. It is how BSC can be seen

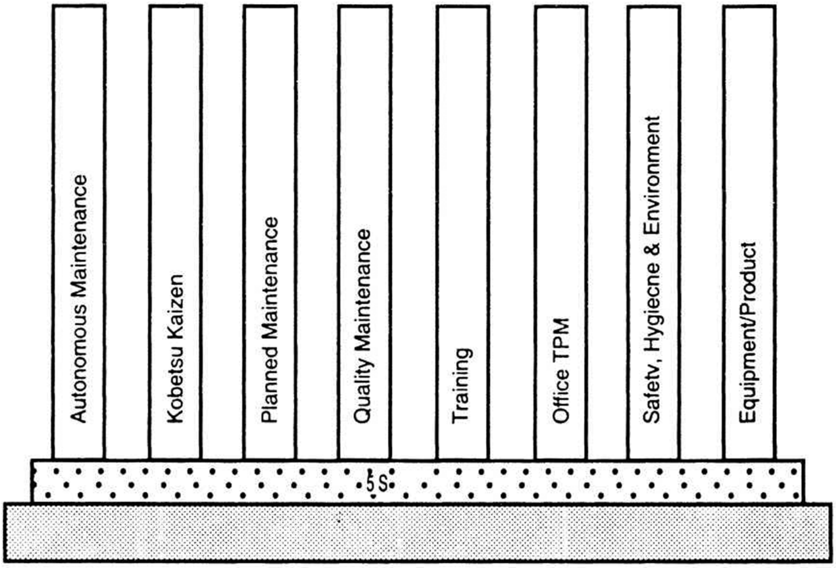


FIGURE 30.1. House of TPM.

to balance the financial perspective with customer, process, and employee perspectives. Strategies employed are mapped on four sections called perspectives—Financial, Customer, Internal Business Processes, and Learning & Growth—with links emanating from the bottom level of the Learning & Growth perspective (basically, employee perspective) to the Internal Business Process perspective and along with finally linking the Financial perspective. Linking is done keeping in mind the causal relationship between various items defining various perspectives. After the strategy mapping has been completed, a scorecard for everyone, right from the chief executive to the lowest executive in the hierarchy, is prepared, enlisting these identified factors where they are supposed to focus. While doing so, utmost care is taken to ensure that the scorecard of a subordinate is linked properly to the scorecard of its peer in a quantifiable manner. Therefore, efforts are needed to define how the performance of each one shall be measured against each of these factors. Naturally, the target values are to be defined against those which the performance has to be measured.

The above-defined four perspectives are not sacrosanct. Some may like to merge closely similar twos of the perspective and would like to give different names to it. Some companies go for one more perspective as Corporate Social Responsibility, which is generally kept in the bottom of the strategic map.

APPENDIX A

STEEL SPECIFICATIONS

In modern times when commodities are manufactured and sold, particularly in engineering industries, every product is certified as of a specific type, grade, and possessing certain minimum required properties for the buyer to be reassured. All these requirements are covered under the term quality of the product. This can be done on an individual manufacturer's and/or buyer's basis. It would then be required to be done for every customer individually. This is far too complicated in the present-day world. Some neutral agencies have therefore been historically evolved and have been entrusted to evolve quality standards of a particular product, to ensure its minimum performance under a given set of conditions. Such information pertaining to the quality of any particular product has been accepted as the specifications of that product. Each such specification of a product is given a unique code number, which is not duplicated for any other product, in the same category of products by the same certifying agency. Steels are no exception to these accepted norms.

Steels are therefore depicted individually by a unique code number to indicate, as far as possible, fully the unique quality of the product. These are known as steel specifications. Such specifications have been developed by many nations, and within them, by one or more different neutral professional agencies. These thus become the benchmark standards for the specific products. The specification code carries the name of that agency and a unique number that certifies the quality of that product. These are unique code numbers characteristic of that agency for the specific product.

The evolution of specifications is a unique characteristic of industrial evolution. Naturally the countries where the industrial revolution occurred initially are chiefly the UK, Germany, France, and the USA, and where the specifications have been extensively developed for almost any and every industrial product. Practically every country is trying to develop its own such specifications, under its own code numbers, for more and more goods. This is at times done even by adopting the existing specifications of some other standards with minor local variations.

For steels the above four and those of Japan, Russia, and China are available. In India, Indian Standards Institution (ISI) has been entrusted with this work. There are several such IS steel specifications evolved for general use. Standards are commonly used to denote properties of steels sought or produced. There are various standards from the USA, like SAE, AISI, API, and so on. The BS or EN series are popular standards from the UK. German standards are known as DIN. Indian standards are known as ISI. With expanding world trade, international standards have also been evolved to avoid confusion because of varying national standards, for trading the product, and are known by IS or SI (international standards) code.

For steels it is the chemistry that is primarily quoted in every specification. It includes the contents of useful alloying elements and may also contain the maximum limits of undesirable elements. But since steel is not fully described only by its chemistry, additional properties are mentioned as per the application needs.

It may contain mechanical properties like tensile strength and percent of elongation as very primary needs for almost every application. It helps to economize steel requirements for a given application. It may however mention one or more other mechanical properties depending upon the criticality of its use. For several applications physical dimensional properties like size, its variation, surface smoothness, and so on may be relevant. Even outer appearance like color, luster, etc., is also included in the specifications of steels. Steel properties are very much a function of its microstructure. Hence microstructure, grain-size, cleanliness level, etc., are also included in specifications. The chemical quality of inclusions, their sizes, shapes, and distribution are very important properties for certain applications and may be included in the specifications. Steels vary considerably in properties with respect to non-uniformity like surface defects, internal porosities, non-uniform microstructure, segregation, and so on and all of these also form an important part of steel specifications.

In fact there is no limit to prescribing any and every property, if it influences performance of a given steel for a certain application, and it therefore forms an inalienable part of its specifications. The more detailed the specifications, the better would be the quality of the product ensured, for a given application.

The specifications are constantly under scrutiny to improve upon them and user's critical comments are taken into account in improving and revising them, from time to time, to better suit the given application. The specification code/standard invariably shows the year of its first formation and also the year of any revision.

The specifications are of vital importance for the first time user of that product, since it gives advance assurance of its performance. These are equally useful for designers who guarantee performance of their design based on the specifications of the materials used therein.

APPENDIX B

IRON AND STEEL (U.S. AND WORLD)[1]

By Michael D. Fenton

The world steel industry continued to add more production capacity during the period from 2009 to 2014 despite continuing weak demand as the world economy recovered slowly from the economic downturn in 2008–9. By 2014, global overcapacity was an estimated 330 million metric tons (Mt) with an average utilization rate of less than 80%. North American steel plants managed to limit overcapacity, whereas substantial overcapacity developed in Central Asia, the European Union (EU), Russia, and South America. Overcapacity in China, estimated at 200 Mt, has reached the point where the Chinese Government has limited bank financing to add capacity and has required the closure of small, unprofitable, and polluting plants. Implementation by local governments that are concerned about economic development and social stability continues to be an issue. Similar problems of closures and planned closures took place in Belgium, France, Italy, and Serbia. Apparent consumption in China, the world's leading consumer of steel, increased steadily from 101 Mt in 1995 to a high of 766 Mt in 2013 and then decreased to 740 Mt in 2014 (World Steel Association, 2015a).

Production

Raw steel production in the United States was about 88.2 Mt in 2014, up slightly from that in 2013 (table 1). U.S. capacity utilization for raw steel

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[1] Source: USGS https://minerals.usgs.gov/minerals/pubs/commodity/iron_&_steel/

[2] Domestic survey data and tables were prepared by Hoa P. Phamdang, statistical assistant, and world production tables were prepared by Glenn J. Wallace, international data coordinator.

production, which had dipped to a low of 41% in April 2009, rose steadily to a high of 80.2% in August 2014, ending at 74.6% in December 2014. The American Iron and Steel Institute (AISI) estimated raw steel production capacity in 2014 to be 114 Mt, up slightly from that in 2013. Production represented 77.5% of estimated capacity, up from 76.7% in 2013 (American Iron and Steel Institute, 2015, p. 73).

Integrated steel producers smelted iron ore to make liquid iron in blast furnaces and used basic oxygen furnaces (BOFs) to refine the liquid iron, with some steel scrap, to produce raw liquid steel. The BOF process was used to make 33.0 Mt of steel in the United States (American Iron and Steel Institute, 2015, p. 70). The use of this process decreased to 37.4% of total steel production in 2014 from 39.4% in 2013. Blast furnaces in the United States were operated by three companies at 11 locations in 2014. AK Steel and Severstal North America closed one blast furnace each, both in Dearborn, MI, during 2014 (Iron and Steel Technology, 2015b, p. 294).

Minimills and specialty mills are nonintegrated steel producers that use electric arc furnaces (EAFs) to melt low-cost raw materials (primarily scrap). They also employ continuous casting machines and hot-rolling mills that are often closely coupled to casting operations. Specialty mills include producers of electrical alloys, stainless, and tool steel; high-temperature alloys; forged ingots; and other low-volume steel products. About 58 companies operated about 109 EAF facilities in the United States during 2014 (Iron and Steel Technology, 2015a, p. 158). These mills produced 55.2 Mt of steel in 2014, 5% more than in 2013, and accounted for 62.6% of total steelmaking (American Iron and Steel Institute, 2015, p. 70).

Raw liquid steel is mostly cast into semifinished products in continuous casting machines. Only 1.5% of U.S. production was cast in ingot form in 2014 and subsequently rolled into semifinished forms, a slightly higher percentage than that of 2013. Continuous casting production was 86.8 Mt, or 98.5% of total steel production, slightly higher than in 2013 (American Iron and Steel Institute, 2015, p. 70).

Consumption and Shipments

Steel mill products are produced either by forging or by rolling into forms normally delivered for fabrication or use. Some companies purchase semifinished steel mill products from other steel companies and use them to produce finished steel products. The accumulated shipments of all companies less the shipments to other reporting companies are identified as net shipments to avoid double counting.

U.S. apparent steel consumption, an indicator of economic growth, which had decreased to a low of 63 Mt in 2009, after a high of 120 Mt in 2006, increased to 108 Mt in 2014 from 99 Mt in 2013. Net shipments of steel mill products by U.S. companies increased by 3% to 89.1 Mt compared with those of 2013 (American Iron and Steel Institute, 2015, p. 23). Compared with those in 2013, shipments of construction products, the leading single end-use market for steel, increased by 10.6% in 2014; automotive product shipments decreased slightly; shipments of agricultural products increased by 15%; shipments of industrial machinery, equipment, and tools increased by 9.8%; oil, gas, and petrochemical shipments increased by 10.1%; steel service center shipments decreased slightly; lumbering, mining, and quarrying industries shipments decreased by 17%; shipments of appliances increased by 8.3%; and shipments of containers, packaging, and shipping material decreased by 50% (American Iron and Steel Institute, 2015, p. 27).

Prices

The U.S. Bureau of Labor Statistics (2015) producer price index for steel mill products increased slightly to 200.2 in 2014 from 195.0 in 2013 (1982 base = 100) (table 1). The average monthly price of hot-rolled steel sheet fluctuated within a narrow range from a high of $447 per metric ton in January 2014 to a low of $599 per metric ton in December 2014 (American Metal Market, 2014).

Foreign Trade

Export shipments of steel mill products by AISI reporting companies decreased to 10.9 Mt from 11.5 Mt in 2013 (table 4). Canada received the largest amount of U.S. exported steel products, 5.8 Mt, 5% less than that in 2013 (table 4). Mexico was again the second-ranked export receiving country, 3.8 Mt, 3% more than that in 2013. Domestic imports of steel mill products increased by 37.9% to 40.2 Mt from 29.2 Mt in 2013 (table 4). Brazil, Canada, China, the EU, Japan, Mexico, the Republic of Korea, Russia, and Turkey were major sources of steel mill product imports in 2014 (table 4).

Imports of semifinished steel (table 6) by steel companies are taken into consideration in evaluating apparent consumption (supply) of steel mill products in the United States and the share of the market represented by imported steel. To avoid double counting the imported semifinished steel and the products produced from it, the amount of semifinished steel consumed by companies that also produced raw steel is subtracted from domestic consumption. Between 1993 and 2014, semifinished steel imports ranged between 2.5 and 9.6 million metric tons per year (Mt/yr). Prior to 1993, the amount was less than 0.2 Mt/yr. Taking the imported semifinished steel into consideration, the share of the U.S. steel market represented by imported steel was an estimated 37% in 2014 compared with 30% in 2013.

World Review

World production of pig iron totaled about 1.19 billion metric tons (Gt), slightly more than that in 2013 (table 9). China continued to be the leading producer of pig iron in the world, producing 712 Mt, slightly more than that of 2013, followed by Japan (83.9 Mt), India (55.2 Mt), Russia (51.5 Mt), the Republic of Korea (46.9 Mt), the United States (29.4 Mt), Germany (27.4 Mt), Brazil (27.0 Mt), and Ukraine (24.8 Mt). Russia and Ukraine were the only major pig iron producers in the Commonwealth of Independent States (CIS), where production in 2014 decreased by 5% from that in 2013. In South America, the only major pig iron producer was Brazil. Germany was the leading producer in the EU, producing about 27.4 Mt, slightly more than that in 2013.

World capacity for direct-reduced iron (DRI) production in 2014 was estimated to be about 108 Mt/yr (Midrex Technologies, Inc., 2015, p. 11). DRI production worldwide increased by 6.6% to 71.3 Mt in 2014 from 66.9 Mt in 2013 (table 9). The leading producer of DRI was India (20.4 Mt), followed by, in descending order of tonnage, Iran (14.6 Mt), Saudi Arabia (6.46 Mt), Mexico (5.98 Mt), and Russia (5.20 Mt). In 2014, additional DRI capacity of almost 16 Mt/yr was under construction in China, Egypt, India, Russia, the United States, and Venezuela. The leading technology, according to declining order of production, was the Midrex process, followed by coal-based and HYL/Energiron.

World production of raw steel was 1.67 Gt, 3% more than that in 2013 (table 10). Steel production during 2014 increased in North America (1.8%), decreased in the CIS (2.5%), and decreased in the EU (1.0%).

Positive growth also took place in India (6.5%) and China (5.6%). As in previous years, production varied widely among major regions of the world. China produced 49% of world total crude steel in 2014. Countries in Asia produced about 68% of the world's steel; the EU, 10%; North America, 7%; and the CIS, 6%.

According to the World Steel Association (2015b), China was the top steel producer in the world during 2014 (823 Mt), with its leading steelmaker, Hebei Iron & Steel Group Co. Ltd. (47.1 Mt), placing third behind the world's leading steelmakers—ArcelorMittal (98.1 Mt) and Nippon Steel and Sumitomo Metal Corp. (49.3 Mt). Japan was the world's second-ranked steel producer (111 Mt), followed by the United States (88.2 Mt), India (86.5 Mt), Russia (71.5 Mt), and the Republic of Korea (71.0 Mt). These six countries accounted for 75% of world production. Russia and Ukraine remained the leading producers in the CIS. China ranked first in the world for total exports of steel during 2014 (92.9 Mt), followed by Japan (41.3 Mt) and the EU (37.1 Mt). The United States ranked first in the world for total imports of steel in 2014 (41.4 Mt), followed by the EU (32.4 Mt). Increases in steel production during 2014 took place in Africa (3.1%), Asia (3.9%), the CIS (2.7%), the EU (1.0%), and North America (1.8%). Steel production decreases during 2014 took place in South America (3.3%) and Africa (4.7%).

Outlook

The expansion or contraction of gross domestic product (GDP), the broadest measure of a nation's economic activity, may be considered a predictor of the health of the steelmaking and steel manufacturing industries, worldwide and domestically. The World Bank's forecast of global GDP growth for 2015, 2016, and 2017 is 2.8%, 3.3%, and 3.2%, respectively, after 2.6% in 2014 (World Bank, The, 2015). The International Monetary Fund (2015) projection of GDP growth is 3.3% for 2015 and 3.8% for 2016. The U.S. Federal Reserve's September 2015 projections for GDP rate of growth for the United States are 2.1% for 2015, 2.3% for 2016, and 2.2% for 2017 (Board of Governors of the Federal Reserve System, 2015). The 2014 rate of GDP growth for China was 7.4% and is projected to be 6.8% in 2015 and 6.3% in 2016. The rate of GDP growth for India was 7.3% in 2014 and is projected to be 7.5% in 2015 and 2016 (International Monetary Fund, 2015).

MEPS (International) Ltd. forecast total world steel production in 2015 to be 1.62 Gt, down slightly from that in 2014 (World Steel News, 2015). MEPS also forecast an increase to 1.64 Gt in 2016.

World apparent steel consumption (ASC) is forecast to be 1,513 Mt and 1,523 Mt in 2015 and 2016, respectively, after consumption in 2014 was 1,540 Mt. (World Steel Association, 2015c). China's ASC is expected to decrease to 672 Mt in 2016 from 711 Mt in 2014. The ASC in India is expected to increase to 87.6 Mt in 2016 from 75.9 Mt in 2014. The ASC in the United States is expected to decrease to 105 Mt in 2016 from 107 Mt in 2014. The EU's ASC is expected to increase to 153 Mt in 2016 from 148 Mt in 2014. In Japan, the 2015 ASC is expected to decrease to 66 Mt in 2016 from 68 Mt in 2014. The ASC of the CIS is expected to decrease to 49.9 Mt in 2016 from 56.1 Mt in 2014.

REFERENCES CITED

American Iron and Steel Institute, 2015, Annual statistical report 2013: Washington, DC, American Iron and Steel Institute, 120 p.

American Metal Market, 2014, Custom pricing reports: New York, NY, American Metal Market. (Accessed August 4, 2014, via http://www.amm.com.)

Benteler Steel/Tube GmbH, 2013, Benteler Steel/Tube breaks ground at $975M U.S. facility in Louisiana: Houston, TX, Benteler Steel/Tube GmbH press release, September 16. (Accessed October 10, 2014, at http://www.fabricatingandmetalworking.com/2013/10/benteler-steeltube-breaks-ground-at-97SM-u-s-facility-in-louisiana/)

Board of Governors of the Federal Reserve System, 2014, Minutes of the Federal Open Market Committee, Summary of economic projections, September 16–17, 2014: Board of Governors of the Federal Reserve System. (Accessed June 24, 2016, at http://www.federalreserve.gov/monetarypolicy/fomcminutes20140917ep.htm.)

Board of Governors of the Federal Reserve System, 2015, Minutes of the Federal Open Market Committee, Summary of economic projections, September 16–17, 2015: Board of Governors of the Federal Reserve System. (Accessed January 27, 2016, at http://www.federalreserve.gov/monetarypolicy/fomcminutes20150917ep.htm.)

Business Wire, 2013, Borusan Mannesmann pipe partners with Siemens for its first U.S. manufacturing facility: New York, NY, Business Wire press release, June 4. (Accessed April 23, 2014, at http://www.businesswire.com/news/home/20130604006484/en/Borusan-Mannesmann-Pipe-Partners-Siemens-U.S.-Manufacturing#.VFJrmPnIako.)

Ernst & Young, 2013, Global steel 2013—A new world, a new strategy: Ernst & Young, 44 p. (Accessed September 5, 2014, at http://www.ey.com/Publication/vwLUAssets/Global_steel_2013_-_A_new_world_-_a_new_strategy/$FILE/Global_Steel_Report_India_conference_Feb_2013.pdf.)

Ernst & Young, 2014, Global steel 2014: Ernst & Young, 41 p. (Accessed September 5, 2014, at http://www.ey.com/Publication/vwLUAssets/EY_-_Global_steel_2014/$FILE/EY-Global-steel-2014.pdf.)

International Monetary Fund, 2015, Slower growth in emerging markets, a gradual pickup in advanced economies: New York, NY, International Monetary Fund press release, July 24. (Accessed January 27, 2016, at http://www.imf.org/external/pubs/ft/weo/2015/update/02/pdf/0715.pdf.)

Iron and Steel Technology, 2015a, AIST2015 EAF roundup: Warrendale, PA, Iron and Steel Technology, January, 280 p.

Iron and Steel Technology, 2015b, AIST2015 North American blast furnace roundup: Warrendale, PA, Iron and Steel Technology, March, 352 p.

Midrex Technologies, Inc., 2015, 2014 world direct reduction statistics: Charlotte, NC, Midrex Technologies, Inc., 14 p. (Accessed August 4, 2014, via http://www.midrex.com/.)

Miller, L.B., 2014, Developments in the North American iron and steel industry—2013: Warrendale, PA, Iron and Steel Technology, April, p. 192–212.

Nguyen, Lananh, 2012, U.S. oil output to overtake Saudi Arabia's by 2020: Bloomberg News, November 12. (Accessed October 10, 2014, at http://www.bloomberg.com/news/2012-11-12/u-s-to-overtake-saudi-arabia-s-oil-production-by-2020-iea-says.html.)

Smith, Grant, 2014, U.S. seen as biggest oil producer after overtaking Saudi Arabia: Bloomberg News, July 4. (Accessed October 10, 2014, at http://www.bloomberg.com/news/articles/2014-07-04/u-s-seen-as-biggest-oil-producer-after-overtaking-saudi.)

U.S. Bureau of Labor Statistics, 2015, Producer price index highlights: Washington, DC, U.S. Bureau of Labor Statistics. (Accessed January 6, 2016, at http://www.bls.gov/xg_shells/ro4xgppihi.htm.)

World Bank, The, 2015, Global economic prospects: Washington, DC, The World Bank, June. (Accessed January 27, 2016, at http://www.worldbank.org/en/publication/global-economic-prospects/summary-table.)

World Steel Association, 2015a, Steel statistical yearbook 2015: Brussels, Belgium, World Steel Association.

World Steel Association, 2015b, World steel in figures 2015: Brussels, Belgium, World Steel Association. (Accessed January 13, 2014, at http://www.worldsteel.org/publications/bookshop/product-details.~World-Steel-in-Figures-2015~PRODUCT~World-Steel-in-Figures-2015~.html.)

World Steel Association, 2015c, World steel short range outlook 2015–2016: Brussels, Belgium, World Steel Association press release, April 20. (Accessed January 27, 2016, at https://www.worldsteel.org/media-centre/press-releases/2015/worldsteel-Short-Range-Outlook-2015---2016.html.)

World Steel News, 2015, MEPS expects a slight recovery in global steel production in 2016: World Steel News, November 11. (Accessed January 27, 2016, at http://www.worldsteelnews.com/meps-expects-a-slight-recovery-in-global-steel-production-in-2016/.)

GENERAL SOURCES OF INFORMATION

U.S. Geological Survey Publications

Iron. Ch. in United States Mineral Resources, Professional Paper 820, 1973.

Iron and Steel. Ch. in Mineral Commodity Summaries, annual.

Iron and Steel (Fe). Ch. in Metal Prices in the United States Through 2010, Scientific Investigations Report 2012–5188, 2013.

Iron and Steel Scrap. Mineral Industry Surveys, monthly.

Iron and Steel Scrap (Fe). Ch. in Metal Prices in the United States Through 2010, Scientific Investigations Report 2012–5188, 2013.

Iron and Steel Slag. Ch. in Mineral Commodity Summaries, annual.

Iron and Steel Slag. Ch. in Minerals Yearbook, annual.

Iron Ore. Ch. in Mineral Commodity Summaries, annual.

Iron Ore. Ch. in Minerals Yearbook, annual.

Iron Ore. Mineral Industry Surveys, monthly.

OTHER

American Metal Market, daily.

Annual Statistical Report, American Iron and Steel Institute, Washington, DC.

Direct from Midrex quarterly, Midrex Direct Reduction Corporation.

Directory of Iron and Steel Plants, Association of Iron and Steel Engineers, Pittsburgh, PA.

HYL, the Iron & Steel Technology Division of HYLSA, S.A. de C.V. HYLReport.

Iron and Steel. Ch. in Mineral Facts and Problems, U.S. Bureau of Mines Bulletin 675, 1985.

Iron and Steelmaker, American Institute of Mining, Metallurgical, and Petroleum Engineers—Iron and Steel Society, Warrenton, PA.

Making, Shaping, and Treating of Steel, Association of Iron and Steel Engineers, Pittsburgh, PA.

Metal Bulletin, weekly.

Steel Manufacturers Association, Washington, DC.

Steel Statistical Yearbook, International Iron and Steel Institute, Brussels, Belgium.

Steel Times International.

APPENDIX C

TABLES (US AND GLOBAL)

SOURCE: *IRON AND STEEL*/MICHAEL D. FENTON/USGS MINERALS YEARBOOK 2014

RAW STEEL: WORLD PRODUCTION, BY COUNTRY[1, 2, 3]

(Thousand metric tons)

Country[4]	2010	2011	2012	2013	2014
Albania	390[e]	464	382	401	400
Algeria	696	440	500	440	415
Argentina	5,138	5,655	4,995	5,188	5,488
Australia	7,140	6,538	4,904	4,640	4,607
Austria	7,206	7,474	7,421	7,400[e]	7,876
Azerbaijan	129	234	268	223[r]	288
Belarus	2,672	2,779	2,869	2,395	2,513
Belgium	7,973	8,026	7,386	7,093	7,331
Bosnia and Herzegovina, ingot production	591	649	700	722	793
Brazil[5]	32,928[r]	35,220[r]	34,524[r]	34,163[r]	33,897
Bulgaria	740	834	633	522	612
Burma[e]	25	25	25	25	25
Canada	13,003	12,891	13,507	12,400[e]	12,730
Chile[5]	1,011	1,615	1,683	1,321	1,079

China[6]	637,230	685,280	716,540	779,040	822,698
Colombia	1,209	1,290	1,324	1,297	1,208
Croatia	103	163	1	135[r]	167
Cuba	280[r]	282[r]	279[r]	267[r]	258
Czech Republic	5,180	5,583	5,072	5,171	5,360
Ecuador	368	525	479	562	662
Egypt	6,700	6,486	6,600	6,754	6,495
El Salvador	64	100	72	72[e]	121
Ethiopia, all from scrap[e]	150	130	130	130	130
Finland	4,029	3,989	3,759	3,517	3,807
France	15,416	15,800	15,607	15,685	16,143
Germany	43,830	44,284	42,661	42,641	42,943
Greece	1,839	1,993	2,000	2,000[e]	1,022
Guatemala	274	294	334	385[r]	395
Hungary	1,678	1,733	1,543	883[r]	1,152
India	68,976	73,471	77,561	81,213	86,530
Indonesia	3,700	3,600	2,300	2,644[r]	4,428
Iran[e]	12,000	13,000	14,500	15,400	16,331
Israel[e]	430	430	430	430	430
Italy	25,751	28,700	27,227	24,058	23,714
Japan	109,599	107,601	107,232	110,571	110,666
Jordan[e]	150	150	100	100	100
Kazakhstan	3,338	3,699	2,607	2,800[e]	3,681
Kenya[e]	260	290	290	290	410
Korea, North[e]	1,300	1,300	1,300	1,300	1,300
Korea, Republic of	58,914	68,519	39,321	81,213	71,036
Latvia[e]	655	515	800	NA	NA
Libya	825	100	300	715	968

Luxembourg	2,563	2,521	2,232	2,090[r]	2,193
Macedonia	291	386	225	146	145
Malaysia	5,693	5,941	5,612	5,700[e]	5,700
Mexico	16,870[r]	18,110[r]	18,073[r]	18,420	18,995
Moldova	242	321	317	190	344
Montenegro	48	61	28	20[e]	30
Morocco	455	460	475	558	502
Netherlands	6,651	6,900	6,867	6,713	6,964
New Zealand	853	844	912	900[e]	881
Norway	514	620	590	605	600
Pakistan	415[r]	358[r]	198[r]	165[r]	142
Paraguay[e]	59	30	44[7]	45	47
Peru	880	877	981	1,090	1,078
Philippines	1,050	1,200	1,260[r]	1,308[r]	1,196
Poland	7,996	8,777	8,543	8,198	8,558
Portugal	1,351	1,942[r]	1,960[r]	2,050[r]	2,070
Qatar	1,705	1,821	2,100	2,236	3,047
Romania	3,896	3,811	3,417	3,071	3,158
Russia	66,800	68,100	68,500[e]	69,400	71,461
Saudi Arabia[e]	5,000	5,300	5,200	5,400	6,291
Serbia	1,254	1,324	346[8]	396	583
Singapore[e]	728	650	700	700	700
Slovakia	4,580	4,236	4,250	4,000[e]	4,705
Slovenia	566	706	600	600[e]	615
South Africa	7,617	7,546	6,938	7,200[e]	6,550
Spain	16,343	15,591	15,600[e]	15,600[e]	14,249
Sweden	4,844	4,866	4,326	4,404	4,539
Switzerland	1,320	1,400	1,500[e]	1,500[e]	1,475
Syria	63	63	10	10[e]	5
Taiwan	18,975	21,927	19,927	21,466	23,121

Thailand	4,145[r]	4,238[r]	3,728[r]	3,579[r]	4,095
Trinidad and Tobago	572	604	624	616	483
Tunisia	115	119	180	180[e]	109
Turkey	29,030	34,103	35,885	34,650	34,035
Uganda[e]	59	65	60	60	60
Ukraine	33,559	35,332	32,912	33,160	27,170
United Arab Emirates[e]	500	2,000	2,800	2,878[7]	2,390
United Kingdom	9,709	9,478	9,819	11,855	12,120
United States	80,500	86,400	88,700	86,900	88,200
Uruguay	65	81	78	90	94
Uzbekistan	731	733	736	740[e]	730
Venezuela	2,207	2,300	2,360	2,250	1,485
Vietnam	4,314	4,900[r]	4,298[r]	5,774[r]	5,847
Zimbabwe[e]	14	15	15	15	15
Total	1,430,000	1,520,000	1,510,000[r]	1,620,000	1,670,000

[e] Estimated. [r] Revised. NA Not available.

[1] World totals, U.S. data, and estimated data are rounded to no more than three significant digits; may not add to totals shown.

[2] Steel formed in solid state after melting, suitable for further processing or sale; for some countries, includes material reported as "liquid steel," presumably measured in the molten state prior to cooling in any specific form.

[3] Includes data available through February 10, 2016.

[4] In addition to the countries listed, Hong Kong, Mauritania, Mozambique, and Sri Lanka are known to have steelmaking plants, but available information is inadequate to make reliable estimates of output levels.

[5] Does not include castings.

[6] Figures reported by the State Statistical Bureau that the Government of China considers as official statistical data.

[7] Reported figure.

[8] Smederevo plant closed July 31, 2012.

MATERIALS CONSUMED IN BLAST FURNACES AND PIG IRON PRODUCED[1]

(Thousand metric tons)

Material	2013	2014
Iron oxides:[2]		
Pellets	38,200	37,500
Sinter[3]	5,420	5,360
Total	43,600	42,900
Scrap[4]	1,970	2,840
Coke[2]	8,130	10,000
Pig iron, produced	30,300	29,400

[1] Data are rounded to no more than three significant digits; may not add to totals shown.
[2] Source: American Iron and Steel Institute.
[3] Includes sintered ore and pellet fines, dust, mill scale, and other revert iron-bearing materials; also includes some nodules.
[4] Mainly briquetted turnings and borings, shredded scrap, and so forth; scrap produced at blast furnaces and remelt not included.

DISTRIBUTION OF SHIPMENTS OF STEEL MILL PRODUCTS, BY STEEL TYPE, PRODUCT, AND MARKET[1]

	Quantity (thousand metric tons)		Percentage	
	2013	2014	2013	2014
Shipments by steel type:				
Carbon steel	81,200	83,600	93.82	93.78
Alloy steel	3,140	3,270	3.62	3.67
Stainless steel	2,220	2,270	2.56	2.55
Total	86,600	89,100	100.00	100.00
Steel mill products:				
Ingots, blooms, billets, and slabs	972	1,110	1.12	1.25
Wire rods	2,190	2,150	2.53	2.42
Structural shapes, heavy	5,580	5,570	6.44	6.25
Plates, cut lengths	6,330	6,550	7.31	7.35

Plates, in coils	3,100	2,970	3.58	3.33
Rails	846	902	0.98	1.01
Railroad accessories	241	276	0.28	0.31
Bars, hot-rolled	4,380	4,830	5.06	5.42
Bars, light-shaped	1,760	1,870	2.04	2.10
Bars, reinforcing	6,560	7,000	7.57	7.85
Bars, cold finished	1,260	1,190	1.45	1.33
Pipe and tubing, standard pipe	645	771	0.75	0.87
Pipe and tubing, oil country goods	2,500	2,680	2.89	3.01
Pipe and tubing, line pipe	476	540	0.55	0.61
Pipe and tubing, mechanical tubing	624	630	0.72	0.71
Pipe and tubing, pressure tubing	35	34	0.04	0.04
Pipe and tubing, stainless	12	12	0.01	0.01
Pipe and tubing, structural	53	51	0.06	0.06
Pipe for piling	1	--	0.00	--
Wire	376	363	0.43	0.41
Tin mill products, blackplate	57	30	0.07	0.03
Tin mill products, tinplate	1,370	1,300	1.59	1.46
Tin mill products, tin free steel	347	333	0.40	0.37
Tin mill products, tin coated sheets	91	88	0.11	0.10
Sheets, hot-rolled	19,600	20,600	22.59	23.14
Sheets, cold-rolled	10,400	10,200	12.07	11.45
Sheets and strip, hot dip galvanized	13,300	14,000	15.41	15.74
Sheets and strip, electrogalvanized	1,390	1,230	1.60	1.38
Sheets and strip, other metallic coated	1,440	1,260	1.66	1.41
Strip, hot-rolled	40	40	0.05	0.04
Strip, cold-rolled	558	494	0.64	0.55
Total	86,600	89,100	100.00	100.00
Shipments by markets:				
Service centers and distributors	23,300	23,100	26.86	25.89
Construction	18,900	20,900	21.83	23.45

Automotive	13,100	13,000	15.12	14.61
Machinery	852	1,000	0.98	1.12
Containers	1,970	1,930	2.28	2.17
All others	28,500	29,200	32.93	32.76
Total	86,600	89,100	100.00	100.00

— Zero.

[1] Data are rounded to no more than three significant digits, except percentages; may not add to totals shown.

Source: American Iron and Steel Institute.

U.S. EXPORTS OF IRON AND STEEL PRODUCTS[1]

(Thousand metric tons)

	2013	2014
Steel mill products:		
Ingots, blooms, billets, and slabs	443	289
Wire rods	144[r]	138
Structural shapes, heavy	1,010	765
Steel piling	19	13
Plates, cut lengths	1,200	1,330
Plates, in coils	644	568
Rails, standard	107	122
Rails, other	85	65
Railroad accessories	44	41
Bars, hot-rolled	552	577
Bars, light-shaped	101	88
Bars, concrete reinforcing	494	483
Bars, cold-finished	150	158
Tool steel	99	98
Pipe and tubing, standard pipe	74	89
Pipe and tubing, oil country goods	397	492
Pipe and tubing, line pipe	526	296

Pipe and tubing, mechanical tubing	103	99
Pipe and tubing, stainless	45	46
Pipe and tubing, nonclassified	372	351
Pipe and tubing, structural	259	232
Pipe for piling	17	20
Wire	163	164
Tin mill products, blackplate	4	7
Tin mill products, tinplate	125	130
Tin mill products, tin free steel	12	13
Sheets, hot-rolled	941	895
Sheets, cold-rolled	896	870
Sheets and strip, hot-dip galvanized	1,200	1,250
Sheets and strip, electrogalvanized	351	338
Sheets and strip, other metallic coated	266	289
Sheets and strip, electrical	102	88
Strip, hot-rolled	257	207
Strip, cold-rolled	306	318
Total	11,500	10,900
Fabricated steel products:		
Structural shapes, fabricated	440	414
Rails, used	4	2
Railroad products	152	201
Wire rope	18	21
Wire, stranded products	39	37
Wire, other products	114	147
Springs	190	176
Nails and staples	24	26
Fasteners	431	474
Chains and parts	44	49
Grinding balls	135	144
Pipe and tube fittings	44	40

Other[2]	240	328
Total	1,880	2,060
Grand total	13,400	13,000
Cast iron and steel products:		
Cast steel pipe fittings	23	23
Cast iron pipe and fittings	90	136
Cast steel rolls	1	(3)
	2013	2014
Cast iron and steel products—Continued:		
Cast grinding balls[4]	47	54
Granules, shot and grit[5]	36	41
Other castings	86	88
Total	283	342

[r] Revised.

[1] Data are rounded to no more than three significant digits; may not add to totals shown.

[2] Includes shapes cold formed, sashes and frames, fence and sign post, architectural and ornamental work, and conduit.

[3] Less than ½ unit.

[4] Cast grinding balls [Harmonized Tariff Schedule of the United States (HTS) code 7325.91.000].

[5] Granule, shot and grit (HTS code 7205.10.000).

Sources: American Iron and Steel Institute and the U.S. International Trade Commission.

U.S. IMPORTS OF IRON AND STEEL PRODUCTS[1]

(Thousand metric tons)

	2013	2014
Steel mill products:		
Ingots, blooms, billets, and slabs	6,640	9,610
Wire rods	797	1,470
Structural shapes-heavy	519	829
Steel piling	115	134
Plates, cut lengths	923	1,690
Plates, in coils	1,070	2,020
Rails and railroad accessories	345	338

Bars, hot-rolled	1,620	1,490
Bars, light-shaped	150	179
Bars, reinforcing	1,100	1,310
Bars, cold-finished	287	348
Tool steel	137	159
Pipe and tubing, standard pipe	793	823
Pipe and tubing, oil country goods	2,990	3,640
Pipe and tubing, line pipe	2,150	2,140
Pipe and tubing, mechanical tubing	532	678
Pipe and tubing, pressure tubing	56	68
Pipe and tubing, stainless	118	144
Pipe and tubing, nonclassified	17	23
Pipe and tubing, structural	382	467
Pipe for piling	15	15
Wire	647	736
Tin mill products, blackplate	39	74
Tin mill products, tinplate	487	633
Tin mill products, tin free steel	156	166
Sheets, hot-rolled	2,710	3,940
Sheets, cold-rolled	1,430	2,680
Sheets and strip, hot-dip galvanized	1,880	3,010
Sheets and strip, electrogalvanized	104	141
Sheets and strip, other metallic coated	653	926
Sheets and strip, electrical	86	67
Strip, hot-rolled	75	100
Strip, cold-rolled	168	175
Total	29,200	40,200
Fabricated steel products:		
Structural shapes, fabricated	828	897
Rails, used	86	73

	2013	2014
Railroad products	169	238
Wire rope	133	141
Wire-stranded products	262	295
Wire, other products	178	188
Springs	345	391
Nails and staples	560	598
Fasteners	1,080	1,140
Chains and parts	127	143
Grinding balls	85	109
Pipe and tube fittings	396	428
Other[2]	492	516
Total	4,740	5,160
Grand total	33,900	45,400
Cast iron and steel products:		
Cast steel pipe fittings	140	142
Cast iron pipe and fittings	35	45
Cast steel rolls	13	13
Cast iron and steel products—Continued:		
Cast grinding balls[3]	9	12
Granules, shot and grit[4]	24	23
Other castings	270	288
Total	491	523

See footnotes at end of table.

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[1] Data are rounded to no more than three significant digits; may not add to totals shown.

[2] Includes shapes cold formed, sashes and frames, fence and sign post, architectural and ornamental work, and conduit.

[3] Cast grinding balls [Harmonized Tariff Schedule of the United States (HTS) code 7325.91.000].

[4] Granule, shot and grit (HTS code 7205.10.000).

Sources: American Iron and Steel Institute and the U.S. International Trade Commission.

U.S. IMPORTS OF STAINLESS STEEL[1]

(Metric tons)

Product	2013	2014
Semifinished	124,000	144,000
Plate	161,000	105,000
Sheet and strip	338,000	428,000
Bars and shapes	126,000	152,000
Wire and wire rods	68,900	80,000
Pipe and tube	118,000	144,000
Total	935,000	1,050,000

[1] Data are rounded to no more than three significant digits; may not add to totals shown.

Source: American Iron and Steel Institute.

COAL AND COKE AT COKE PLANTS[1, 2]

(Thousand metric tons)

	2013	2014
Coal, consumption	19,500	18,500
Coke:[3]		
Production	13,900	NA
Exports	762	NA
Imports	125	NA
Consumption, apparent	13,000	NA

NA Not available.

[1] Data are rounded to no more than three significant digits.

[2] Includes furnace and merchant coke plants.

[3] Coke production and consumption do not include breeze.

Source: U.S. Energy Information Administration, Quarterly Coal Report, DOE/EIA-0121(2015/02Q).

PIG IRON AND DIRECT-REDUCED IRON: WORLD PRODUCTION, BY COUNTRY[1, 2, 3, 4]

(Thousand metric tons)

Country[5]	2010	2011	2012	2013	2014
Algeria	696	360	350	360[e]	360[e]
Argentina:					
Direct-reduced iron	1,566	1,650	1,606	1,466	1,663
Pig iron	2,532	2,795	2,078	2,650	2,766
Australia	6,259	5,369	3,480	3,435	3,282
Austria	5,621	5,815	5,751	6,152	6,029
Belgium	4,688	4,725	4,072	4,343	4,388
Bosnia and Herzegovina	621	685	750	759	860
Brazil	30,878[r]	33,243[r]	30,745[r]	30,000[r]	27,016
Burma[e]	2	2	2	2	2
Canada:					
Direct-reduced iron	600	262	319	1,250[e]	1,550
Pig iron	7,666	7,323	7,654	6,079	6,728
Chile	635	1,072	1,065	775	584
China[6]	597,330	640,510	657,900	708,970	711,600
Colombia	327	295	345	300	234
Czech Republic	3,987	4,137	3,936	4,041	4,152
Egypt:					
Direct-reduced iron[e]	2,965	2,932	3,068	3,432	2,882
Pig iron	600	600	550	600[e]	550[e]
Finland[e]	10,033[7]	12,145[7]	12,000	12,000	12,000
France	10,137	9,700[e]	9,531	10,276	10,866
Germany:					
Direct-reduced iron[e]	450	380	560	500[r]	570

Pig iron	28,560	27,943	27,048	26,910	27,379
Hungary	1,325	1,317	1,229	628	801
India:					
Direct-reduced iron	24,831	21,252	19,677	14,637	20,366
Pig iron	39,560	43,624	47,969	50,256	55,166
Indonesia, direct-reduced iron[e]	1,360	1,230	520	760[r]	120
Iran:[e]					
Direct-reduced iron	9,400	10,400	11,600	14,500	14,551
Pig iron	2,500	2,500	3,000	2,010	2,782
Italy	8,549	9,800	9,418	6,935	6,371
Japan	82,283	81,028	81,405	83,849	83,872
Kazakhstan	2,984	3,141	2,707	2,850[e]	3,185
Korea, North[e]	900	900	900	900	900
Korea, Republic of	31,228	42,213	41,734	40,855	46,909
Libya, direct-reduced iron	1,270	165	508	950	998
Malaysia, direct-reduced iron	2,390	2,876	2,329	2,500[e]	2,400
Mexico:					
Direct-reduced iron	5,368	5,854	5,587	6,134	5,976
Pig iron	4,707	4,607[r]	4,611[r]	4,911[r]	5,116
Morocco[e]	15	15	15	15	15
Netherlands[8]	5,799	5,943	5,909	5,685	5,868
New Zealand[e]	667	659	669	680	670
Norway[e]	100	100	100	100	102

Oman, direct-reduced iron	--	1,110	1,460	1,470	1,450
Pakistan[e]	483	433	249	216	217
Paraguay	81[e]	42	67	69[r]	71
Poland	3,638	3,975	3,944	4,011	4,637
Portugal[e]	100	100	100	100	100
Qatar, direct-reduced iron	2,157	2,230	2,420	2,390	2,549
Romania	1,726	1,595	1,450	1,600	1,631
Country[5]	2010	2011	2012	2013	2014
Russia:[e]					
Direct-reduced iron	4,700	4,900	5,200	5,200	5,200
Pig iron	48,000	48,000	50,500	51,000	51,474[7]
Saudi Arabia, direct-reduced iron[e]	5,500	5,800	5,700	6,000	6,460
Serbia	1,265	1,226	312[9]	365[e]	550
Slovakia	3,649	3,346	3,519	3,617	3,838
South Africa:					
Direct-reduced iron	1,120	1,414	1,493	1,410[e]	1,560
Pig iron	5,429	4,604	4,599	4,930[e]	4,690
Spain	3,572	3,540	3,081	3,949	3,958
Sweden	3,447	3,240	5,253	2,896[r]	3,078
Taiwan	9,358	12,940	11,800	13,400[e]	14,440
Trinidad and Tobago, direct-reduced iron	1,752	1,706[r]	1,684	1,750	1,633
Turkey[e]	7,677[7]	8,200	8,600	9,200	9,364
Ukraine	27,361	28,881	28,514	29,089	24,787
United Kingdom	7,235	6,600	7,252	9,512	9,705

United States	26,800	30,200	32,100	30,300	29,400
Venezuela, direct-reduced iron	3,793	4,470	4,472	2,571	1,402
Total	1,110,000	1,180,000	1,200,000[r]	1,250,000[r]	1,260,000
Of which:					
Direct-reduced iron[10]	69,200	68,600[r]	68,200	66,900	71,300
Pig iron[11]	1,040,000	1,110,000	1,130,000[r]	1,180,000	1,190,000

[e] Estimated. [r] Revised. — Zero.

[1] World totals, U.S. data, and estimated data are rounded to no more than three significant digits; may not add to totals shown.

[2] Production is pig iron unless otherwise specified.

[3] Direct-reduced iron is obtained from ore by reduction of oxides to metal without melting.

[4] Does not include ferroalloy production except where otherwise noted. Includes data available through April 28, 2016.

[5] Vietnam may have produced limited quantities of pig iron during 2010–14, but output is not reported and available information is inadequate to make reliable estimates of output levels.

[6] Figures reported by State Statistical Bureau that the Government of China considers to be official statistical data.

[7] Reported figure.

[8] Includes blast furnace ferroalloys.

[9] Smederevo plant closed July 31, 2012.

[10] Listed separately.

[11] Includes unspecified pig iron and direct-reduced iron.

APPENDIX D

STANDARD STEEL SPECIFICATIONS FOR CONSTRUCTION OF ROADS AND BRIDGES ON FEDERAL HIGHWAY PROJECTS FP-14 (2014)

Source: U.S. Department of Transportation

Federal Highway Administration

Federal Lands Highway

Section 707 672

Section 707. —METAL PIPE

707.01 Ductile Iron Culvert Pipe. Conform to ASTM A716 for the sizes specified.

707.02 Metallic-Coated Corrugated Steel Pipe. Furnish pipe, special sections (such as elbows, branch connections, and prefabricated flared end sections) and coupling bands conforming to AASHTO M 36 and either AASHTO M 218, AASHTO M 274, or AASHTO M 289 for the dimensions and thicknesses specified.

Fabricate underdrain pipe from steel sheets with a minimum thickness of 0.052 inches (1.32 millimeters). Use any class of perforation specified in AASHTO M 36.

707.03 Aluminum-Alloy Corrugated Pipe. Furnish pipe, special sections (such as elbows, branch connections, and prefabricated flared end sections) and coupling bands conforming to AASHTO M 196 for the dimensions and thicknesses specified.

Fabricate underdrain pipe from aluminum sheets with a minimum thickness of 0.048 inches (1.22 millimeters). Use any class of perforation.

707.04 Asphalt-Coated Pipe. Furnish pipe, special sections (such as elbows, branch connections, and prefabricated flared end sections), and coupling bands conforming to Section 707 as applicable for the kinds of pipes to be coated.

Coat the pipe with asphalt material conforming to AASHTO M 190 for the type of coating specified. Coat special sections (such as elbows, branch connections, and end sections) and coupling bands according to AASHTO M 190. Coat flared end sections with an asphalt coating conforming to AASHTO M 190, Type A or a field applied asphalt mastic coating conforming to AASHTO M 243.

707.05 Steel Structural Plate Structures. Furnish structures and assembly fasteners for connecting plates conforming to AASHTO M 167 for the sizes and types specified.

707.06 Aluminum-Alloy Structural Plate Structures. Furnish structures and assembly fasteners for connecting plates conforming to AASHTO M 219 for the sizes and types specified.

707.07 Asphalt-Coated Structural Plate Structures. Furnish structures conforming to either Subsection 707.05 or 707.06 as applicable. Coat with an asphalt coating conforming to AASHTO M 190, Type A or a field applied asphalt mastic coating conforming to AASHTO M 243.

If the asphalt coating is applied to the plates before field erection, identify each plate's nominal metal thickness by painting the data on the inside surface of the plates after coating. Other methods of plate identification may be used if approved.

707.08 Polymer-Coated Steel Pipe. Furnish pipe, special sections (such as elbows and branch connections) and coupling bands conforming to AASHTO M 245, Grade 250/250 and AASHTO M 246, Grade 250/250.

707.09 Reserved. Section 707 673.

Page 707.10 Slotted Drain Pipe. Furnish pipe conforming to AASHTO M 36 and either AASHTO M 218, AASHTO M 274, or AASHTO M 289 for the dimensions and thicknesses specified.

Fabricate the pipe with either angle slots or grate slots and as shown in the plans.

Furnish grate assemblies for the grate slot drain conforming to ASTM A1011, SS Grade 36. Galvanize slot angles and grate slot assemblies according to Subsection 725.10.

707.11 Metallic-Coated Spiral Rib Pipe. Furnish pipe, special sections (such as elbows and branch connections), and coupling bands conforming to AASHTO M 36, Type IR and Type IIR, AASHTO M 218, AASHTO M 274, or AASHTO M 289 for the dimensions and thicknesses specified.

707.12 Aluminum-Alloy Spiral Rib Pipe. Furnish pipe, special sections (such as elbows and branch connections) and coupling bands conforming to AASHTO M 196, Type IR and Type IIR for the dimensions and thicknesses specified.

707.13 Concrete-Lined Corrugated Steel Pipe. Furnish pipe, special sections (such as elbows and branch connections), and coupling bands conforming to Subsection 707.02 for the dimensions and thicknesses specified. Fully line the pipe and special sections with concrete according to ASTM A849, Class C.

707.14 Invert-Paved Corrugated Steel Pipe. Furnish pipe, special sections (such as elbows and branch connections), and coupling bands conforming to Subsection 707.02 for the dimensions and thicknesses specified. Pave the invert of the pipe and special sections with concrete or asphalt material according to ASTM A849, Class A or Class C.

707.15 Cast Iron Soil Pipe and Fittings. Conform to ASTM A74, Class SV for the designated sizes.

707.16 Seamless Copper Water Tube and Fittings. Conform to ASTM B88, Type L for the designated sizes.

707.17 Gaskets for Metal Pipe.

(a) O-ring gaskets for flexible metal pipe. Conform to ASTM C1619, Class C.

(b) Continuous flat gaskets for flexible metal pipe with flat bands or bands with projections.

(1) Conform to ASTM D1056.

(2) Gasket thickness ½ in (13 mm) greater than the nominal depth of pipe corrugations

(c) Continuous flat gaskets for flexible metal pipe with corrugated bands.

(1) Conform to ASTM D1056.

(2) Gasket thickness 3/8 in (10 mm)

707.18 Gaskets for Ductile Iron Pipe. Conform to ASTM A746.

Section 709. —REINFORCING STEEL AND WIRE ROPE

709.01 Reinforcing Steel.

(a) General. Furnish the following information with each shipment of steel to the project:

(1) Name and location of the steel rolling mill;

(2) Manufacturing process;

(3) Heat numbers;

(4) Sizes;

(5) Specifications;

(6) Copies of mill test analyses for chemical and physical tests; and

(7) Consignee and destination of shipment.

(b) Reinforcing bars. Furnish deformed, Grade 60 (420) bars conforming to AASHTO M 31 or AASHTO M 322.

(c) Epoxy-coated reinforcing bars. Furnish bars conforming to Subsection 709.01(b). Conform to ASTM A775.

Inspect the reinforcing bars after the near white blast cleaning. Reject bars with steel slivers or scabs. Selective sorting and rejection at the fabricator's shop may avoid unnecessary delays and subsequent rejection of bars during the precoating inspection at the coating applicator's shop.

Coat epoxy-coated reinforcing steel in a plant certified by CRSI as a fusion bonded epoxy applicator.

(d) Tie bars. Furnish deformed, Grade 60 (420) bars conforming to AASHTO M 31.

(e) Hook bolts. Furnish plain, Grade 60 (420) bars conforming to AASHTO M 31 with M14 rolled threads or M16 cut threads. Furnish a threaded sleeve nut capable of sustaining a minimum axial load of 15,000 pounds (67 kilonewtons).

(f) Dowel bars. Conform to AASHTO M 254, Type A or Type B. Use plain round bars, without burring or other deformation restricting free movement in the concrete. Paint half the length of each dowel bar with one coat of tar paint. When the paint dries and immediately before placing the dowels, lubricate the painted end to prevent concrete from bonding to the painted end.

For expansion joints, furnish a dowel cap that snugly covers 2±¼ inches (50±6 millimeters) of the dowel, has a closed end, and has a suitable stop to hold the closed end 1 inch (25 millimeters) from the end of the dowel bar.

Lubricants for Type B dowels may be medium setting emulsified asphalt or a flaked graphite. Lubricants are not required for Type A coated dowel bars.

Furnish dowel assemblies that hold dowel bars within ¼-inch (6-millimeter) tolerance vertically and horizontally during concrete placement and permit unrestricted movement of the pavement slab. Section 709 677

Use wire conforming to AASHTO M 32 for dowel assemblies. Coat dowel assemblies with the same material as the dowel bar. Recoat or repair damaged coatings equivalent to the manufacturer's original coating.

(g) Deformed steel wire. Conform to AASHTO M 225.

(h) Steel welded wire reinforcement, plain, for concrete. Conform to AASHTO M 55.

(i) Cold-drawn steel wire. Conform to AASHTO M 32.

(j) Welded deformed steel wire fabric. Conform to AASHTO M 221.

(k) Fabricated deformed steel bar or rod mats. Conform to AASHTO M 54.

(l) Low alloy steel deformed bars. Conform to ASTM A706.

709.02 Prestressing Steel.

(a) Fabricate from one of the following:

(1) Stress-relieved steel wire, AASHTO M 204, Type BA or Type WA;

(2) Uncoated seven-wire steel strand, AASHTO M 203, Grade 270 (1860); or

(3) High-strength steel bars, AASHTO M 275, Type II

(b) Submit representative samples from prestressed members fabricated off site. In the case of wire or strand, take the sample from the same master roll.

(1) Pretensioning method. Furnish a sample at least 6 feet (1.8 meters) long of each strand size from each coil.

(2) Post-tensioning method. Furnish samples of the following lengths.

(a) For wires requiring a head, 15 feet (5 meters).

(b) For wires not requiring a head, sufficient length to make up one parallel-lay cable 5 feet (1.5 meters) long consisting of the same number of wires as the cable to be furnished.

(c) For strands furnished with fittings, 5 feet (1.5 meters) between near ends of fittings.

(d) For bars to be furnished with threaded ends and nuts, 5 feet (1.5 meters) between threads at ends.

Section 717. —STRUCTURAL METAL

717.01 Structural Steel.

(a) Structural carbon steel. Conform to the following:

(1) Primary bridge members AASHTO M 270, Grade 36T (250T)

(2) Fracture critical bridge members AASHTO M 270, Grade 36F (250F)

(3) Other shapes, plates, and bars AASHTO M 270, Grade 36 (250)

(b) High-strength low-alloy structural (HSLA) steel. Conform to the following:

(1) Primary bridge members AASHTO M 270, Grade 50T, 50ST, and welded members 50WT, or HPS 50WT (345T, 345ST, 345WT, or HPS 345WT)

(2) Fracture critical bridge members AASHTO M 270, Grade 50F, 50SF, and fracture critical welded members 50WF, or HPS 50WF (345F, 345SF, 345WF, or HPS 50SF)

(3) Other shapes, plates, and bars AASHTO M 270, Grade 50, 50S, 50W, or HPS 50 (345, 345S, 345W, or HPS 345)

(c) High-strength quenched and tempered (QT) steel. Conform to the following:

(1) Primary bridge members AASHTO M 270, Grade 70WT, HPS 70WT, 100T, 100WT, or HPS 100WT (485WT, HPS 485WT 690T, 690WT, or HPS 690WT)

(2) Fracture critical bridge members AASHTO M 270, Grade 70WF, HPS 70WF, 100F, 100WF, or HPS 100WF (485WF, HPS 485WF 690F, 690WF, or HPS 690WF)

(3) Other shapes, plates, and bars AASHTO M 270, Grade 70W, HPS 70W, 100, 100W, or HPS 100W (485W, HPS 485W, 690, 690W, or HPS 690W)

(4) Hollow structural sections ASTM A500, Grade B, ASTM A501, ASTM A847, or ASTM A618

(d) Bolts and nuts. Conform to ASTM F1554, Grade 36 (250) or ASTM A307 as shown in the plans. Furnish nuts conforming to ASTM A563 for appropriate grade and size of anchor bolt. Heat treat nuts to be galvanized to Grade DH or Grade DH3. Lubricate galvanized nuts with a lubricant containing a visible dye. Section 717 709

(e) High-strength bolts, nuts, and washers. Conform to ASTM A325 or ASTM A490 as specified. Use Type 1 bolts with steels other than weathering steel. Use Type 3 bolts with weathering steel.

ASTM A325, Type 1 bolts may be either hot-dip galvanized according to AASHTO M 232, Class C or mechanically galvanized according to ASTM B695, Class 50. Retest galvanized bolts after galvanizing according to ASTM A325. Do not galvanize ASTM A490 bolts.

Galvanize washers, nuts, and bolts of any assembly by the same process. Overtap the nuts to the minimum required for the fastener assembly and lubricate with a lubricant containing a visible dye.

Except as noted below, for ASTM A325 bolts, use nuts conforming to ASTM A563, Grades DH, DH3, C, C3, or D. For ASTM A490 bolts, use nuts conforming to ASTM A563, Grades DH or Grade DH3. Heat treat nuts to be galvanized to Grade DH. Lubricate galvanized nuts with a lubricant containing a visible dye.

Provide plain nuts with a minimum hardness of 89 HRB. Use only Grade C3 or Grade DH3 nuts with ASTM A325, Type 3 bolts. Use only Grade DH3 nuts with ASTM A490, Type 3 bolts.

Use washers conforming to ASTM F436.

717.02 Steel Forgings. Conform to AASHTO M 102, Classes C, D, F, and G.

717.03 Pins and Rollers. Furnish pins and rollers more than 9 inches (230 millimeters) in diameter from annealed carbon steel forgings conforming to AASHTO M 102, Class C.

Furnish pins and rollers 9 inches (230 millimeters) or less in diameter from either annealed carbon steel forgings conforming to AASHTO M 102, Class C or cold finished carbon steel shafting conforming to AASHTO M 169, Grade 1016 to Grade 1030 inclusive, with a minimum Rockwell Scale B hardness of 85. The hardness requirement may be waived if the steel develops a tensile strength of 70,000 pounds per square inch (480 megapascals) and a yield point of 36,000 pounds per square inch (250 megapascals).

For pin threads, conform to ANSI B1.1, *Unified Inch Screw Threads (UN and UNR Thread Form)*, Class 2A. Thread pin ends with a diameter of 1$^3/_8$ inches (35 millimeters) or more with 6 threads to the inch (25 millimeters).

717.04 Castings.

(a) **Steel castings.** Conform to AASHTO M 103, Grade 70-36 (485-250).

(b) **Chromium alloy steel castings.** Conform to AASHTO M 163, Grade CA15.

(c) **Gray iron castings.** Conform to AASHTO M 105, Class No. 30B, unless otherwise specified. Make the castings without pouring faults, sponginess, cracks, blow holes, and other defects in positions affecting strength and value for the service intended. Boldly fillet the castings at angles and make the arrises sharp and perfect. Sand blast castings or otherwise effectively remove the scale and sand to present a smooth, clean, and uniform surface.

(d) **Malleable iron castings.** Conform to ASTM A47, Grade 35018, unless otherwise specified. For workmanship, finishing, and cleaning, conform to Subsection 717.04(c). Section 717 710

717.05 Welded Stud Shear Connectors. Conform to AASHTO M 169 and Article 11.3.3, *Welded Stud Shear Connectors* of AASHTO, *LRFD Bridge Construction Specification*.

717.06 Steel Pipe. Furnish galvanized steel pipe conforming to ASTM A53, Type F, standard weight class, and plain ends for the designation specified in the contract.

717.07 Galvanized Coatings. When specified, galvanize structural steel according to AASHTO M 111.

717.08 Sheet Lead. Furnish common desilverized lead conforming to ASTM B29. Furnish sheets in a uniform 0.125±0.030-inch (6±1-millimeter) thickness without cracks, seams, slivers, scale, and other defects.

717.09 Steel Grid Floors. Conform to ASTM D5484, Type I. Furnish galvanized steel grid floors unless painting is specified.

717.10 Bearings.

(a) **Elastomeric bearings, plain or laminated.** Conform to AASHTO M 251.

(b) **High load rotational spherical bearings.** Conform to ASTM D5977.

717.11 Polytetrafluoroethylene (PTFE) Surfaces for Bearings. When PTFE surfaces are specified for bearings, which are not listed in Subsection 717.10, conform to the following:

(a) **PTFE resin.** Furnish virgin PTFE resin material conforming to ASTM D4894 or ASTM D4895.

(b) **Filler material.** Furnish milled glass fibers, carbon, or other approved inert material.

(c) **Adhesive material.** Furnish epoxy resin adhesive conforming to AASHTO M 235.

(d) **Unfilled PTFE sheet.** Furnish unfilled PTFE sheet made from PTFE resin. Conform to Table 717-1.

(e) **Filled PTFE sheet.** Furnish filled PTFE sheet made from PTFE resin uniformly blended with filler material. Do not exceed 15 percent filler content using fiberglass or 25 percent filler content using carbon fibers. For filled PTFE sheets containing glass fibers or carbon, conform to Table 717-1.

(f) **Fabric containing PTFE fibers.** Furnish fabric made from oriented multifilament PTFE fluorocarbon and other fibers or from a mixture of PTFE fibers made from twisted, slit PTFE tape and other fibers as required by proprietary designs. Conform to Table 717-1. Section 717 711

Table 717-1

Polytetrafluoroethylene Sheeting

Property	ASTM Method	Sheet Unfilled	Sheet with 15% Glass Fibers	Sheet with 25% Carbon Fibers	Woven Fabric
Min. tensile strength	D638 or D2256	2,800 lb/in2 (19 MPa)	2,000 lb/in2 (14 MPa)	1,300 lb/in2 (9 MPa)	24,000 lb/in2 (165 MPa)
Min. elongation	"	200%	150%	75%	35%
Min. specific gravity	D792	2.16±0.03	2.20±0.03	2.10±0.03	–
Melting point	D4591	623±2 °F (328±11 °C)	621±18 °F (327±10 °C)	621±18 °F (327±10 °C)	–

Section 722. —ANCHOR MATERIAL

722.01 Anchorage Devices. For post-tensioning, furnish anchorage devices capable of holding the prestressing steel at a load producing a stress of not less than 95 percent of the guaranteed minimum tensile strength of the prestressing steel.

Use a steel distribution plate or assembly to effectively distribute the compressive stresses from the anchoring device to the concrete. If the anchorage device is sufficiently large and is used with a steel grillage embedded in the concrete, the distribution plate or assembly may be omitted. Conform to the following:

(a) Do not exceed 3,000 pounds per square inch (21 megapascals) for the final unit compressive stress on the concrete directly beneath the plate or assembly.

(b) Do not allow bending stresses induced by prestressing to exceed the yield point of the material in the plates or assemblies, or cause visible distortion in the anchorage plate when 100 percent of the nominal load is applied.

722.02 Anchor Tendons. Furnish material conforming to the following:

(a) Prestressing steel. Conform to one of the following:

(1) Wire, uncoated stress-relieved for ASTM A421 prestressed concrete

(2) Steel strand, uncoated seven-wire AASHTO M 203 stress-relieved for prestressed concrete

(3) Steel strand, uncoated, seven-wire, ASTM A779 compacted, stress-relieved for prestressed concrete

(4) Uncoated high-strength steel bar for AASHTO M 275 prestressed concrete

(b) Couplers. Supply couplers that are capable of developing 95 percent of the minimum specified ultimate tensile strength of the tendon.

(c) Sheathing. Conform to one of the following:

(1) Free-stressing length.

(a) Polyethylene plastic tubing. Conform to ASTM D1248, Types II, III, or IV with a minimum wall thickness of 60 mils (1.5 millimeters).

(b) Hot-melt extruded polypropylene tubing. Conform to ASTM D4101, cell classification PP 210 B5554211 with a minimum wall thickness of 60 mils (1.5 millimeters).

(c) Hot-melt extruded polyethylene tubing. Conform to ASTM D3350 and ASTM D1248 Type III high-density with a minimum wall thickness of 60 mils (1.5 millimeters). Section 722 731

(d) Steel tubing. Conform to ASTM A500 with a minimum wall thickness of 0.20 inches (5 millimeters).

(e) Steel pipe. Conform to ASTM A53, Schedule 40 minimum.

(f) Plastic pipe. Conform to ASTM D1785, Schedule 40 minimum.

(2) Bonded length.

(a) High-density corrugated polyethylene tubing. Conform to AASHTO M 252 with a minimum wall thickness of 30 mils (0.75 millimeters).

(b) Corrugated, polyvinyl chloride tubes. Conform to ASTM D1784, Class 13464-B.

(c) Fusion-bonded epoxy. Conform to ASTM A775 with a minimum film thickness of 15 mils (0.4 millimeters).

(d) Corrosion inhibiting compounds. Use grease conforming to PTI, *Recommendations for Prestressed Rock and Soil Anchors*.

(e) Centralizers and spacers. Fabricate centralizers and spacers from material, except wood, that are not deleterious to the prestressing steel.

(f) Anchorages. Furnish material conforming to PTI, *Recommendations for Prestressed Rock and Soil Anchors*.

For strand tendons, supply anchorages that permit lift-off testing without the jack engaging the strand.

Furnish steel plates conforming to ASTM A36 or ASTM A588 for bearing plates. Provide grout tube holes in the bearing plates.

722.03 Rock Bolts. Furnish deformed bars (tendon), bearing plates, washer, nuts, and other accessories for tendons conforming to ASTM F432 and as follows:

(a) Tendon. Furnish hollow core bars conforming to ASTM A615. Use minimum Grade 75 (520) steel.

(b) Coupler. Supply couplers that are capable of developing 95 percent of the minimum specified ultimate tensile strength of the tendon.

(c) Bearing plate. Furnish bearing plates with grout tube holes.

(d) Corrosion inhibiting compounds. Use grease conforming to PTI, *Recommendations for Prestressed Rock and Soil Anchors*.

(e) **Centralizers and spacers.** Fabricate centralizers and spacers from any type of material, except wood, that is not deleterious to the tendon.

(f) **Corrosion protection.** Furnish steel with either fushion bonded epoxy coating or hot dip galvanizing for corrosion protection. Use fusion bonded epoxy coating conforming to ASTM A775 with a minimum film thickness of 15 mils (0.4 millimeters). Use hot dip galvanizing conforming to ASTM A153 with a minimum thickness of 3.9 mils (0.1 millimeters). Section 722 732

722.04 Soil Nails.

(a) **Tendon.** Furnish deformed bars conforming to one of the following:

(1) Deformed bars, Grade 60 (420) or Grade 75 (520) ASTM A615.

(2) Deformed bars, Grade 150 (1035) ASTM A722 Provide new, straight, continuous, undamaged, bare, epoxy-coated, or encapsulated bars.

(b) **Coupler.** Furnish couplers that are capable of developing the full, ultimate tensile strength of the tendon as certified by the manufacturer.

(c) **Fusion bonded epoxy coating.** Apply epoxy coating conforming to ASTM A775, with the exception of the bend test requirements. Electrostatically apply the coating to a minimum thickness of 12 mils (0.3 millimeters). The coating at the wall anchorage end of epoxy-coated bars may be omitted over the length provided for threading the nut against the bearing plate.

(d) **Encapsulation.** Furnish corrugated polyethylene pipe conforming to AASHTO M 252 or corrugated polyvinyl chloride pipe conforming to ASTM D1784, Class 13464.

APPENDIX E

INDIAN STEEL INDUSTRY

In This Chapter

- Historical
- World Steel Production
- Alloy Steel Making
- Mini-Steel Plants

31.1. Historical

Historical tradition indicates a highly developed iron and steel culture in India in various areas, particularly near Hyderabad and Tiruchirapally, for more than three thousand years. The famous Iron Pillar near Kutub Minar at Delhi is still considered to be a metallurgical wonder. It still cannot be duplicated in terms of its high corrosion resistance in spite of great strides in iron and steel making technologies of the present time. The swords, daggers, etc., belonging to the various periods of history bear testimony to the skill of the early workers in India. The tradition continued uninterrupted until the last century. The production of molten iron by smelting in small shaft furnaces, using charcoal as a reducer as well as a heat producer, its refining to wrought iron and conversion of wrought iron to steel by a process resembling the cementation process, must have been in general the process of steel making then. The skill and art of making steel was passed on from generation to generation, in typical Indian oral style with scant written records. People knowing this art are still available, and in some very remote areas they make steel in this traditional way even today.

31.1.1. Modern Historical

The first modern ironworks that met with moderate success was established at Barakar in Bihar in 1875. It suffered many reverses and finally passed into the hands of Bengal Iron and Steel Company. The Indian Iron and Steel Company was started in 1918 at Hirapur near Asansol for the production of pig iron alone.

The pioneering efforts of Bengal Iron and Steel Company inspired Jamshedji Tata whose efforts, against many odds, finally culminated in the establishment of the first ever integrated steel plant in India in a place now named after him. This is the Tata Iron and Steel Company, referred to until recently as TISCO. This is now named Tata Steel. This plant at Jamshedpur (railway station Tatanagar) was established in 1908 and it started production in 1911-12.

The Mysore Iron and Steel works at Bhadrawati (Karnataka), now named Visvesvaraya Iron and Steel Ltd. (VISL) started in 1918 with a small charcoal blast furnace. In 1936 it was expanded into an integrated steel plant.

The Bengal Iron and Steel Company and the Indian Iron and Steel Company were amalgamated into an integrated steel plant in 1936 in the name of the latter at Burnpur, known as IISCO.

TISCO, IISCO, VISL, and a few electric arc furnaces were the ones that were making steel in India until 1954. The side-blown converters were used for making steel particularly for World War II purposes at some odd places. The Second Five-Year Plan, with emphasis on large-scale industrialization, started in 1955. The existing steel plants were asked to modernize and expand their steel production capacities.

Hindustan Steel Pvt Ltd. was established by the Government of India to establish steel plants, and accordingly three new steel plants, each of 1 Mt capacity steel ingots were established at each of three places. These were established with foreign collaboration. The Rourkela Steel Plant (RSP) was established by Germans, Bhilai Steel Plant (BSP) by Russians, and Durgapur Steel Plant (DSP) by British collaboration.

Although steel making capacities were to be increased very rapidly after the Second Plan, this did not materialize because of economical constraints, in the subsequent five-year plans. The Bokaro steel plant was to be established in the Third Five-Year Plan (1960-65) but it was delayed far too long; it was finally established with Russian collaboration and started iron production in 1973, and steel production in 1976. It was designed to produce 1.7 Mt of ingot steel in its first stage. It was also envisioned to set up steel plants of 2 Mt capacity each at Vishakhapatnam (Andhra Pradesh), Salem (Tamilnadu), and Hospet (Karnataka); these were also delayed, and only Vishakhapatanam could be started in 1993, and Salem Special Steel Plant in the mid-1980s.

The government of Maharashtra set up a true mini-steel plant at Chandrapur with 33 mVA electric smelting furnace for iron production, and two LD vessels of 15 t capacity each and a two-strand concast machine in the early 1970s. The electric furnace now produces Fe-Mn, and the steel making part has been sold off.

Much of the production from the integrated steel plants is of ordinary bulk steel in the form of flat products, rails, wheels, axles, tires, structural, merchant products, etc. The old plants like Tata Steel and IISCO produce most of the above varieties. The new plants set up after the 1950s, however, were designed to produce not all but a certain type of products, e.g., Rourkela was designed to produce only flat products like plates (more than 7 mm thick), sheets (more than 0.22 mm thick), strips, galvanized sheets, tin plates, etc.; Bhilai was designed to produce rails, structures, etc.; and Durgapur was designed to produce mainly structurais (section products). This had to be so because of some advantages of steel making technologies then available. This is no longer restricted and any steel can be produced by the modern steel making technology and therefore any of the above products can be produced with equal ease.

Having brought under public sector exclusively, steel production was controlled by the government to the extent that even some of the steel products were sold under administered controlled prices, the same anywhere in India. One joint plant committee, JPC, used to oversee this aspect of steel marketing as an integral part of the socialistic pattern of society at that time. Commensurate with this, the Steel Authority of India Ltd was set

Table 31.1

Steel Production Capacities of Various Plants in India (2007)

	Crude Steel Mt
SAIL	
Bhilai	5.0
Rourkela	1.9
Durgapur	1.8
Bokaro	4.5
IISCO	1.0
Rashtriya Ispat Nigam, Vizag	3.5
Alloy steel Plant, Durgapur	0.25
Salem Steel Plant, Salem	0.22
VISL, Bhadravati	0.077
Private Sector	
Tata Steel	5.0
Essar Steel	3.6
Jindal South West(Jindal Vijayanagaram)	1.6
Jindal Stainless	0.05
Jindal Steel and Power	2.9
Ispat Ind	3.0
Saw Pipes	0.95
Mahindra Ugine	0.11
Mukand	0.44
Tata Metallics	0.7
Uttam Steels	0.6
Kalyani Steels	0.3
Electro Steel Castings	0.3
Sesa Goa	0.22
Lloyds Steel Ind	0.72
Usha Martin	0.36

up to control steel production and distribution in India. The public sector steel plants came directly under this authority of the Government of India, since 1971.

In spite of this control, several mini-steel-plants based on scrap-based arc furnaces were set up in the early 1970s and they contributed a fair proportion of steel production in India.

The early 1990s saw a sea change in the policies of the Government of India. Globalization and liberation on the industrial front became the key word in tune with the changing world economic scenario. As a result, setting of new steel plants was freely allowed, i.e., this sector was delicensed.

The present steel output in India is around 60 Mt per annum, out of which almost 35-40% comes from DRI-EAF/IF route on a very small scale.

31.2. World Steel Production

World steel production has increased very rapidly in the first decade of the 21st century, particularly in view of the world trade agreement and overall economic improvements in two of the most populous countries, like China and India. It reached nearly 1340 Mt for the year 2007. Major steel producing countries are shown in Table 31.2 below.

Table 31.2
Major steel producing countries with their capacities (2006)

Country	*Mt/annum*
China	420
Japan	116
USA	98
Russia	71
South Korea	49
Germany	47
Ukraine	40
India	44
Brazil	31
Italy	31
World Total	1237

31.3. Alloy Steel Making

Until recently, the large integrated steel plants were chiefly meant to produce bulk steels. They did produce low alloy steels on a large scale by mere ladle additions. Many had one or two EAFs to cater to high alloy steel demands or for home consumption. The VISL was an exception to this general pattern. It produced chiefly alloy steels.

In order to meet alloy steel demands, Durgapur Alloy Steel Plant (ASP, Durgapur) was established in the public sector during the Second Five-Year Plan with Japanese and Canadian collaboration to produce initially 10,000 t of alloy steels per annum. It now produces 0.4 Mt per annum.

The Salem Steel Plant was established to cater to stainless steel demands.

Alloy steel was mainly produced by small-scale mini-steel plants based on electric furnaces, melting scrap.

Very special requirements of steels for defense, aircraft, space applications, and so on, could not be indigenously met until steel making facilities were established under Mishra Dhatu Nigam, at Hyderabad in the public sector, in the 1960s.

The scenario has since almost completely changed. Now the usual constructional bulk steels are mainly produced by mini-steel plants using steel scrap and/or carbon-based DRI. Only heavy sections are produced by large steel plants. The demand for micro-alloyed steels has increased so much that these are regularly produced by all major steel plants. Low alloy steels are also produced and consumed on a very large scale.

31.4. Mini-Steel Plants

Although a few electric furnaces were in operation in India in the 1960s, a mushroom growth occurred in the 1970s because of the Government of India's changed policies looking at the shortfall in steel production in the late 1960s and early 1970s. The delay in setting up the Bokaro steel plant and others ultimately culminated in this change. The low gestation period for setting up these plants, lower initial investments, adequacy of technical know-how for setting these plants, lower managerial skills requirements, and so on, in fact led to these rapid developments. Although they were doing well in the early years, erratic electric supply in some parts and its costs made their economics precarious. Since the mid-1980s, this sector has been facing a variety of problems, and as a result many have been closed down.

The situation rapidly changed after the liberalization of 1991. Because of an increase in world steel demand, in particular of China for preparing for the 2008 Olympics, and increased domestic steel consumption due to sustained rising Indian GDP since 1991, the EAF and IF furnace-based mini-steel plants once again flourished; literally a few hundred of them are now actively producing steel to meet constructional grade steel requirements, mainly for local consumption and steel castings.

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