THE MANUFACTURE OF IRON AND STEEL

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VOLUME TWO

STEEL PRODUCTION

UNIFORM WITH THIS VOLUME

THE MANUFACTURE OF IRON
AND STEEL

Volume One : Iron Production

THE MANUFACTURE
OF
IRON AND STEEL

By

G. REGINALD BASHFORTH

F.I.M.

VOLUME TWO

STEEL PRODUCTION

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PREFACE TO VOLUME TWO

In the preparation of Volume Two I have endeavoured to review the modern methods of steel production whilst keeping in mind the original objects of this work. The basic principles underlying the various processes, and the plant and equipment employed, have been surveyed as fully as space would permit. Where possible, the text has been illustrated. It is hoped that this survey will provide a suitable textbook for the general student of metallurgy and at the same time prove a handy reference book for the busy industrialist. A fairly extensive list of references is given for those readers who require further information.

As I present this volume I must pay tribute to the numerous friends who have assisted me in so many ways. It is impossible to mention them all by name, but my thanks are especially due to the following persons and organizations for their permission to quote data and to use illustrations which have appeared in their publications:

The Council of the Iron & Steel Institute;
The Council of the Staffordshire Iron & Steel Institute;
The Council of the West of Scotland Iron & Steel Institute;
The Council of the Cleveland Institute of Engineers;
The American Institute of Mining & Metallurgical Engineers;
The British Iron & Steel Research Association;
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G. R. Lewington, Esq. (*British Steelmaker*);
W. E. Benbow, Esq. (*Iron and Steel*);
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Alistair McLeod, Esq. (*Iron and Coal Trades Review*).

I would also express my thanks to my colleagues at Round Oak—Mr. T. K. Pattinson for his help with the proofs and to Messrs. W. A. Akhurst, E. Richards and J. Kerr for their assistance with the illustrations. My sincerest thanks are again recorded to Miss D. M. Sutcliffe for her many helpful suggestions and much constructive criticism, and for her care and diligence in preparing the manuscript.

Stourbridge G. R. B.
8th *June*, 1949.

VOLUME TWO

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CHAPTER 1

THE BESSEMER PROCESS

Historical Note

A great advance in steel-making occurred in 1855, when Henry Bessemer took out his first patent for the manufacture of malleable iron and steel by blowing a current of air through molten pig iron. The details of his invention were the subject of a paper presented to the British Association in 1856, entitled "The Manufacture of Malleable Iron and Steel Without Fuel".

After many years of patient research and hard work, during which time he overcame many practical difficulties and much opposition, Bessemer commenced the manufacture of steel by his process, in Sheffield. The process consisted of the production of tool and high carbon steel by the partial decarburization of Swedish pig iron. Later, by the complete decarburization of the pig iron, he succeeded in producing very mild steels which were partly recarburized by the addition of speigeleisen or ferro-manganese, immediately before casting. R. Mushet, in 1856, had shown that a certain percentage of manganese was necessary in the finished steel to render it workable.

Since the time of Bessemer, the process which bears his name has been modified very little. The chief developments have been in the design of various items of plant, and in the methods of control.

In his original process Bessemer used "acid" pig iron, that is, pig iron low in sulphur and phosphorus, but high in silicon. The converter, or receptacle in which the blowing was performed, was lined with an acid material (ganister or silica bricks) and the slag formed from the products of oxidation were acid in character. The process became known, therefore, as the Acid Bessemer Process.

About 1877, P. C. Gilchrist completed his research which led to the use of dolomite (calcined magnesian limestone) as a basic lining. The introduction of this dolomite lining for converters rendered possible the use of phosphoric pig iron, and led to the growth of the Basic Bessemer Process.

General Principles

When a current of air is blown through molten pig iron, which contains carbon, silicon and manganese, these elements, together with some iron, are oxidized; the carbon escapes as a gas, whilst the silicon and manganese, along with any iron that may be oxidized, form a mixed ferrous-manganese silicate slag. This process is known as the Acid Bessemer Process.

The oxidation of these elements, especially the silicon, produces a large amount of heat, which not only maintains the temperature and fluidity of the molten pig iron, but actually increases its temperature. When the silicon in the pig iron is very high the heat evolved may be excessive, and additions of cold steel scrap are made to the converter so as to lower the temperature.

When the oxidation is completed, the very mild steel produced will contain a considerable amount of oxygen and maybe hydrogen, and if cast in this condition would produce useless ingots. This condition is rectified by spiegeleisen or ferro-manganese additions, which are usually made as the metal is poured from the converter into the casting ladle.

All the reactions taking place in the converter are reflected in the flame which issues from the mouth of the converter. During the early part of the blow, when the manganese and silicon are being oxidized, the flame is short and of low luminosity, gradually increasing in length and luminosity as the blow proceeds and the carbon is eliminated. When decarburization is completed, the flame "drops", and blowing is discontinued. Any prolongation of blowing after the drop of the flame results in oxidation of iron and this is indicated by the amount of brown fumes in the flame.

In the Basic Bessemer Process the blowing is prolonged after the flame drops, and this period is known as the after-blow. It is during the after-blow that the phosphorus is eliminated. The pig iron used in the basic process has a low silicon and manganese content, but is high in phosphorus. It is the oxidation of phosphorus that produces the necessary heat. in basic process

The removal of phosphorus necessitates a basic slag and a basic lined converter. The basic slag is formed from lime which is added to the converter just before the molten pig iron is poured into it.

The procedure adopted in either the acid or basic process is similar. The molten pig iron is poured into the converter, which is in an horizontal position—turned down—so that the metal is away from the tuyeres which supply the current of air and which are fixed in the bottom of the converter. The blast is then put on, and the converter rotated into a vertical position. Immediately the flame drops, the vessel is turned down again and the blown metal is poured into a casting ladle. The finishing additions are usually made at this stage.

Therefore, the process, either acid or basic, consists of three stages:

(1) A supply of hot metal.
(2) A suitable vessel for blowing the air through the molten pig iron.
(3) A casting ladle and ingot moulds, with auxiliary plant, for teeming the steel into ingot form.

Hot Metal Supply

The molten metal may be supplied in three ways:

(1) Melting pig iron in a cupola.
(2) Hot metal direct from the blast furnace.
(3) The use of a mixer, which receives the blast-furnace metal and acts as a reservoir from which molten metal can be drawn as required.

Cupolas

In the older Bessemer plants the molten metal was always supplied from cupolas. A cupola is really a small blast furnace, being a cylindrical-shaft furnace built of steel plates. It is usually lined with firebricks as a backing, and faced with ganister or some similar siliceous refractory.

Cupolas are generally charged through a door in the side of the furnace near the top, above which is a stack to carry away any flame or fumes. Near the bottom the diameter is decreased somewhat to form a hearth or receptacle for the molten metal, which can be tapped off as required. At the top of the hearth, where the cupola narrows, are the tuyeres through which the blast is admitted. The general design and proportions of a cupola are shown in Fig 1.

A moderate-sized cupola, of say 8 to 10 ft. external diameter, with 5 or 6 tuyeres, will melt approximately 400 to 500 tons of pig iron per 24 hrs. The blast is employed usually at about 2 lb. per sq. in. The size of the tuyeres depends on the volume of air required, and this is a function of the fuel consumed, which varies from 1 to 2½ cwt. per ton of pig iron melted. Each cwt. of coke, assuming that the whole of the carbon is burnt to CO_2, would require approximately 15,000 cu. ft. of air.

Generally in cupola operation the CO/CO_2 ratio in the escaping gases is unity, from which it follows that each lb. of coke requires 7·3 lb. of air (that is approximately 100 cu. ft. of air at 60 deg. F.).

The actual rate of melting varies with the fuel ratio and the rate of combustion. The rate of combustion depends on the pressure and volume of the blast employed. To some extent the pressure employed is governed by the diameter of the cupola, the greater the diameter the greater must be the pressure to penetrate to the centre of the cupola.

The pressure of the blast also affects the position of the combustion zone. A light blast burns the coke near to the tuyeres, whereas a heavy blast drives the combustion zone higher up the cupola.

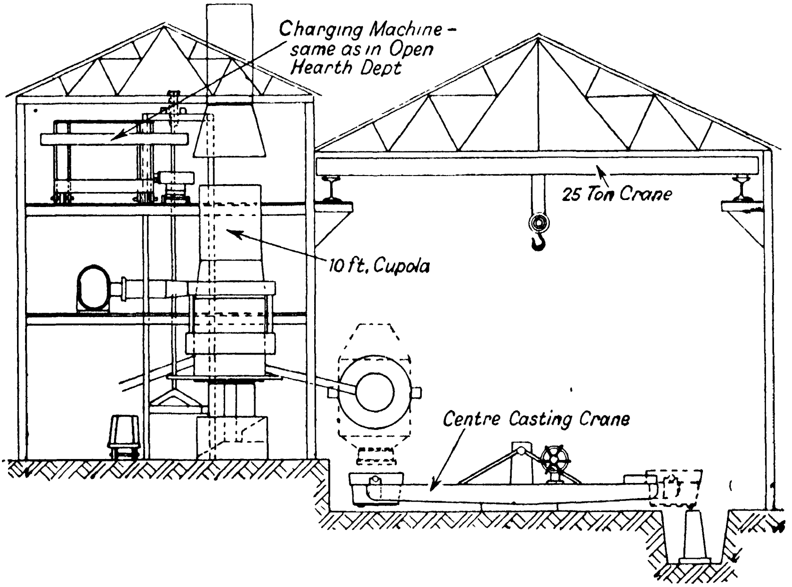


FIG. 1. Layout of cupola Bessemer shop.

When starting up a cupola a small fire is built in the hearth, the tap hole, etc., being left open to supply the necessary air for combustion. As the fire kindles, coke is charged and a light blast applied. The tap hole is then stopped up and all openings at the bottom of the cupola closed, and the charging is commenced. Alternate layers of coke and pig iron are charged as required, and the blast put on at its normal working pressure. Some lime or limestone is charged along with the coke and pig iron. The purpose of this lime is to flux the ash of the coke.

The changes which take place in the cupola are discussed in detail in Volume One.

Direct Metal

In plants where the molten metal is transferred direct from the blast-furnace plant to the Bessemer shop, hot metal ladles of the types shown in Fig. 2 are used. These ladles are steel shells mounted on railway bogies. The steel shell is lined with firebricks.

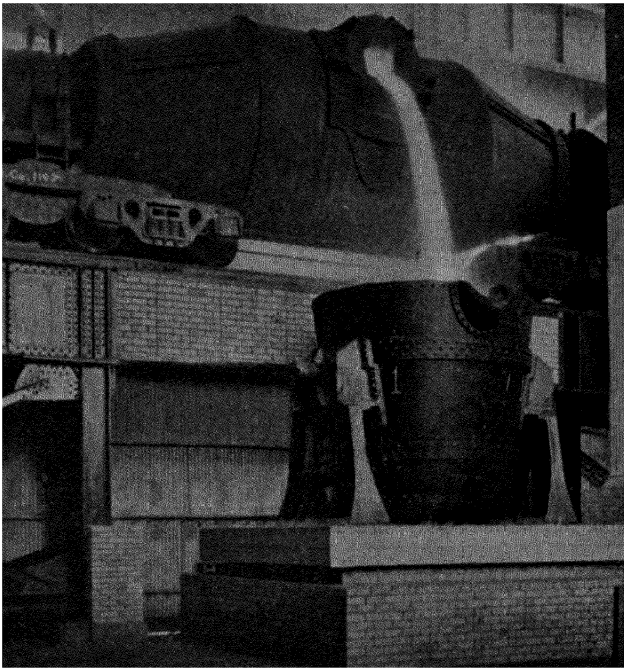


FIG. 2. Hot metal transfer car and ladle. Pouring hot metal from a 175-ton brick-lined Pugh mixer ladle car into a 50-ton open-hearth transfer ladle standing on a scale.
(By courtesy of the Iron and Steel Institute).

The use of metal direct from the blast furnace is not satisfactory, however, for the following reasons:

(1) It is difficult to synchronize the tapping of the blast furnace with the requirement of the Bessemer plant.
(2) The temperature of the molten metal is likely to vary over a wide range.
(3) The composition of the molten metal will vary widely from cast to cast, and render standardization of Bessemer shop practice impossible.

It is therefore customary to use mixers for receiving the hot metal from the blast furnace.

Metal Mixers

A metal mixer serves a threefold purpose:

(1) It acts as a reservoir of molten metal, being a buffer between the blast furnace and Bessemer plant.
(2) It mixes the various grades of pig iron supplied by the blast furnace, thereby supplying a more regular quality of molten metal to the converters. This supply of a pig iron having a more uniform analysis enables a greater degree of standardization of Bessemer shop practice, with its attendant improvements in outputs and the quality of steel produced.
(3) It also absorbs the pig iron produced by the blast furnace during the week-end when the Bessemer plant is idle.

The mixer is a cylindrical steel shell, lined with silica bricks, and mounted on heavy rollers, which enables it to be tipped in order to pour out the molten metal as required. They vary in capacity from 100 up to 1,000 tons. Mixers may be heated with blast-furnace gas. The burners for admitting the blast-furnace gas are usually situated in the roof of the mixer, sometimes one burner is placed near the pouring-out chute.

Swinden and Cawley[1] in their description of the 400-ton mixer in use in the Acid Bessemer Shop at the Workington Plant of the United Steel Co., point out that it is not their custom to heat the mixer, except during the week-end when the Bessemer plant is closed down. Their view is that "the flame developed by unrecuperated blast-furnace gas is lower than that of the molten metal poured into the mixer, and it is, therefore, possible to obtain a higher temperature in the mixer by keeping radiation and convection losses down to a minimum by closing all openings and shutting off the gas". They use a burner in the pouring-out spout which is in constant use, and which, therefore, cannot be effectively closed. This spout is kept hot in order to avoid the chilling of the metal as it runs from the mixer.

The roof burners are used for heating up the mixer from the cold start or at week-ends for maintaining the heat of the mixer.

A typical design of mixer is shown in Fig. 3. Mixers are generally rotated by hydraulic cylinders or an electrically operated rack. The movement of a mixer containing several tons of molten metal demands

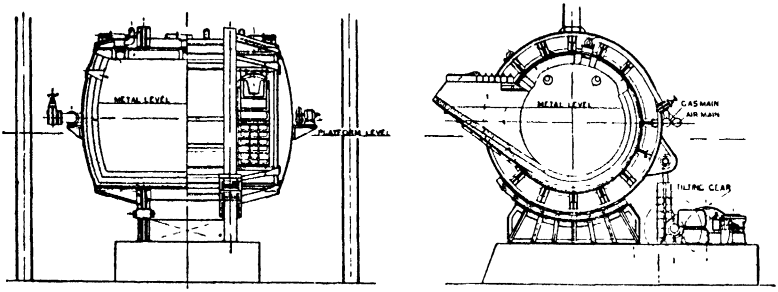


FIG. 3. 400-ton inactive mixer.
(By courtesy of " Iron and Steel Industry ".)

the provision of safeguards in case of hydraulic or electric power failure, and frequently mixers are provided with alternative methods of rotation. At the plant described by Swinden and Cawley,[1] the mixer is rotated by an electrically driven rack, but it is also provided with a steam-driven

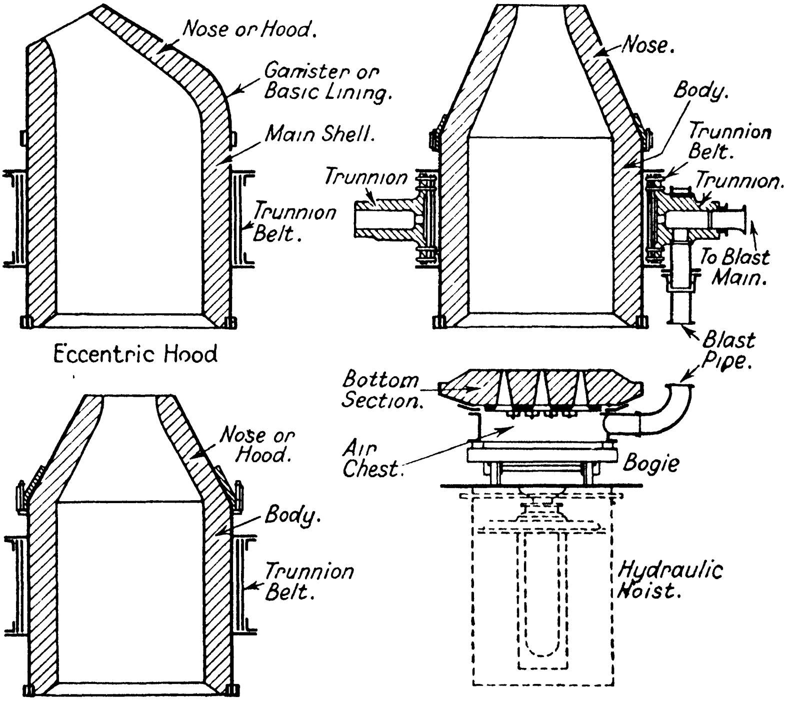


FIG. 4. Bessemer converter.

unit, which is coupled through a clutch to the same shaft as the electric tilting motor.

The Converter

The converter is the essential part of the plant. It is the receptacle in which the molten metal is subjected to the interaction of the blast. It is a vessel made of steel plates, having the shape shown in Fig. 4. The steel shell is lined with refractory material, either acid or basic, according to the process being employed. The actual method of lining the converter and the refractory materials used are described in Chapters 2 and 3.

The converter is made up of three sections; the hood or top portion, the body or middle portion, and the bottom. These sections are secured to each other by catches or cotters. This sectional arrangement facilitates re-lining the converter.

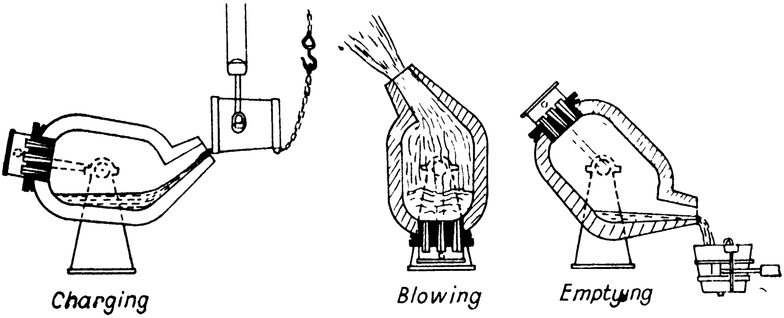


FIG. 5. The stages of the Bessemer process.

The tuyeres, which admit the air, are fixed in the bottom of the converter, and beneath this is a circular air chest. The blast is conveyed to the air chest by a pipe running down the side of the converter from the air trunnion.

The converter is supported by two trunnions. One of these trunnions is hollow and is connected with the air pipe and air chest, to which it conveys the air supplied by suitable blowing engines.

The other trunnion is attached to a toothed wheel which works in a rack and enables the converter to be rotated through an angle of at least 200 deg. to meet the various stages of the process—charging, blowing and pouring, as shown in Fig. 5.

These two trunnions are fixed to the trunnion belt, a stout steel band which surrounds the body of the converter. Usually the shell may

TABLE I

DIMENSIONS OF CONVERTERS

Capacity	Diameter of Lining	Thickness of Lining	Height of Body	Height of Hood	No. of Tuyeres	Thickness of Bottom	Size of Tuyere Holes	No. of Holes in Tuyeres	Process
10 tons	8 ft. 0 in.	12 in.	9 ft. 0 in.	6 ft. 0 in.	16	1 ft. 6 in.	¼ in.	18	Acid
15 tons	9 ft. 0 in.	12 in.	7 ft. 6 in.	7 ft. 0 in.	20	1 ft. 6 in.	¼ in.	18	Acid
18 tons	8 ft. 3 in.	12 in.	16 ft.	0 in.	24	1 ft. 8 in.	⅜ in.	12	Acid
25 tons	10 ft. 6 in.	15 in.	20 ft.	0 in.	30	2 ft. 3 in.	½ in.	10	Acid
25 tons	14 ft. 6 in.	14 in. Top. 23½ in. at the Base.	22 ft.	7 in.	—	2 ft. 10 in.	—	—	Basic

be lifted out of the trunnion belt for re-lining, and a spare shell is kept ready lined to permit quick replacement.

In some cases the hood of the converter is made vertical instead of inclined. This enables slag to be more easily removed, but, unless the converter is worked at a slight tilt, the sparks are more prone to be thrown on to the central platform.

Some dimensions of typical converters are given in Table I.

Casting Pit

The casting-pit layout varies considerably in different plants. In some cases it is a circular pit sunk below the ground. In the centre of such a pit would be a fixed ladle crane. This crane carries the ladle in such a manner that it can either be brought under the mouth of the converter to receive the metal, or carried over the moulds, which are set around the circumference of the sunken casting pit.

In most modern shops the ladle is supported by an overhead crane, which carries the ladle over moulds which may be set in a pit or on bogies. Casting-pit design and practice, however, will be dealt with fully in Chapter 10.

The Blowing Engine

The air required for the process is usually supplied by direct acting blowing engines, operated in pairs so as to obtain a regular and steady blast. Some plant engineers advocate the use of electrically driven rotary blowers. These are positive air blowers, embodying the positive displacement principle, which are capable of delivering a predetermined weight of air without any surges or pulsations.

The actual amount of air required depends upon the total impurities which have to be oxidized. Many authorities suggest that the average figure is approximately 5 to 8 tons of air per ton of iron blown. At N.T.P. this is equivalent to 16,560 cu. ft. of air per ton of iron blown. Since the process takes between 18 and 20 mins, a 20-ton converter would require approximately 16,500 to 18,500 cu. ft. of air per min.

The pressure at which this air is delivered varies between 10 and 28 lb. per sq. in., depending largely on the size of the converter. There are, however, other factors which affect this pressure. The pressure of the blast must be sufficient to keep the metal away from the tuyeres and to force the air through the molten metal. During the early part of the blow, when the metal is at its minimum temperature, a higher pressure is necessary than during the later stages of the blow, when the metal is more fluid and offers less resistance to the passage of the blast, owing to its increased temperature.

The volume of air supplied during a given period of time varies directly with the pressure of the blast and the total area of the tuyeres, but inversely with the resistance of the molten metal to passage of the blast. The pressure exerted by the metal varies as above, but averages about 4 lb. per sq. in. per foot of metal depth.

In America experiments have been conducted in an attempt to blow at a constant volume, rather than at a constant pressure, throughout the blow. The theory behind this work is that if the full pressure is maintained during the later part of the blow, when the metal offers less resistance to the blast, oxidation will proceed too fast, with detrimental results to the quality of the steel. It has, however, been difficult to demonstrate any advantage to be gained by limiting the supply of air towards the end of the blow.

General Arrangement of Plant

The general arrangement of a Bessemer plant varies considerably. In the older shops, two converters, placed on the circumference of a circular casting pit, were employed, the hoods of which pointed in opposite directions, and immediately behind the converters were the cupolas for supplying the hot metal. A system of chutes was so arranged that the molten cupola metal could be run direct into the converter when in the horizontal position.

In the centre of the circular casting pit was a ladle crane, so designed as to receive the blown metal by placing the ladle under the nose of the converter and transferring it to the ingot moulds, which were set in a sunken pit on the opposite side of the circle.

At some convenient place a smaller cupola for melting the ferro-manganese and other ladle additions was provided.

In modern shops the converters are set in a line with their hoods pointing in the same direction, away from the other parts of the plant. The mixer is situated at a convenient place for receiving the molten metal from the blast furnace and for transferring it to the converters as required. The whole shop is served by overhead travelling cranes of suitable capacity, which pour the molten metal into the mixer from the ladles which are brought from the blast furnace. These same cranes are used for handling the transfer ladles which convey the hot metal from the mixer to the converter and the teeming ladles which receive the finished steel from the converter and carry it to the ingot moulds.

A separate shop, provided with drying and baking ovens, is usually provided for preparing the converter bottoms. A typical layout is show in Fig. 6.

One of the outstanding features of a modern Bessemer shop are the devices for weighing the metal at various stages of the process. The ladle receiving the molten metal from the mixer is placed on a weighing stand, frequently provided with a dial indicator, so that when the desired quantity of metal has been drawn off, the mixer is turned up.

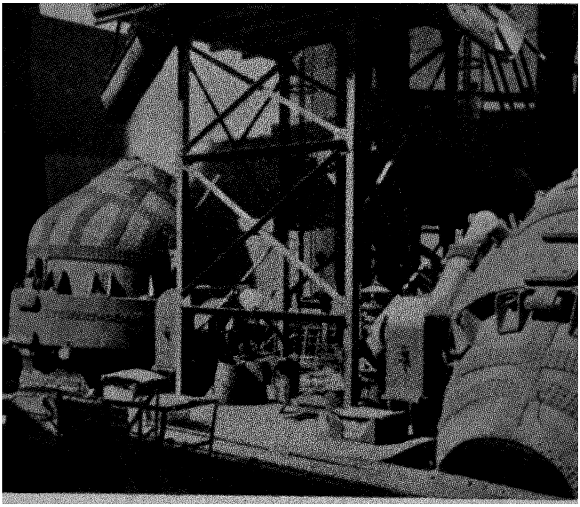


FIG. 6. Layout of Bessemer plant.
(By courtesy of the " British Steelmaker ".)

A similar device is provided for weighing the ferro-manganese and other ladle additions.

This ability to weigh the raw materials used in the process, renders possible a high degree of standardization, which results in maximum outputs and more accurate control of the final product.

Advantages of the Bessemer Process

Not only does the Bessemer process enable the production of very soft steels, but it has undoubted economic advantages which, with the greater control of the final product rendered possible by modern methods and equipment, are very attractive. Enormous outputs can be obtained from comparatively small plants.

In America a plant comprising two 10-ton capacity converters is producing between 10,000 and 12,000 tons of steel ingots per week. Two new plants which have recently been built, consisting of three 25-ton converters, are rated at 700,000 to 800,000 tons per annum.

REFERENCE

[1] SWINDEN and CAWLEY. *Iron and Steel*, April, 1939.

CHAPTER 2

THE ACID BESSEMER PROCESS

Selection of Raw Materials

In the Acid Bessemer Process the raw materials are selected in the knowledge that only carbon, silicon and manganese can be eliminated. There is no elimination of sulphur or phosphorus, and therefore the pig iron selected must be free from these elements. Carbon and silicon produce the heat necessary for the process. Therefore the following points must be considered:

Silicon, the oxidation of which produces most of the heat required to maintain and indeed increase the temperature of the molten metal, should average about 2·5 per cent. Should the silicon fall below 1·0 per cent., the metal is liable to blow cold, owing to insufficient heat being produced by the oxidation of the silicon. When the silicon exceeds about 3·0 per cent. the heat produced by its oxidation may be excessive and cause the metal to blow too hot. If the blow is too hot there is a danger of the carbon being eliminated before the silicon, and the residual silicon in the metal renders the resultant steel unserviceable.

Silicon passes into the slag as a mixed silicate of iron and manganese, and therefore the higher the silicon the greater the loss of iron in the slag.

British practice favours 2·0 to 2·5 per cent. silicon, whilst in America pig iron containing 1·5 per cent. silicon is preferred.

Carbon is usually maintained about 1 per cent. higher than the silicon content, and 3·0 to 3·5 per cent. is considered ideal. At Workington the mixer metal used in the converter runs as high as 4·0 to 4·3 per cent. The higher the carbon, the more prolonged the blow and therefore an excessively high carbon presents no advantage except perhaps a more fluid metal at the commencement of the blow.

Manganese to some extent controls the type of slag produced, and is thus of considerable importance. To produce a "dry" slag the manganese should not exceed half the percentage of silicon, that is, it should range from 0·75 to 1·0 per cent. If the manganese exceeds this figure it also delays the process. When working very high manganese pig iron there is a tendency for the MnO formed during the early stage of the blow to attack the lining, causing excessive wear.

Sulphur and Phosphorus tend to increase slightly owing to the fact that these elements are not eliminated during the process, but are actually concentrated as the blow proceeds and the bulk of the charge shrinks. They should not exceed 0·050 per cent., and for preference should be under 0·040 per cent.

Acid Bessemer Pig Iron should, therefore, have the following range of analysis:

Carbon	3·5 per cent.	– 4·0 per cent.
Silicon	2·0 per cent.	– 2·5 per cent.
Sulphur		– 0·040 per cent. *Max.*
Phosphorus		– 0·040 per cent. *Max.*
Manganese	0·75 per cent.	– 1·0 per cent.

Lining the Converter

As already explained, the slag formed in the acid Bessemer process is chemically acid and the materials used for lining the converter must also be acid.

In many English plants the refractory material used is ganister. The ganister used is an argillaceous sandstone containing sufficient clayey matter to give a good bond. It frequently approximates to the following analysis:

	Per cent.
SiO_2	94·0
Al_2O_3	1·5
FeO	1·0
CaO	0·5
MgO	0·1
Alkalis	0·2
Water	2·7
Total	100·00

The body and hood of the converter are inverted on a platform, and a wooden plug, the outside of which corresponds to the shape of the converter lining, is placed in position. The ganister is then rammed thoroughly around this plug in a manner similar to that employed for ramming a basic converter, which is described in detail in Chapter 3.

Some American works use an artificial mixture called monolithic instead of ganister. Monolithic is generally composed of 50 to 60 per cent. coarsely crushed quartz (not over two inches in dimensions), 17 to 25 per cent. finely ground fireclay, and the balance made up of old bricks and/or loam sand. The lining is installed in a similar manner to the ganister lining and gives very good results.

In some plants the body of the converter is lined with silica bricks, and, except for certain parts which are subject to abnormal wear, lasts for several months. This lining varies from 12 to 15 in. in thickness according to the size of the converter.

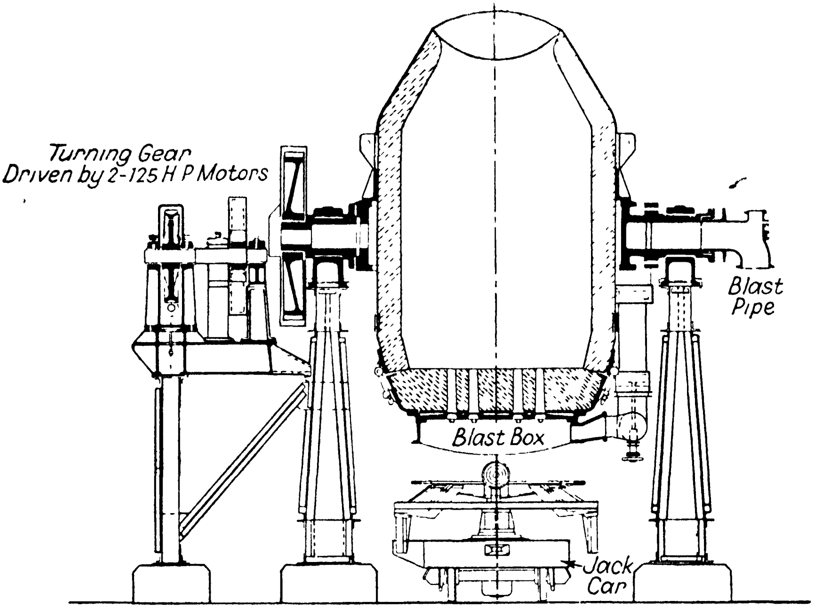


FIG. 7. A typical 25-ton acid converter.
(*By courtesy of " Iron and Steel "*)

The bottom section of the converter, which is subject to more rapid wear, and may have to be replaced after some 15 to 25 heats, is attached to the body by means of cotters, thereby making it easily and quickly replaced.

The body of the converter can be removed from the trunnion belt for rebricking or relining. A spare body can be kept for replacement, and thereby ensure continuity of production. A typical acid converter is shown in Fig 7.

The bottom section is made up with 8 to 30 tuyeres, according to the capacity of the converter. These tuyeres, which are slightly tapered cylindrical firebricks, varying from 18 to 34 in. long, perforated longitudinally, are supported by a plate—known as the bottom plate. They are inserted narrow end up and prevented from dropping through by means of clamps. Should a tuyere fail during the life of the bottom, it can either be blocked off or replaced by removing the air chest.

The space between the tuyeres is filled up either with rammed ganister or monolithic, or else bricked with firebricks and a mixture of crushed firebricks and fireclay rammed into any crevice with pneumatic hammers.

The size of the holes in the tuyere bricks, which may be 7 to 18 in number, vary from $\frac{1}{4}$ in. to $\frac{1}{2}$ in. in diameter. The most important factor is the actual area per ton of steel capacity. This varies in different countries, in America it is usually 6 sq. in. per ton; in Germany 0·75 sq. in. per ton, whilst in England it ranges from 3 to 4 sq. in. per ton.

Before use the lining of the converter is thoroughly dried. This may be done by means of a coke fire and a very low blast. The bottom section, which as already stated is more frequently changed, is usually dried in a special oven provided in the shop where they are prepared.

Reaction in the Converter

The impurities, C, Si, and Mn, are not apparently removed by direct oxidation with the oxygen of the blast. As would be expected by the law of mass action, iron is first oxidized to Fe_3O_4:

$$3\ Fe + 2O_2 \rightarrow Fe_3O_4$$

This iron oxide then reacts with the impurities:

$$\begin{aligned} 2\ Si &+ Fe_3O_4 \rightarrow 3\ Fe + 2\ SiO_2 \\ 4\ Mn &+ Fe_3O_4 \rightarrow 3\ Fe + 4\ MnO \\ 4\ C &+ Fe_3O_4 \rightarrow 3\ Fe + 4\ CO \end{aligned}$$

The carbon monoxide produced at the lower levels of the converter burns to CO_2 at the throat and produces a luminous flame.

Some of the higher oxide of iron is only partially reduced:

$$2\ Fe_3O_4 + Si \rightarrow 6\ FeO + SiO_2$$

This FeO, together with the MnO formed, combines with the SiO_2 to form a highly siliceous ferrous-manganese silicate slag.

The Blow

These reactions taking place in the converter are reflected in the type of flame issuing from the throat. As soon as the molten metal has been poured into the converter, whilst in an horizontal position, the blast is turned on and the converter is rotated into a vertical position (see Fig. 5). Immediately a small flame and a large quantity of sparks issue from the throat of the converter. After two or three minutes the flame begins to increase, and the quantity of sparks to decrease. This indicates the removal of silicon and manganese, and it continues until the carbon begins to burn off, when the flame increases in size and luminosity. When most of the carbon has been removed, the flame diminishes and becomes less luminous until the flame drops, which indicates the complete removal of carbon. If blowing is continued beyond this point, reddish fumes, indicating the oxidation of iron, are seen at the throat. Over-blowing in this manner would result in over-oxidized metal and a loss of yield, which may be very appreciable.

Immediately the flame "drops", the converter should be turned down (rotated into an horizontal position) and the metal poured into a teeming ladle. As the metal is being poured into this teeming ladle the necessary finishing and alloy additions are made.

It is obvious that the quality of the resultant steel depends to a great extent on the skill and alertness of the blower (the man in charge of the blowing operation), but with trained men and a high degree of standardization, a very high quality product can be made.

During recent years both the spectroscope and photoelectric cells have been introduced to assist the blower to control the process. They have not been generally adopted, however, because many operatives have found no advantages accruing from their use.

As shown by Swinden and Cawley:[1]

(1) The changes in the spectrum are not as clearly defined as those seen by direct observation of the flame.

(2) The spectroscope offers no help in controlling the temperature of the blow, a factor of supreme importance.

(3) The blower must anticipate, and invariably finds direct observation of the flame a better guide than having to have his eyes fixed on an instrument.

On the other hand, at an American plant, referred to by H. W. Graham,[2] comprising five converters, photoelectric cells have been employed with success. Their records show that this innovation has resulted in a narrower range of temperature at the end of the blow, giving greater freedom from surface defects and a sounder structure in the centre of billets and bars. They also claim a higher workability in machining, forming and fabricating the finished steel.

An interesting description of the application of a direct-vision spectroscope to the control of the blow of a Tropenas Converter, is given by Jazwinski.[3] A very faint continuous spectrum appeared at the beginning of the blow and grew stronger as the flame became brighter. The most pronounced yellow band appeared at the commencement of the boil, and when ejects ceased, a green band appeared, faint at first. During the carbon flame two other green bands and two red bands became visible, and at times a blue band was discernible. When all the bands disappeared the air was shut off. The position of these bands in the spectrum is shown in Fig. 8, whilst Table II gives the changes in

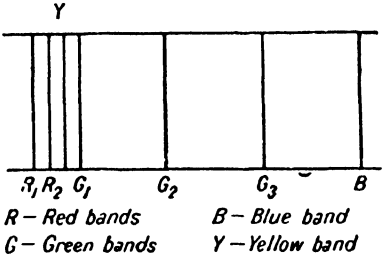


FIG. 8. The spectrum of a blow.
(By courtesy of the Iron and Steel Institute.)

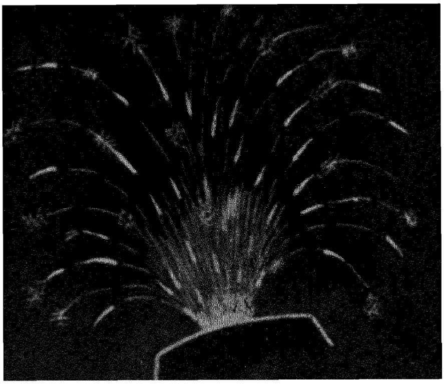
TABLE II

CHANGES IN THE SPECTRUM DURING A BLOW

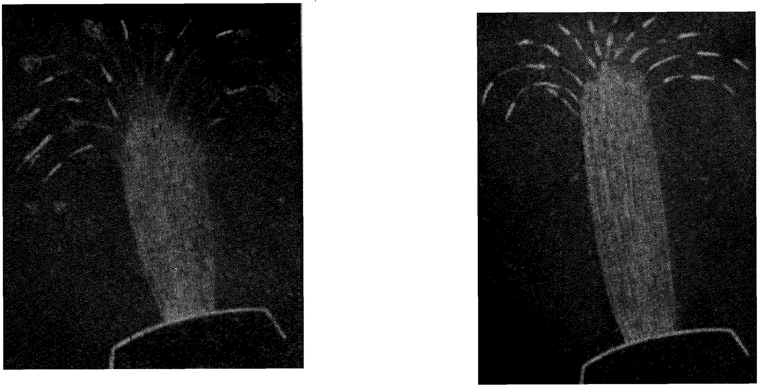
The figures indicate the number of minutes that had elapsed after the beginning of the blow when a certain band or bands appeared or disappeared

Exp. No.	Appearance of Yellow Band	Appearance of Green Bands		Appearance of Red Bands	Disappearance of Bands	Wind Shut off	Blown Metal Carbon per cent.
		Faint	Clear				
1	12	—	37	37	40	40	0·08
2	12	30	32	32	37	37	0·08
3	9·5	33	36	38	43·5	43·5	0·09
3	10·4	24	33	—	40	40	0·08
5	4	23	27·5	27·5	32	32	0·07
6	9	33·5	33·5	35·5	38	38	0·09
7	10	27·5	32	32	39·5	39·5	0·09
8	9	22	28	28	34	34	0·08
9	12	28	39	39	42	43	0·05*

* Overblown metal.



(*a*) The Start of the Blow



(*b*) The Boil

(*c*) After Ejects have Ceased

(*d*) The Carbon Flame

(*e*) The Shortened Flame at the End of the Blow

FIG. 9. The appearance of the Bessemer flame.
(*By courtesy of the Iron and Steel Institute.*)

the spectrum at various stages of the blow. It is claimed that this application has reduced the likelihood of overblown heats and considerably improved the control of quality.

The progress of a typical heat is shown in the graph in Fig. 10, whilst Figs. 11 and 12 show the progress of American low-silicon blow and a high-manganese blow respectively.

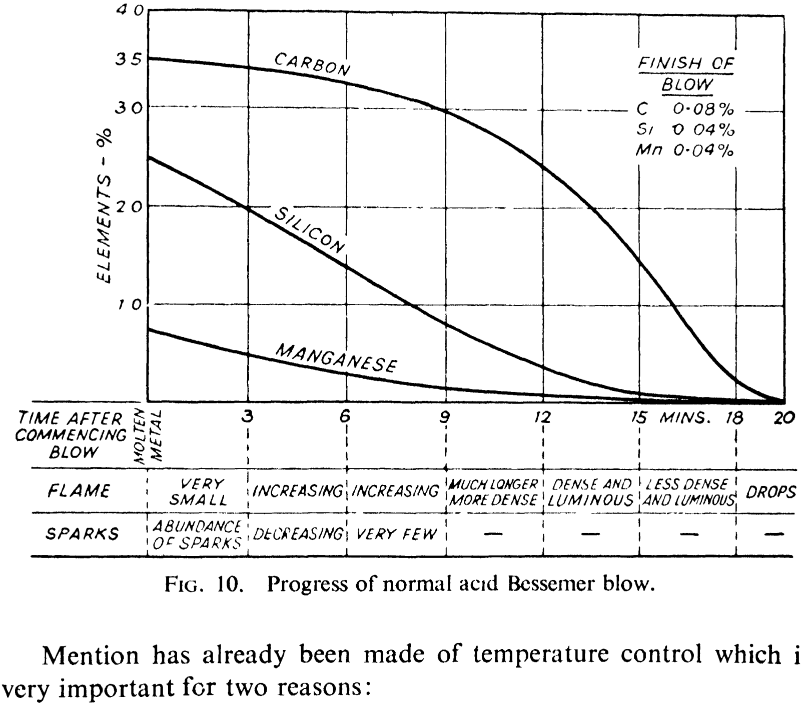


FIG. 10. Progress of normal acid Bessemer blow.

Mention has already been made of temperature control which is very important for two reasons:

(1) During the early stages of the blow, an excessive temperature, due to high silicon content of the original metal, may result in the carbon being eliminated before the silicon. The carbon flame appears and drops in a normal manner, but the blown metal contains a high residual silicon which makes the steel unserviceable. This excessive temperature may be corrected by turning the converter down after a few minutes' blowing, and making a cold scrap addition before proceeding with the blow. At some works this scrap can be added by means of a drop-bottom bucket held by an overhead crane whilst the blow is still in progress.

The reason for the removal of carbon before silicon in an excessively hot blow is due to the more exothermic nature of the

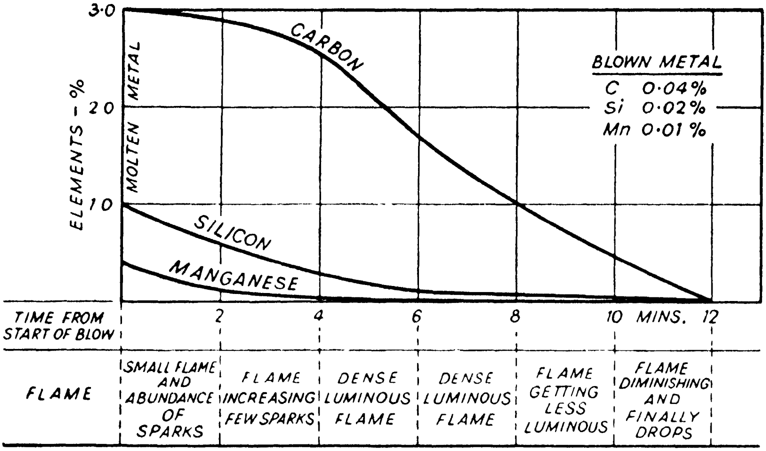


Fig. 11. Progress of a low-silicon blow.

silicon reaction which will be explained later. Excessive heat tends to favour the reaction producing less heat units and to retard the reaction of a more exothermic nature.

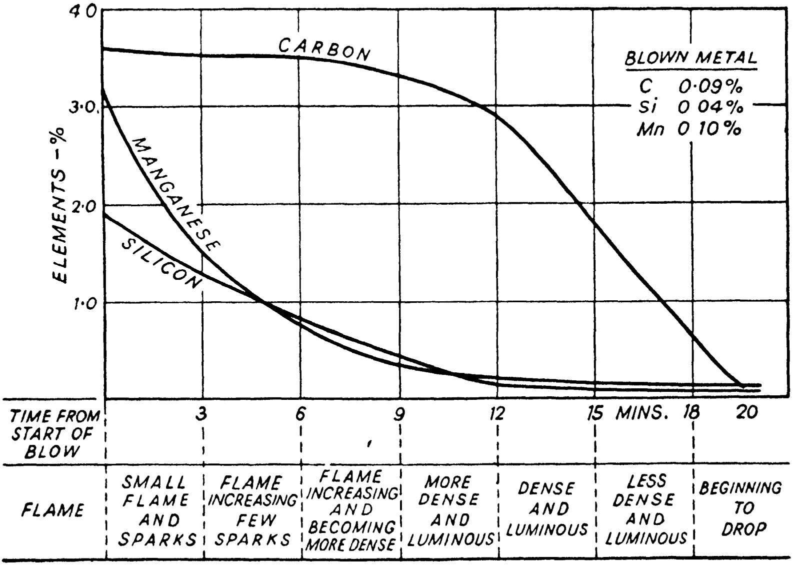


Fig. 12. Progress of high-manganese blow.

(2) Towards the end of the blow, excessive heat will cause casting-pit troubles, resulting in cracked ingots. This aspect, however, will be discussed in more detail when dealing with casting-pit practice. If the temperature, however, is too low, the metal will skull the ladle and also give rise to scabby ingots, which produce inferior bars.

This control of temperature demands considerable skill and judgment by the blower, and attempts at pyrometric control have so far failed. Invariably excessive temperature is indicated by the ejection of large splashes of metal from the mouth of the converter. This naturally means additional loss of metallic yield, which may be considerable and must be speedily corrected.

Ladle Additions

Alloy or finishing additions are usually made as the metal runs from the converter into the casting ladle. If large quantities of additions are being made, they are normally pre-melted in a small cupola provided for that purpose.

The purpose of these ladle additions is twofold:

(1) To deoxidize the blown metal.

(2) To correct the analysis of the blown metal to the required specification.

The carbon may be adjusted by an addition of anthracite coal to the ladle. Seven pounds of coal will raise 20 tons of metal 0·01 per cent. carbon. A considerable amount of carbon, however, will be introduced by the spiegeleisen or ferro-manganese, and anthracite addition may be unnecessary.

Normally ferro-manganese contains 78 per cent. manganese and 7 per cent. carbon. About 25 per cent. of the manganese is lost by oxidation when added to blown metal, leaving an effective percentage of:

$$\frac{75 \times 78}{100} = 58{\cdot}5 \text{ per cent.}$$

Spiegeleisens may be obtained with varying manganese contents. They usually contain 10 per cent. to 35 per cent. manganese and 4·5 per cent. carbon.

When making low-carbon steels, ferro-manganese possessing a high Mn : C ratio is employed, whereas in the manufacture of high-carbon steels a spiegeleisen with a low Mn : C ratio is used.

An example of ladle additions on a 20-ton heat are given in the following calculations:

Alloys Available

Ferro-manganese	C 7·0	Mn 78
Spiegeleisen	C 4·5	Mn 14·6

Specification required	C 0·10	Mn 0·40
Analysis of blown metal	C 0·06	Mn 0·05
Additions required	C 0·04	Mn 0·35

Manganese required = 0·35 per cent.

Multiplied by tonnage 20 × 20 = 400 cwt.

Divided by effective per cent. Mn 58·5 = 140·00 cwt. per cent.

Ferro-manganese required = 2·4 cwt. approx.

The addition of 2·4 cwt. of ferro-manganese to a 20-ton heat would give 2·4 × 7 = 16·8 cwt. per cent. of carbon, which will raise the metal $\frac{16\cdot8}{400} = 0\cdot042$ per cent. in carbon.

Therefore the analysis of the finished steel, after ladle additions, would be:

	C	Mn
Analysis of blown metal	0·060	0·050
Introduced by ferro-manganese	0·042	0·350
Analysis of finished steel	0·102	0·400 per cent.

When making a steel with a higher carbon content, spiegeleisen containing C 4·5 per cent. and Mn 22 per cent. is generally used:

	C	Mn
Specification required	0·40	0·70
Analysis of blown metal	0·06	0·05
Elements required	0·34	0·65

Manganese required multiplied by the tonnage in cwt. and divided by the effective percentage of manganese in the alloy, which is $\frac{75}{100} \times 22$ or 16·5 per cent., gives the amount of spiegeleisen required, thus:

$$0{\cdot}65 \text{ per cent.} \times 400 \text{ cwt.} = 260{\cdot}00 \text{ cwt. per cent.}$$

$$\frac{260{\cdot}00 \text{ cwt. per cent}}{16{\cdot}5 \text{ per cent.}} = 15{\cdot}75 \text{ cwt. of alloy.}$$

This addition of spiegeleisen would also give carbon to the extent of 15·75 × 4·5 divided by 400, which is 0·177 per cent. carbon. Therefore:

Carbon required	0·34 per cent.
Carbon in spiegeleisen ..	0·18 per cent.
leaving a balance to be added of	0·16 per cent.

This would require 16 × 7 = 112 lb. of anthracite coal.

At some works the carbon content is adjusted by the addition of mixer metal instead of using anthracite coal. If the mixer metal contains 3·5 per cent. carbon, 2·5 per cent. silicon and 0·75 per cent. manganese, then:

$$\frac{0{\cdot}16 \times 400}{3{\cdot}5} = 18{\cdot}28 \text{ cwt. of mixer metal.}$$

This addition would also introduce:

Manganese $\frac{18{\cdot}25 \times 0{\cdot}75}{420} = 0{\cdot}032$ per cent.

Silicon $\frac{18{\cdot}25 \times 2{\cdot}5}{420} = 0{\cdot}108$ per cent.

An allowance for these elements must be made when calculating the spiegeleisen addition.

The silicon content required in the finished steel can also be adjusted by use of ferro-silicon alloy containing either 45/50 per cent. or 75/80 per cent. silicon. The silicon loss is usually about 25 per cent.

These ladle losses depend on local practice and the degree to which local practice is standardized.

Slag Control

These ladle additions give rise to oxidation products which may be trapped in the steel as non-metallic inclusions. If the steel is sufficiently hot and the ladle additions are not excessive they should rise to the surface, giving clean steel. If the metal is cold, however, they may be trapped, giving rise to dirty steel and its subsequent troubles.

Frequently a heat that has been overblown may be dirty due to the excessive amount of deoxidation products produced, and to them being trapped in the steel as non-metallic inclusions.

Non-metallic inclusions in the finished steel may be also aggravated by incorrect slag control. In the acid Bessemer process, the aim is to produce a so-called "dry" slag. This slag is a highly refractory silicate of iron and manganese, which remains in the converter when the metal is poured into the casting ladle. Such a slag approximates the following range of composition:

> SiO_2—63/68 per cent., FeO—12/18 per cent., MnO—12/18 per cent., Fe_2O_3—1/3 per cent., Al_2O_3—2/4 per cent., with small amounts of CaO and MgO.

Any increase in FeO or MnO increases the fluidity of the slag, and if they are sufficiently high a "wet" slag results.

The production of a "dry" slag, therefore, depends on the FeO and MnO content of the slag. A high Mn content in the original pig iron tends to produce a "wet" slag. In order to obtain a "dry" slag the Si/Mn ratio in the metal supplied to the converter should be of the order of 2·0 or 2·5. When this ratio falls to lower values "wet" slags are produced.

"Wet" or fluid slags also result when the metal is overblown. This overblowing increases the oxidation of iron and an increased FeO content in the slag renders it more fluid.

The type of slag produced in the acid Bessemer process is a contrast to that produced in the open-hearth process. An active and fluid slag, which plays an important role in the refining of the metal, must be maintained in the open-hearth furnace, whereas an inactive slag, which is merely a receptacle for the products of oxidation, is the aim in the acid Bessemer process.

Slag control is particularly important when producing rimming steel in the acid Bessemer converter. As shown by Swinden and Cawley[4] a "wet" slag will cause an ingot, which would normally rise 2 or 3 in. in the mould, to become a rising ingot with all the subsequent troubles connected with an ingot of that type.

Converter Gases

The analysis of the gases leaving the mouth of the converter changes considerably as the blow progresses. During the early part of the blow the CO_2 is high and the CO is low. This ratio is reversed at the end of the blow. Figures given by Snelus[5] are shown in Table III.

TABLE III

CONVERTER GASES

Blowing Time	CO_2	CO	O	H	N
2 minutes ..	10·71	—	0·92	88·37	
4 " ..	8·59	3·95	—	0·88	86·58
6 " ..	8·30	4·52	—	2·00	85·28
10 " ..	3·58	19·59	—	2·00	74·83
12 " ..	2·30	29·30	—	2·16	66·24
14 " ..	1·34	31·11	—	2·00	65·55

The reasons for this fact may be twofold:

(1) At the commencement of the blow the flame temperature will normally be about 1,350 deg. C. rising to 1,580 deg. C. at the end of the blow. This increased temperature at the end of the blow will favour the production of CO.

(2) The carbon elimination reaches its peak after the elimination of the silicon and manganese, during the later stages of the blow.

Gases in the Steel

The gas content of acid Bessemer steel is of considerable interest. Much valuable work on the subject has been done by Swinden and Cawley.[14]

Hydrogen in acid Bessemer steel is normally lower than in similar qualities of open-hearth steel. The hydrogen shows a very pronounced decrease during the early stages of the blow, after which it remains fairly constant except at the finish of the blow when it increases very slightly.

The oxygen dissolved in the metal tends to increase as the blow progresses. This increase is slow until the carbon has dropped to about 1 per cent., after which the oxygen content increases rapidly. The oxygen content of the finished steel, however, is no higher, in some cases much lower, than the oxygen content of similar grades of open-hearth steel.

There is, therefore, no justification for the view sometimes advanced that Bessemer steel is necessarily over-charged with oxygen and gases.

The nitrogen content of acid Bessemer steel is higher than that of similar types of open-hearth steel. It increases progressively during the blow. The increase appears to be accelerated by the carbon elimination.

It has been suggested that the nitrogen content is dependent on the blowing time, the longer the time of the blow, the higher being the nitrogen content. A longer blow generally results in a high finishing temperature which would increase the solvent power of steel for nitrogen. It is, therefore, not clear whether the nitrogen content is a function of the blowing time or finishing temperature or both.

Tables IV and V show the behaviour of these gases during blowing, and typical gas contents of acid Bessemer and basic open-hearth steels, based on the researches of Swinden and Cawley.[1]

TABLE IV

Changes in Gas Contents

	C	Mn	Si	O	H Ml per 100g	N
	%	%	%	%		%
Mixer Metal ..	4·1	0·8	2·0	0·003	3·0	0·002
Blown Metal .	0·05	0·04	Trace	0·06	1·0	0·013
Finished Steel after alloy addition.	0·08	0·40	Trace	0·017	1·0	0·013

TABLE V

Typical Gas Contents of Steels

Steel	C%	Si%	Mn%	O%	N%
Acid Bessemer	0·16	0·06	0·59	0·015	0·014
Acid Bessemer	0·09	Trace	0·37	0·018	0·011
Basic Open	0·16	0·20	0·53	0·031	0·005
Hearth	0·14	Trace	0·47	0·019	0·004
Acid O.H.	0·10	0·11	0·47	0·026	0·004

The higher nitrogen content probably accounts for the higher tensile strength and the increased resistance to wear which acid Bessemer steels possess when compared with open-hearth steels of similar carbon and manganese contents. It also appears to improve the machinability and reaction to cold work.

Uses of Acid Bessemer Steel

British Standard Specifications permit a wide range of uses for acid Bessemer steel. It is pre-eminently suitable for rails where resistance to wear is desired. It is also employed for railway axles, tyres, springs, sleepers, wire, edge tools, saws, screws, tubes and case-hardened parts.

In America a large tonnage is used for screw and butt-welded tube manufacture.

Dephosphorizing Acid Bessemer Steel

An interesting paper describing a rapid method of dephosphorizing acid Bessemer steel in use in the U.S.A., was read by G. M. Yocom[6] before the American Institute of Mining and Metallurgical Engineers in 1941.

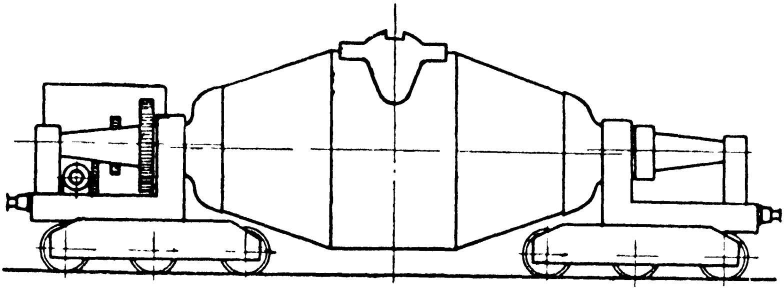


FIG. 13. A 100-ton travelling metal mixer.

The raw materials are carefully controlled so as to produce a "dry" slag. This is done by using a mixture of direct and cupola metal so that the molten metal charged into the converter approximates Si 1·40 per cent., Mn 0·55 per cent. or a Si/Mn ratio of 2·5 : 1. By careful control of the blow this produces a thick, viscous slag, ranging SiO_2 63/68 per cent., FeO 12/18 per cent., MnO 12/18 per cent. This slag can be raked off and the metal and slag completely separated.

As the metal, free from slag, is run from the converter into the casting ladle, a dephosphorizing mixture is added to the stream. This mixture consists of 50 per cent. lime, 30 per cent. roll scale, and 20 per cent. flux. When melted, this mixture gives a slag of the following composition: CaO 48 per cent., Fe_2O_3 28 per cent., SiO_2 7 per cent. and Al_2O_3 7 per cent.

The dry mixture is held in a hopper and runs down a chute into the stream. No advantage was gained when the mixture was melted before being added.

The success of the process appears to depend on a vigorous but controlled ladle reaction. A mild ladle reaction results in a high phosphorus content in the finished steel.

This method has enabled the phosphorus content of the blown metal to be reduced from 0·100 per cent. down to 0·02/0·040 per cent.

Thermochemistry of the Process

As already stated, the elimination of C, Si, and Mn in the acid converter produces heat, so that the metal is actually hotter at the end of the blow than when poured into the converter.

Some idea of the amount of heat evolved is given in the following calculation. The thermochemical data, such as the heats of formation, are given on p. 455.

	C	Si	Mn
Analysis of mixer metal	3·50	2·50	0·75
Analysis of blown metal	0·05	0·05	0·05
Impurities eliminated	3·45	2·45	0·70

Therefore, for every 100 lb. of mixer metal blown, the following heat is evolved:

		C. Th. Units.
Carbon, 2/3 oxidized to CO	= 2·30 × 2400	= 14,810
1/3 oxidized to CO_2	= 1·15 × 8080	
Silicon oxidized to SiO_2	= 2·45 × 7830	= 19,183
Manganese oxidized to MnO	= 0·70 × 1724	= 1,206
Iron approx. 4 per cent. becomes FeO	= 4·00 × 1173	= 4,692
	Total C. Th. Units	= 39,893

This calculation assumes that the C, Si, and Mn exist as free elements, whereas they actually exist as compounds such as Fe_3C, FeSi, MnC_2, etc. A certain amount of heat, therefore, must be absorbed in breaking up these compounds and this heat must be deducted from the above figure. It is difficult to make this correction unless the constitution of the molten metal is known. Even if the whole of the manganese existed as carbide, which has the highest heat of formation of those compounds existing in the pig iron, the heat absorbed would be (0·7 × 2110) = 1,477 C. Th. Units per 100 lb. of metal blown. This would leave a total heat production of 38,416 C. Th. Units per 100 lb. of pig iron converted.

To this total must be added the heat due to slag formation. Since the final slag will approximate 65 per cent. SiO_2 and 12 per cent. to 18 per cent. FeO and MnO, the total quantity of slag produced in converting 100 lb. of molten metal to the above analysis will be:

$$100 \text{ lb. metal} = 2{\cdot}45 \text{ lb. Si} = 5{\cdot}39 \text{ lb. } SiO_2$$

$$5{\cdot}39 \text{ lb. of } SiO_2 \text{ will produce } \frac{59{\cdot}39 \times 100}{65} \text{ lb. of slag}$$

$$= 8{\cdot}3 \text{ lb. of slag.}$$

0·7 lb. of manganese of 0·9 lb. of MnO pass into this slag giving:

$$\frac{0{\cdot}9 \times 100}{8{\cdot}3} = 10{\cdot}8 \text{ per cent. MnO.}$$

Therefore the FeO in the slag will be in the higher range, say 18 per cent. That means that:

$$\frac{18 \times 8{\cdot}3 \times 56}{100 \times 72} = 1{\cdot}16 \text{ lb. of iron pass into the slag.}$$

From the thermochemical data on p. 455, the heat due to slag formation will, therefore, be $(0{\cdot}9 \times 5023) + (1{\cdot}16 \times 3443) = 7{,}510$ C. Th. Units per 100 lb. of metal converted.

Thermal Balance

When preparing a thermal balance sheet of a Bessemer blow, Richards[7] suggests that the following headings should be considered:

(*a*) *Heat Input*

(1) Heat in body of the converter.
(2) Heat in the molten pig iron.
(3) Heat in the alloys added.
(4) Heat in the blast (if warm).
(5) Heat developed by oxidation.
(6) Heat developed by slag formation.

(*b*) *Heat Expended*

(1) Heat in body of converter at the end of blow.
(2) Heat in the finished steel.
(3) Heat in the slag.
(4) Heat in the gases escaping.

(5) Heat in the fume, etc.
(6) Heat absorbed in decomposing the moisture in the blast.
(7) Heat required to separate constituents of the original pig iron.
(8) The balance being the heat lost by radiation and conduction.

Sexton and Primrose[8] summarize the thermal balance sheet as follows:

Heat Input	Per cent.
Heat carried in by molten metal	= 33
Heat due to warm converter and slag formation	= 7
Heat due to oxidation of carbon	= 24
Heat due to oxidation of silicon	= 26
Heat due to oxidation of manganese	= 10
Heat due to oxidation of iron	
	100

Heat Expenditure	Per cent.
Heat in molten steel	= 46
Heat in molten slag	= 10
Heat carried away by gases	= 23
Heat lost in decomposing moisture in the blast	= 1
Heat lost by radiation and conduction	= 20
	100

Metallic Yield

The economics of any process depend to some extent on the yield of sound metal obtained from the raw materials used. With a pig iron of the composition used in the previous example, the elimination of impurities would result in a loss of:

Carbon ..	3·45 lb. per 100 lb. of metal.
Silicon ..	2·45 lb. per 100 lb. of metal.
Manganese ..	0·70 lb. per 100 lb. of metal.
Iron	4·00 lb. per 100 lb. of metal.
	10·60 lb. per 100 lb. of metal.

This represents a yield of 89·4 per cent.

A considerable quantity of iron, however, is ejected from the mouth of the converter during the process. This figure may vary from 2 to 3 per cent., so that good practice should give a yield of approximately 87 per cent.

This yield, however, may be seriously reduced by:

(1) Overblowing, which results in excessive oxidation of the metal and loss of iron as FeO.

(2) Excessive ejection of metal from the mouth of the converter.

(3) Cold metal, which results in ladle skulls and reduced weight of ingots.

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CHAPTER 3

THE BASIC BESSEMER PROCESS

Selection of Raw Materials

As already stated, the Basic Bessemer Process employs a basic-lined converter, which not only allows, but necessitates, the maintenance of a basic slag. This basic slag permits the removal of phosphorus and to some extent sulphur. Silicon can be removed in the same way as in the acid process, but the SiO_2 formed must be neutralized in the slag by lime. Therefore, when selecting pig iron for the basic Bessemer process, these factors must be considered.

Carbon is readily removed and the amount present is of little importance.

Silicon is readily removed but its presence prolongs the blow. It also tends to make the blow hot, which may interfere with subsequent dephosphorization. A high silicon content means an increased quantity of lime to neutralize the silica formed and to maintain a basic slag. Since silicon is not required as a heat producer, it should, therefore, be kept as low as possible. A good basic iron should not exceed 1·0 per cent. silicon, the ideal is probably about 0·5 per cent. silicon.

Sulphur. The removal of sulphur is somewhat erratic. It passes into the slag as a mixed sulphide of calcium and manganese. Owing to the uncertainty of its elimination, it is advisable to keep this element under 0·10 per cent.

Manganese. The removal of this element is slower than in the acid process, probably due to the more basic slag carried and to less SiO_2 present. It plays an important part in the process, however, helping to maintain the basicity of the slag, assisting the removal of sulphur and protecting the iron from excessive oxidation during the later stages of the blow. Lister[1] suggests 2·5 to 3·0 per cent. manganese, but other workers specify a lower percentage varying from 0·70 to 1·5 per cent.

Phosphorus is the heat producer in the basic process, in the same way as silicon is in the acid process. Lister[1] suggests 2·0 per cent. as a minimum and 3·0 to 4·0 per cent. as being not uncommon. He recommends 2·5 per cent. as ideal. Many plants, however, are working successfully with a phosphorus content of 1·5 to 2·0 per cent.

One object of the careful selection of the pig iron is the production of a slag high in phosphoric acid, suitable for use as a fertilizer. An abnormally high silicon will naturally tend to lower the P_2O_5 content and lessen the value of the slag as a fertilizer. Considering all factors the ideal range of analysis of a basic Bessemer pig iron appears to be:

Carbon	3·0 to 3·6 per cent.
Silicon	0·6 to 1·0 per cent.
Sulphur	0·08 to not over 0·10 per cent.
Phosphorus	1·8 to 2·5 per cent.
Manganese	1·0 to 2·5 per cent.

Lime used in the process should be as low as possible in silica and sulphur. Roughly 3·0 per cent. of CaO is required to neutralize each 1·0 per cent. of SiO_2, therefore, the available base in a lime containing 91·0 per cent. CaO and 3·0 per cent. SiO_2, is:

	Per cent.
	91 CaO
less 3 × 3 =	9
Available Base	82

whereas if the SiO_2 is only 1·4 per cent., the available base is:

	Per cent.
	91 CaO
less 3 × 1·40 =	4·2
	86·8

Lining of the Converter

In the early days of the Basic Bessemer Process several materials, including burnt lime, were suggested and tried before dolomite (that is, dead burnt magnesian limestone) which has now become universal, was discovered. At first this material was made into specially-shaped bricks, usually in a hydraulically operated press working at a pressure of about three tons per square inch. Tar, which acted as a "bond", was mixed with the dolomite before pressing. These dolomite bricks were jointed with a dolomite-tar mortar.

The modern method of lining a basic converter is to use a mixture of tarred dolomite, which is rammed into position around suitable formers. The dolomite used should approximate to the following composition:

CaO	54·0 to 56·0	per cent.
MgO	34·0 to 38·0	„
SiO_2	3·0 to 3·5	„
Fe_2O_3	1·0 to 2·0	„
Loss	1·0 to 2·0	„

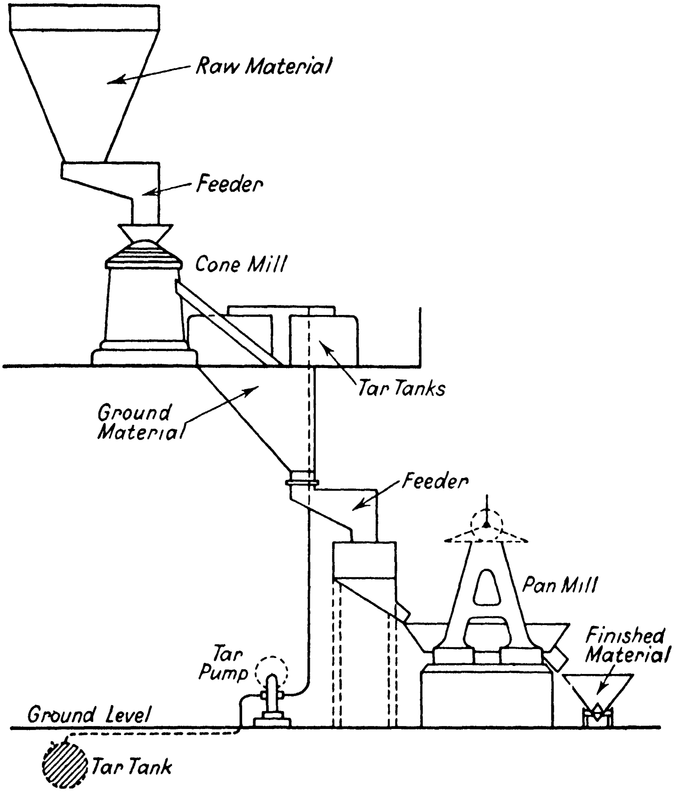


FIG. 14. Dolomite treatment plant for converter bottom.

This is the product of calcining magnesian limestone and is usually supplied in pieces varying from about 1 to 3 in. in size. This material is then crushed to approximately pea-size and mixed with tar.

At one time, various types of crushing machines and mills were used to crush this dolomite to the necessary size, which was subsequently mixed with tar by hand. A quantity of tar was poured on to a heap of dolomite and thoroughly mixed with shovels. The method was crude and a considerable variation in the tar content and the packing density of the mixture resulted. This frequently gave linings which were not uniform and which failed prematurely.

In modern plants considerable attention is given to crushing the dolomite to a standard size and adding the tar in a controlled manner, as shown in Fig. 14. The object is to grade the dolomite so as to obtain a maximum packing density, and the following grading has been recommended:

Grading	Crushed Dry	Milled Tarred
½ in. to ¼ in. mesh	5·0	6·0
¼ in. to 7 „	34·0	33·0
7 to 25 „	24·0	22·0
25 to 75 „	15·0	14·0
75 to 150 „	9·0	8·0
150 to 300 „	13·0	6·0
below 300 „	—	11·0
	100·0	100·0

	Crushed Dry	Milled Tarred
Packing Density ..	2·14	2·43
Loss	2·30	8·20
Tar content	—	5·90

The actual grading of the dolomite and the tar content varies according to the part of the converter for which it is to be used. It is usual to mill material for the bottom for about half an hour, whereas that used for the body is milled for about half that time.

The tar content is varied as follows:

Lining converter noses..	5·0 to 6·0	per cent. tar.
Converter bottoms ..	6·0 to 7·0	„ „
Body of converters ..	7·0 to 8·0	„ „
Mixture for jointing and fettling	15·0 to 20·0	„ „

During the crushing and mixing, temperature control is important. The tar is maintained at a temperature of 90 to 100 deg. C. and the tarred dolomite from the pan-mill varies from 50 to 65 deg. C.

The bottom section of the converter is detached from the body for ramming. Usually the converter body is inverted and rammed with its wide end upright. In some modern shops it is rammed nose upwards as shown in Fig. 15. A steel former is raised by the hydraulically-operated bogie into the base of the converter and, when secured with

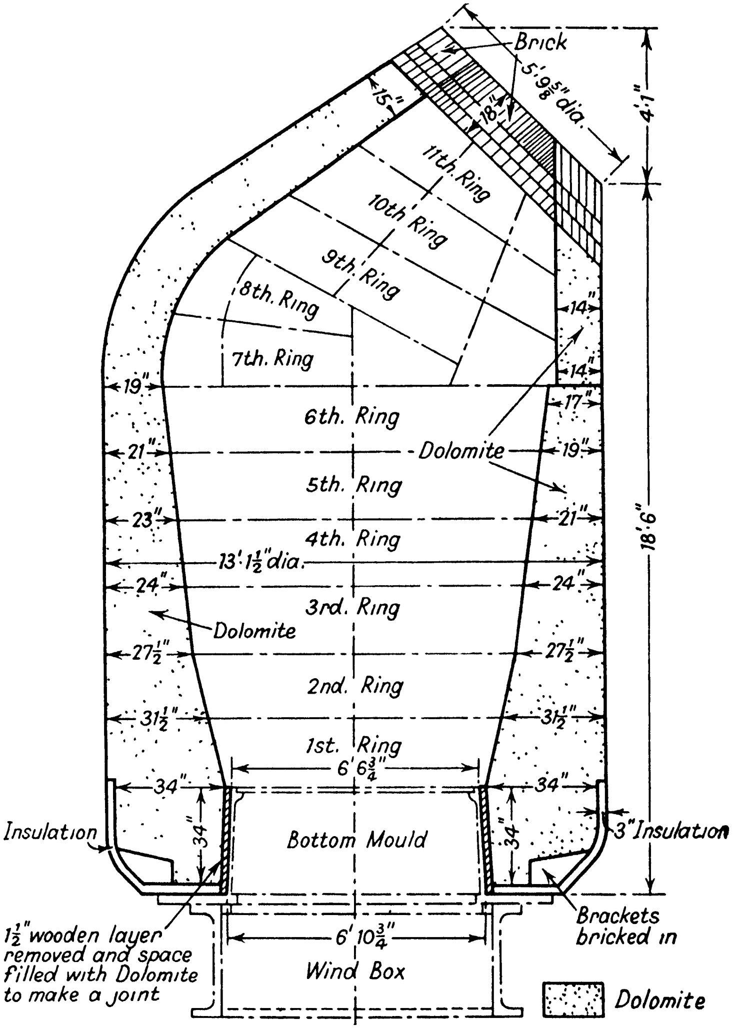


FIG. 15. Section through 25-ton basic Bessemer converter.

cotters, tarred dolomite rammed into position around it. The former is then built up gradually as shown in the diagram and the ramming continued until within about 1 ft. 6 in. or 2 ft. 0 in. of the mouth of the

CaO	25·0	to	28·0	per cent.
SiO_2	9·0	to	10·0	,,
MgO	15·0	to	16·0	,,
FeO	30·0			,,
Fe_2O_3	5·0	to	6·0	,,
Al_2O_3	8·0	to	10·0	,,

This slag thoroughly coats the lining, filling up any pores in the ramming. Any surplus slag is poured off and the converter is ready for production.

Reactions in the Converter

With the exception of phosphorus, the removal of impurities is similar to the acid Bessemer process. The oxygen of the blast combines with iron to form Fe_3O_4 which brings about the oxidation of the other impurities.

The *silicon* is more rapidly and completely removed because of the lower initial silicon and also the lime ready to combine with the SiO_2 so formed.

The removal of *manganese* is slower than in the acid process, probably owing to two factors:

(1) Mn passes into the slag as MnO SiO_2, and there is a small concentration of SiO_2 available.

(2) Lime tends to replace MnO from its combination with SiO_2, thus: $2MnO\ SiO_2 + 2CaO \rightarrow 2CaO\ SiO_2 + 2MnO$.

Sulphur is removed erratically throughout the blow and passes into the slag as a mixed sulphide of calcium and manganese thus:

$$(1)\quad FeS + Mn \rightleftharpoons Fe + MnS$$

$$(2)\quad FeS + CaO \rightleftharpoons FeO + CaS$$

The removal of sulphur is accelerated by a high manganese content in the original pig iron. This may be due to the iron-manganese reaction in equation (1) which, of course, is reversible. The equilibrium is a function of temperature and the concentration of the reacting substances. A high Mn concentration will favour the reaction proceeding in the direction of the upper arrow.

A falling temperature also favours the reaction proceeding in the direction of the upper arrow.

A high manganese content would also assist the FeS-CaO reaction in equation (2) by reduction of the FeO, thus:

$$FeO + Mn \rightleftharpoons Fe + MnO$$

thereby reducing the FeO concentration on the right and favouring the formation of CaS.

Phosphorus is not removed until all the other impurities have been eliminated. When the carbon has been eliminated the flame drops, and the after-blow commences during which the phosphorus is rapidly removed. This may be represented by the following equation:

$$8\ Fe_3P + 5\ Fe_3O_4 = 39\ Fe + 4\ P_2O_5$$

The P_2O_5 so formed immediately combines the CaO forming tetra-basic phosphate of lime:

$$P_2O_5 + 4\ CaO \rightarrow 4\ CaO\ .\ P_2O_5$$

Slag Control

A basic Bessemer slag usually approximates the following range of analysis

SiO_2	5·0	to	9·0	per cent.
CaO	45·0	to	55·0	,,
Al_2O_3	2·0	to	2·5	,,
CaS	0·1	to	0·2	,,
FeO	7·0	to	12·0	,,
MnO	2·0	to	6·0	,,
P_2O_5	18·0	to	22·0	,,
Fe_2O_3	2·0	to	8·0	,,
MgO	2·0	to	5·0	,,

The lime additions are calculated so as to give a high P_2O_5 slag, suitable for agricultural purposes. For a 25-ton heat, with 2·5 per cent. P in the original pig iron, the lime requirement would be:

25 tons of pig iron at 2·5 per cent = 0·625 tons P
$0{\cdot}625 \times 2{\cdot}3 = 1{\cdot}44$ tons P_2O_5

But 22 tons of P_2O_5 produces 100 tons of slag.

$\therefore$ 1·44 tons of P_2O_5 produces $\frac{100 \times 1{\cdot}44}{22}$ or 6·5 tons of slag.

This slag contains approximately 50 per cent. CaO, therefore the *lime required is* 3¼ *tons.*

Some of this CaO, of course, will combine with the SiO_2 produced by the oxidation of Si, to form a bi-silicate (2 CaO . SiO_2). MnO may replace part of this CaO to give a mixed silicate (CaO . MnO . SiO_2).

The Blow

The converter being hot from the previous blow, the necessary lime is added from an overhead hopper, and the iron is poured into the vessel. The blast is commenced and the vessel rotated into a vertical position. As in the acid process there is very little flame at first, but a considerable amount of sparks and splashes. These sparks are more numerous than in the acid process, due to the presence of lime. After 3 to 5 min., the silicon and some of the manganese having been removed, the flame lengthens and grows in luminosity due to carbon elimination. After about 14 to 16 min., the flame drops, indicating the end of decarburization. The blow is continued, during which period the phosphorus is removed and dense brown fumes issue from the mouth of the converter. These fumes are typical of iron oxidation.

This continuation of the blow is known as the "after-blow", and its duration is carefully timed by the blower. If blowing is prolonged too much, the metal will be over-oxidized and a low yield will result. If the period is insufficient, the dephosphorization will be incomplete.

This time-factor is, therefore, of considerable importance, and demands much skill and judgment by the blower. When using mixer metal of regular analysis, the operation is simple, but if metal is used direct from the blast furnace the process is more difficult and the phosphorus content of the finished steel likely to vary.

The converter, however, may be turned down and a sample taken by means of a long-handled spoon. This sample is cooled in water and broken, after which the fracture is examined. To the experienced eye this will indicate the phosphorus content, as the crystals become smaller as the phosphorus decreases.

The progress of typical basic blow is shown in Fig. 17.

An excessive temperature at the commencement of the after-blow may affect the phosphorus removal, but the temperature is usually corrected, as in the acid process, by scrap additions to the converter. These scrap additions are generally made after about 4 or 5 min. blowing, so as not to interfere with the time of the after-blow, which a later addition may do.

When using a *high silicon pig iron* the process is often worked in two stages. At the commencement a small quantity of iron ore, to assist silicon removal, and sufficient lime to combine with the SiO_2 formed,

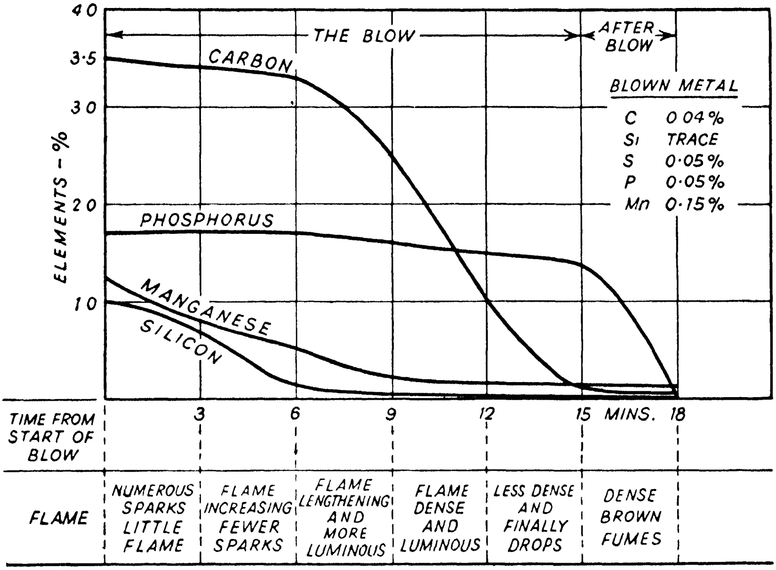


FIG. 17. Basic Bessemer blow.

is added to the converter and the molten metal poured into the vessel. The metal is then blown until the carbon flame begins to appear, which indicates removal of most of the silicon. The vessel is turned down and this siliceous slag is run off.

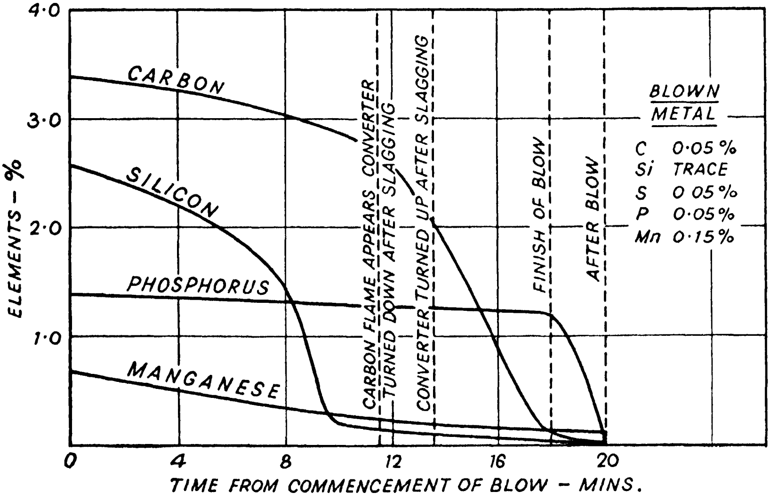


FIG. 18. High silicon basic Bessemer blow.

More lime is then added and the blow continued in the normal manner. Lister[1] gives the following analysis of the slags produced, whilst Fig. 18 indicates the progress of such a blow:

1st Slag	SiO_2	35·45 per cent.
	Fe	3·00 „
	P_2O_5	Trace
2nd Slag	SiO	11·0 to 12·0 per cent.
	Fe	8·0 to 11·0 „
	P_2O_5	14·0 to 20·0 „

This practice was used when blowing Cleveland pig iron.

Modern Basic Bessemer Practice

In recent years several excellent contributions have been made to technical literature dealing with the basic Bessemer process, which has been revived in some works in this country and America.

Most modern plants are run in conjunction with blast furnaces. The blast-furnace metal is transferred to active mixers, where the metal is partly desulphurized and desiliconized. This activated mixer practice is dealt with more fully in Chapter 12.

The use of mixers assures a more uniform hot metal for the converter, thereby allowing a high degree of standardization, resulting in a more regular product.

Kirlie[2] describes the basic Bessemer process at a large English steel plant, where the blast-furnace metal is treated and stored in 1,000-ton active mixers, which are lined with magnesite bricks. The blast furnaces are acid-burdened, and the metal is treated with soda ash in the ladles transferring it to the mixers.

Referring to the same practice, James Mitchell[3] shows that the soda ash treatment reduces the sulphur in the blast-furnace metal from 0·20 down to 0·10 per cent. The metal supplied from the mixers to the converters approximates 1·8 to 2·0 per cent. P, 0·5 per cent. Si, 0·3 per cent. Mn and 0·08 per cent. S.

Owing to the rigid standardization of the practice and modern methods of control, a high-grade product is obtained. The normal duration of the blow is 18 to 20 min., of which 17 to 18 min. constitutes the blow and 2 to 2¼ min. the after-blow.

Mitchell[3] stresses the advantages of large-capacity converters, which give greater control of the product and reduce the dangers due to time-factors.

At the plant described by Kirlie,[2] there are four 25-ton converters, two of which are blown alternately. Sometimes a third converter is in operation, whilst one normally is out for repairs.

Finishing Additions

Finishing additions are usually made to the ladle, although they are sometimes added to the converter. The phosphoric slag should be removed as far as possible before the additions are made, otherwise the FeO content of the slag may be reduced too far and the metal may be rephosphorized. Even when the slag is run off from the converter, some slag invariably finds its way into the ladle and the ladle additions will cause slight rephosphorization.

The silicon and manganese are added either in the form of ferro-manganese, ferro-silicon, silico-manganese or spiegeleisen. The loss of silicon and manganese is slightly higher than in the acid process, normally about 27 to 30 per cent.

Rimming Steel

Kirlie[2] gives the following details of a heat of rimming steel produced in the basic Bessemer converter:

	C	Si	S	P	Mn
Mixer Metal	3·00	0·5	0·09	2·00	0·70
Blown Metal	0·02	Trace	0·05	0·05	0·15
Finished Steel	0·04–0·06	Trace	0·035–0·05	0·04–0·06	0·35–0·45

Capacity of Converter	..	25 tons
Blowing time—Blow ..	..	12 min.
After-blow	..	3½ min.
Total ..	..	15½ min.

Alloy additions:

To converter	100 lb. spiegel (20·0 to 22·0 per cent. Mn).
To ladle	280 lb. ferro-manganese (77·0 to 80·0 per cent. Mn).

Slag produced:

SiO_2	CaO	MnO	FeO	Fe_2O_3	P_2O_5
6·7	51·7	3·0	10·6	3·3	20·2

Temperatures:

	Deg. C.
Mixer metal	1,255
Steel ex-converter	1,620
Slag ex-converter	1,560
Steel teeming	1,555
Steel rimming in moulds	1,520

Metallic Yield

The metallic yield in the basic Bessemer process is lower than in the acid Bessemer process. This is largely due to the amount of iron oxidized during the after-blow. The actual amount depends on how carefully the after-blow is controlled. Allowance must also be made for the metal ejected from the converter during the blow, which may be very high if the blow is hot.

With careful control, however, the losses may approximate:

	Per cent.
Carbon	3·0
Silicon	0·7
Manganese	1·0
Phosphorus	2·0
Iron	8·0
Total	14·7

Therefore the yield may be of the order of 85 to 86 per cent.

Thermal Balance

The sources and expenditure of heat in the basic Bessemer process are similar to the acid process, with the exception that phosphorus replaces silicon as the heat producer. The main thermal considerations are shown in the following abbreviated statement:

	C	Si	P	Mn
Original pig iron	3·05	0·70	2·05	1·15
Blown metal	0·05	Trace	0·05	0·15
Eliminated	3·00	0·70	2·00	1·00

In addition approximately 15 per cent. of iron is oxidized. Therefore, for every 100 *Kilogrammes of metal blown:*

Heat Input:			Calories	Per cent.
Heat in molten metal 100 × 277			27,700	33
Heat of oxidation:				
Si — SiO_2	0·7 × 7830 =	5481		
C — CO_2	1·0 × 8080 =	8080		
C — CO	2·0 × 2450 =	4900		
Mn — MnO	1·0 × 1784 =	1784		
P — P_2O_5	2·0 × 5965 =	10930		
Fe — FeO	15·0 × 1350 =	20250		
		——	51,425	61
Heat of slag formation:				
$2\,CaO.SiO_2$	1·54 × 384 =	591		
$4\,CaO.P_2O_5$	4·60 × 1131 =	5202		
		——	5,793	6
		Total ..	84,918	100

Heat Expended:	Calories	Per cent.
Heat lost in gases and fume, and absorbed decomposing moisture	21,229	25
Heat absorbed in heating lime and in the molten slag	16,983	20
Heat in the finished steel	35,665	42
Heat lost by radiation, etc.	11,041	13
Total ..	84,918	100

Uses of Basic Bessemer Steel

With modern technique, control and standardization, the products of the basic Bessemer process are capable of a wide application. Large tonnages are being used for tube manufacture. The only serious disadvantage is the high nitrogen content, which is common to both Bessemer processes, and which renders the steel unsuitable for certain classes of work. Dr. Jeavons[4] suggests that Bessemer steel is inferior to open-hearth steel for the deep drawing and pressing industry. This is probably due to "the relatively high percentage of nitrogen absorbed during the blow, which tends to reduce the ductility of the finished steel and to intensify troubles due to 'stretcher-strains', 'strain-ageing' and 'blue-brittleness'."[4]

Improved Basic Bessemer Steel

This disadvantage of high nitrogen, which may be aggravated by high phosphorus, has been appreciated for many years and considerable research to overcome these troubles has been undertaken. During the early period of the recent war, the demand for high-quality steels with good cold-forming properties accelerated this work in Germany. This work has been ably reviewed in the technical press.[5]

Observations made over a number of years indicate that the depth of the bath of metal in the converter has an important bearing on nitrogen absorption. When the diameter at the base of the converter has increased due to the wear of the lining and the worn bottom is replaced by a new bottom section, the depth of metal in the converter is reduced. This more shallow bath of metal results in a relatively large quiescent annular space around the tuyeres. These conditions tend to produce a low nitrogen content in the finished steel.

Other investigations showed that under normal operation the nitrogen content of the final steel could be fairly well correlated to the condition of the converter and the final metal temperature. A steady bath agitation, coupled with a low final temperature, favoured the production of low nitrogen steel. The opposite conditions tended to produce high nitrogen contents in the finished steel.

This work led to the development of the "H.P.N." steels, which contained less than 0·050 per cent. P and less than 0·012 per cent. N. Describing the production of these "H.P.N." steels in Germany, Frerich[6] suggests that a medium silicon iron should be used, with a phosphorus content not exceeding 2.0 per cent. The sulphur should be kept as low as possible, whilst differences of 0·5 per cent. to 1·2 per cent. Mn and from 2·0 per cent. to 3·8 per cent. C were unimportant. The converter bottom should not be more than two-thirds to three-quarters used. A normal lime burden was adequate, but the lime employed should be of a good quality and well burnt. The metal should be cooled at the commencement of the after-blow by an addition of iron ore or steel scrap. The quantity of iron ore used varied from 3 per cent. to 5 per cent. of the weight of pig iron being blown. The greater this addition the lower the final temperature of the metal and the lower the nitrogen content of the finished steel. The actual amount of ore employed appeared to depend on the initial temperature of the pig iron and the casting-pit requirements. The finished steel should be deoxidized by one-third solid ferro-manganese added to the converter and two-thirds liquid spiegeleisen added to the ladle. The chief difficulty in controlling this process appears to be the achievement of the correct final temperature, which must be low to obtain a low nitrogen content, but sufficiently high to prevent casting-pit troubles.

The phosphorus and nitrogen contents of the finished steel can also be kept to a minimum by "double-blowing". The principle of this process, described by Hofmann and Peetz,[7] depends on the dephosphorization being performed at the lowest possible temperature, whilst the actual time of contact between the metal and the blast is reduced to a minimum.

In double-blowing the pig iron is charged into the converter in two stages. Approximately half the total pig iron is charged into the converter, together with the whole of the lime necessary to refine the whole charge. This metal is blown down with a full blast until the phosphorus is under 0·10 per cent. Owing to the fact that the bath of metal is shallow, the blow proceeds with practically no ejections, and the comparatively large volume of lime causes the temperature rise during the after-blow to be much less than in a normal practice.

When this portion of the charge has been blown down to the requisite degree, the remainder of the metal is added and the whole charge is blown down to the usual end-point. A full blast is maintained during this second part of the blow, and the ejections and the blowing time are both considerably reduced. At the commencement of the second blow the bath is partially refined and the slag is practically formed and melted. The second part of the blow, therefore, proceeds quietly and dephosphorization takes place without the development of an excessive temperature.

The production of a higher grade of basic Bessemer steel has been the subject of an able paper by Dickie.[8] In this paper Dickie reviews the causes of the greater susceptibility of Bessemer steel to work-hardening and strain-age embrittlement, as compared with open-hearth steels. The fundamental causes are traced to the phosphorus and nitrogen contents of the finished steel.

He[8] shows that low phosphorus steel can be produced by normal blowing, but using less scrap than usual and removing the first slag formed. At this point the phosphorus will approximate 0·050 per cent. to 0·055 per cent. After adding a further quantity of lime, this charge is blown for an additional thirty seconds, during which time the phosphorus will be reduced to about 0·025 per cent. The nitrogen content, however, will vary from 0·015 per cent. to 0·018 per cent.

TABLE VI

Typical Month's Production of Basic Bessemer Steel (Dickie[8])

In each case a single addition of about 3,000 lb. scale was made during the fore-blow to a 25-ton capacity converter. The converter was of the wide bath type.

	Average P%	Average S%	Average N%	Index P plus SN × 1000
Within the specification (301 blows)	0·028	0·031	0·0085	70·5
Outside the specification (89 blows)	0·036	0·033	0·0105	88·5
Total Blows. Average ..	0·030	0·031	0·0090	75·0

TABLE VII

One Week's Production of Improved Basic Bessemer Steel (Dickie[8])

Some 31 per cent. of the blows were treated with two scale additions, the remainder with one scale addition. The silicon content of the iron varied from 0·46 to 1·32 per cent., with an average of 0·72 per cent.

Number of blows made	100
Number within specification	96
Blows accepted	96 per cent.
Average P	0·026 per cent.
Average S	0·030 per cent.
Average N	0·0073 per cent.
Index (P + 5N × 1000)	62·5

The index (P + 5N × 1000) is taken as a guide to the suitability of the steel for certain classes of application. A max. of 80 is suitable for some work, but it may be as low as 65 in certain cases.

Most of this nitrogen is absorbed during the blow, although some of it may exist in the initial pig iron. In the open-hearth process any nitrogen in the charge is eliminated during the carbon boil, whilst in the Bessemer process little absorption appears to take place until the drop of the carbon flame. The evolution of carbon monoxide appears to oppose the absorption of nitrogen. It is in the later stages of the blow and more especially during the after-blow, when the temperature is increasing rapidly, that the maximum increase in the nitrogen content occurs.

After reviewing the effect of time factors, temperature conditions and nitrogen pressures, which are shown graphically in Fig. 19, Dickie[8] shows that five factors affect the amount of nitrogen absorbed. He

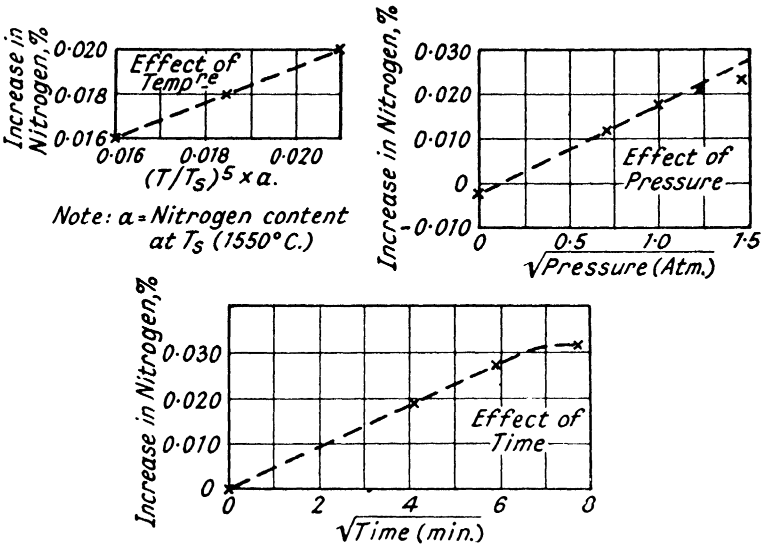


FIG. 19. Effect of temperature, pressure and time on nitrogen absorption in molten steel (after Dickie).

(*By courtesy of the Iron and Steel Institute*)

summarizes these factors and states that the amount of nitrogen absorbed, in limited time periods, is proportional to:

(1) The square root of the partial pressure, p, of the nitrogen gas in contact with the metallic charge.

(2) The square root of the time of contact, t.

(3) The fifth power of the absolute temperature, T.

(4) The surface area, A, of contact between gas and metal.

(5) The coefficient of diffusion, D, which is the percentage of nitrogen absorbed at standard temperature T_s when $P = 1$ $t = 1$, $A = 1$.

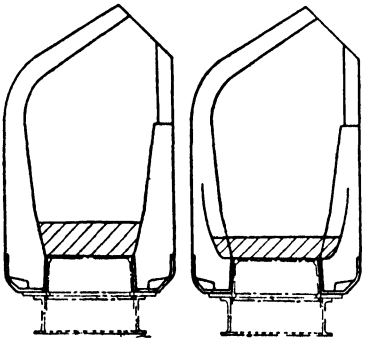
Combining these factors, the percentage of nitrogen will be

$$Q = D\,(T/T_s)^5\,p^{\frac{1}{2}}\,t^{\frac{1}{2}}\,A$$

The diffusion of a gas into a substance is affected by the nature of the surface. In the case of molten steel an oxide film on the surface or a slag covering will retard the absorption of nitrogen.

It is shown that in practice the pressure and time of contact can be reduced concurrently by decreasing the depth of the bath. The effect of a worn lining and a shallow bath of metal on the nitrogen of the steel has already been discussed. The depth can be reduced by increasing the diameter of the converter at the bottom, as shown in Fig. 20A. The type of converter shown in Fig. 20A, which is recommended by Dickie, gives much smaller depth of metal as compared with the design shown in Fig. 20B. It will be noted that the lining of this new-type converter is considerably thinner at the lower part.

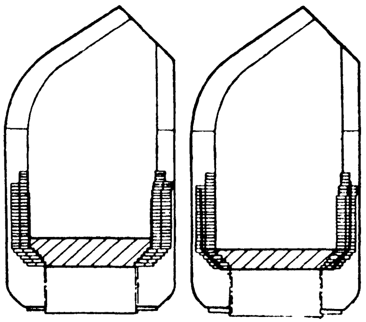
In order to obtain satisfactory life from this thinner lining, the usual rammed dolomite mixture is replaced in the lower portions by a refractory brick construction. In the lining described by Dickie the refractory bricks employed are dolomite and they are giving a satisfactory life.



(*a*) (*b*)

FIG. 20A. Bessemer converter with conventional lining (*a*) start of first bottom, bath 33 in. deep, 8 ft. 2½ in. maximum width: (*b*) wear of lining up to start of fourth bottom (after Dickie).

(By courtesy of the Iron and Steel Institute.)



(*a* (*b*)

FIG. 20B. New type brick lining in shallow bath converter (*a*) start of first bottom and (*b*) wear during life start of fourth bottom (after Dickie).

(By courtesy of the Iron and Steel Institute)

It has also been found that the nose of the converter must be kept open in order to avoid a back pressure. If the nose becomes unduly restricted, the resulting back pressure created causes a rise in the nitrogen content of the steel.

The partial pressure of the nitrogen in contact with the charge can also be reduced by additions of oxides to the charge, or by enrichment of the blast with oxygen, or by the introduction of both these refinements concurrently. The oxide can be used instead of the steel scrap frequently added to cool the metal. In practice it has been found that about 50 per cent. more steel scrap than oxide can be employed to produce the same cooling effect. The use of oxide has shown a considerable decrease in the nitrogen content of the finished steel.

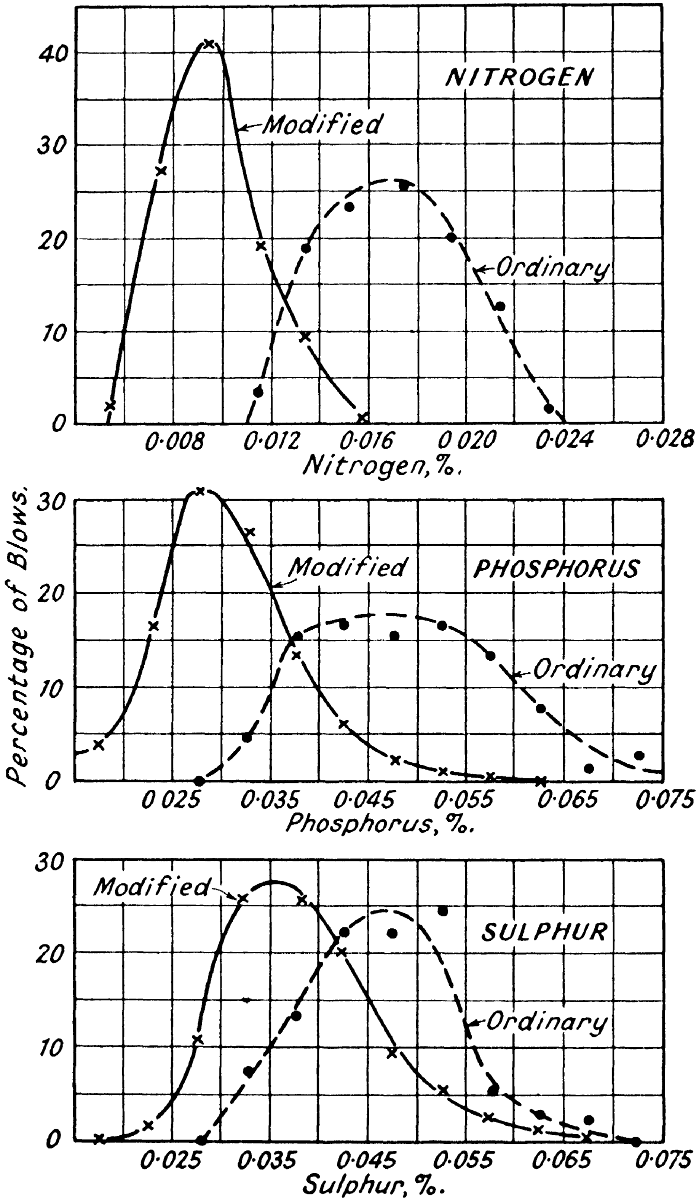


FIG. 21. Effect of Dickie's modified process.
(*By courtesy of the Iron and Steel Institute.*)

The effect of a more shallow bath and the employment of oxides are shown graphically in Fig. 21, whilst Table VI indicates the improvement in quality resulting from this practice. A single oxide addition, averaging 3,000 lb. to a 25 to 26 tons capacity wide-bath converter, has resulted in lower nitrogen contents. This addition replaces the normal scrap addition and is made during the fore-blow.

Oxygen Enrichment of the Blast

The use of oxygen-enriched blast is mentioned by Dickie.[8] Most of the work connected with oxygen enrichment has been performed on side-blown converters and is reviewed in Chapter 11.

The use of oxygen-enriched blast will obviously result in a lowering of the blowing time and increased output. In Bavarie[9] it has been found that a blast containing 30 per cent. of oxygen gave the best results. Any increase beyond this concentration did not result in any reduction of the blowing time or in any increase in the final steel temperatures.

When converting a pig iron containing 0·3 per cent. to 0·5 per cent. Si, 1·5 per cent. to 1·95 per cent. P, and 0·72 per cent. to 0·95 per cent. Mn, it was found possible to employ steel scrap up to the extent of 12 per cent. of the total charged weight. The final dephosphorization was achieved by the use of air only. The enriched blast, containing 30 per cent. of oxygen, was employed during the earlier stages which were reduced from 11½ minutes to 7½ minutes.

Since the employment of enriched blast results in a reduction of the partial pressure of nitrogen, a decrease in the nitrogen content of the finished steel may be anticipated, with a corresponding improvement in the properties.

Use of Pre-Heated Blast

As already stated the ordinary acid Bessemer process requires molten metal high in silicon in order to maintain a sufficiently high temperature, whilst in the basic Bessemer process a high phosphorus content is necessary for the same purpose. In some parts of the world, for example in India, the natural pig iron falls between these two requirements. In order to meet these conditions, Malik and Alvi[10] took out patent rights for the use of hot blast.

Instead of increasing the phosphorus content of the pig iron by additions of ferro-phosphorus or by the inclusion of phosphate rocks in the blast-furnace burden, they recommended pre-heating the blast supplied to the converter. They suggest that the blast could be preheated by passing it through blast-furnace stoves previouslv heated with blast-furnace gas or some other suitable fuel. The temperature of the blast can be varied from 200 to 600 degs. C. according to the iron being converted.

When working a typical Indian pig iron containing:

> C 3·75 per cent., Si 0·75 per cent., P 0·035 per cent. and Mn 0·8 per cent.

they recommend a blast temperature of 600 degs. C. In the case of a pig iron with a higher silicon content, namely over 1·0 per cent. but below 2·0 per cent., cold blast is advocated during the early part of the blow, followed by hot blast to maintain a satisfactory casting temperature until the end of conversion. The blast temperatures and thermal balance sheets of two typical blows given in their specification are reproduced in Tables VIII and IX.

So far as the author is aware, the process set out in detail in this patent has not been applied in practice. It would appear that oxygen enrichment of the blast may serve a similar purpose.

TABLE VIII

SUMMARY OF THERMAL CHANGES. (Malik & Alvi[10])

ANALYSIS OF IRON BLOWN

C	Si	Mn	P
3·75%	0·75%	0·8%	0·35%

BLAST TEMPERATURE 600° C.

HEAT INPUT

Heat in melted pig iron = 30·420 × 10^6 Cals.
Heat in melted preheated lime = 00·309 × 10^6 ,,
Heat in melted hot blast = 7·440 × 10^6 ,,
Heat from oxidation = 23·920 × 10^6 ,,
Heat from formation of slag = 1·110 × 10^6 ,,

Total .. 63·199 × 10^6 Cals.

HEAT EXPENDED

Heat in finished steel = 32 × 10^6 Cals.
Heat in finishing slag = 5·94 × 10^6 ,,
Heat in escaping gases = 23·90 × 10^6 ,,
Heat conducted to the air = 00·10 × 10^6 ,,
Heat radiated = 00·06 × 10^6 ,,

Total .. 62·00 × 10^6 Cals.

TABLE IX

SUMMARY OF THERMAL CHANGES. (Malik & Alvi[10])

ANALYSIS OF IRON BLOWN

C	Si	Mn	P
3·75%	0·5%	0·9%	1·75%

BLAST TEMPERATURE 100° C.

HEAT INPUT

Heat in melted pig iron = 30·420 × 10^6 Cals.
Heat in preheated lime = 1·545 × 10^6 ,,
Heat in blast = 1·370 × 10^6 ,,
Heat from oxidation = 30·660 × 10^6 ,,
Heat from formation of slag = 4·700 × 10^6 ,,

Total .. 68·695 × 10^6 Cals.

HEAT EXPENDED

Heat in finished steel = 32·00 × 10^6 Cals.
Heat in finishing slag = 10·20 × 10^6 ,,
Heat in escaping gases = 27·20 × 10^6 ,,
Heat conducted to the air = 00·10 × 10^6 ,,
Heat radiated = 00·06 × 10^6 ,,

Total .. 69·56 × 10^6 Cals.

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[5] *Iron and Coal Trades Review*, 20th Aug., 1948.
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[8] DICKIE. *Bulletin I. & S. Inst.*, Aug. 1948.
[9] F.I.A.T. *Final Report No.* 1203.
[10] MALIK and ALVI. Patent Spec. No. 431, 260, 1st July, 1935.

CHAPTER 4

THE OPEN-HEARTH PROCESS

Historical

Whilst Bessemer was developing his process, two young German engineers called Siemens were conducting experiments which, although remote from the subject at the time, eventually led to the beginning of the open-hearth process.

The early open-hearth process, like the original Bessemer process, was designed to refine pig iron high in silicon but low in sulphur and phosphorus. In other words, it was an acid process. Later, with the discovery of basic linings, the basic open-hearth process, which has become the chief method of steel production, was introduced.

The furnace used by Siemens was a rectangular structure of the reverberatory type. The process consisted of melting pig iron and decarburizing it with iron ore. A similar process was developed by Martin, who used the same type of furnace. His process consisted of melting pig iron and scrap, whereby the carbon in the pig iron was diluted, and the silicon removed by oxide additions. Later the two processes were combined, so that the charge was made up of pig iron, scrap, and in some cases iron ore. This process was known for many years as the Siemens-Martin process.

General Principles

In the Bessemer process the removal of impurities is effected by the air blown through the molten metal. The oxygen of the blast oxidizes the impurities, carbon escapes from the converter as a gas, and the other oxidation products pass into the slag, which merely acts as a receptacle for these products.

In the open-hearth process the slag is the refining medium. The slag, therefore, in this process, has a twofold purpose:

(1) It is the oxidizing medium by means of which the impurities are removed.

(2) It is the receptacle which retains the oxidation products in a stable form.

The raw materials for the acid process are of a siliceous nature, but low in sulphur and phosphorus. The maintenance of an acid slag prevents the elimination of these elements. The pig iron and scrap are charged, and when melted the slag is rendered fluid and oxidizing by iron-ore additions. These ore additions oxidize the carbon, silicon and manganese present in the charge. The carbon escapes as carbon monoxide, whilst the silica and manganese oxide pass into the slag, which is virtually a ferrous-manganous silicate.

In the basic process a wider range of raw materials may be employed. A basic slag is formed by lime or limestone additions, and this slag effects the removal of sulphur and phosphorus. Siliceous materials should be avoided as far as possible, as excessive lime is required to neutralize the silica formed, and if the slag becomes too siliceous the removal of sulphur and phosphorus is hindered and the basic lining of the furnace is attacked.

In many cases cold or solid pig iron is charged into the furnace and the process is referred to as "the cold pig and scrap" process. At composite works, where blast-furnace metal is available, molten pig iron is employed. This is referred to as the "hot metal" process. Naturally, the employment of hot metal has many advantages, which will be discussed later. A considerable economy in fuel is obtained.

Whether the process is acid or basic, and whether hot or cold pig iron is used, the same type of furnace is employed. It is a reverberatory furnace as shown in Fig. 22.

The fuel used for melting the cold materials and maintaining the heat necessary for refining the metal is frequently producer-gas, which has to be regenerated to obtain a sufficiently high temperature.

The principle of regeneration depends on the passing of the waste products of combustion as they leave the furnace through chambers containing a honeycomb of brickwork. The chamber is hereby heated. After an interval the flow is reversed, so that the gas and air enter on the opposite end of the furnace and are thereby pre-heated as they pass through the hot regenerators. When a furnace has been in operation for some time, these regenerators will attain a temperature of 1,100° to 1,200°C., so that the incoming gas and air are pre-heated considerably. The general design of an open-hearth furnace for producer-gas firing is shown in Fig. 22.

It will be noted that in this diagram the stack is located between the air and gas reversing valves. This is an excellent arrangement, as both circuits can be provided with separate dampers, which allow the flow of waste products through either regenerator to be regulated. This enables the temperature of either regenerator to be controlled.

In composite iron and steel works, linked up with coke ovens, blast-furnace and coke-oven gas is frequently employed instead of producer-gas. In America, and to some extent in this country, oil firing has also been adopted.

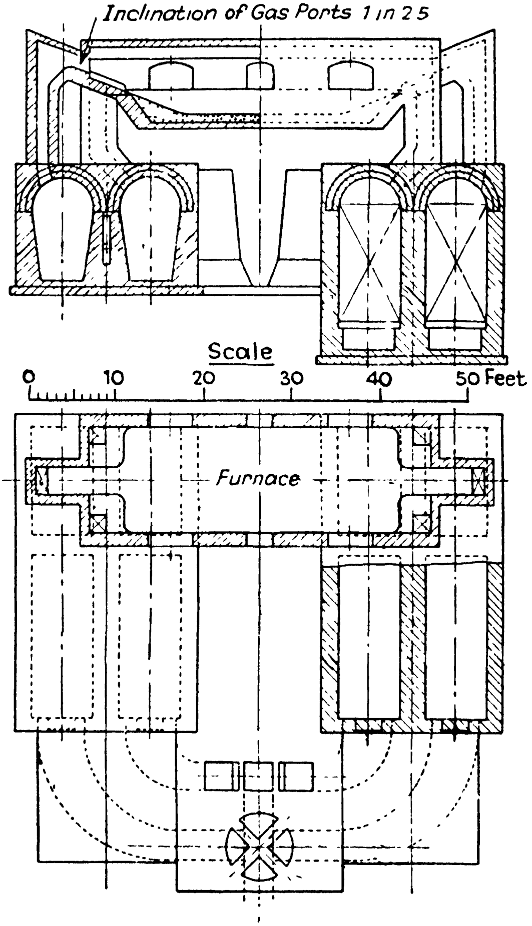


FIG. 22. Reverberatory furnace.

When using higher calorific value fuels, such as natural gas, coke-oven gas or oil, it is only necessary to pre-heat or regenerate the air.

Reversing Valves

A very important feature of the open-hearth furnace is the method of reversal. Various types of reversing valves are employed, the chief being:

Butterfly Valve

As shown in Fig. 23, this valve consists of a cast-iron or steel cover over three flues, the central flue leading to the stack, whilst the other flues lead to the left- and right-hand regenerator respectively. This casing is frequently lined with firebricks. The air or gas enters the casing through the top and is deflected by a tongue to one of the regenerator flues. The waste products, on leaving the furnace, pass up the opposite regenerator flue and are deflected by the underside of the tongue into the stack flue.

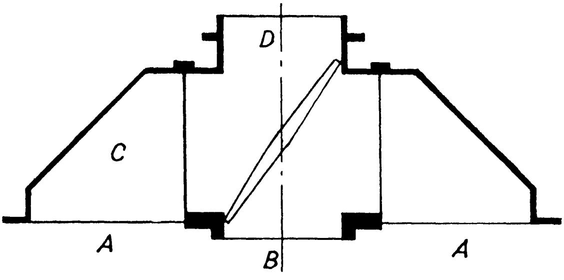


FIG. 23. Butterfly reversing valve. (See Table X.)

This type of valve may be used for either air or gas reversal, but in modern practice it is usually confined to the air circuit. The difficulty in maintaining a gas-tight seating of the tongue renders it unsuitable for gas reversal.

The tongues are very liable to warp with the heat to which they are exposed, and are, therefore, frequently water-cooled. Likewise the seatings are often water-cooled. In such cases, the tongue and seatings are hollow castings, generally steel castings, through which water is circulated freely.

The tongue is supported by a suitable axis, which at one end, outside the valve casing, is attached by a lever, by means of which the tongue can be reversed. This lever can be activated by a hydraulic cylinder or a suitable electrically operated rack.

The size of valve, which is a very important factor in controlling output, varies with the capacity of the furnace. It is essential that the valve, and the flues adjacent to it, are of sufficient capacity to avoid any restriction. This is of extreme importance in dealing with the waste products from the furnace. Typical dimensions of butterfly valves, suited to varying capacity of furnaces, are given in Table X.

TABLE X

Size	A	B	C	D	Capacity of Furnace
	ft. in.	ft. in.	ft. in.	ft. in.	Tons
30	3 3½	3 6	2 10	3 6	40
33	3 9	3 9	3 3	2 9	45
36	4 0	4 2	3 6	3 0	50
42	4 6	4 6	4 0	3 6	70
48	4 9	5 0	4 6	4 0	Over 70 tons

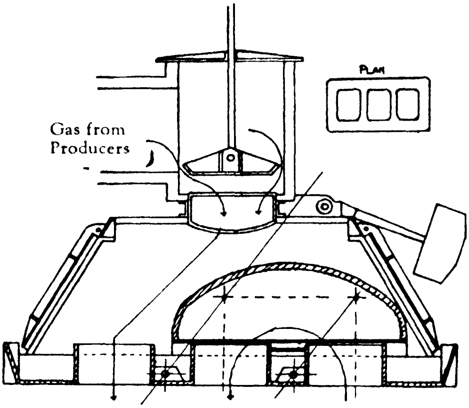


FIG. 24. Wellman hood-type valve.
(*By courtesy of the Staffordshire Iron and Steel Institute*)

The Hood Valve

The hood-type valve is similar in general principle and construction to the butterfly valve, excepting that the functions of the tongue are performed by a hood.

The top of the three flues is shown surmounted by a water trough, as shown in Fig. 24. The hood, which covers the central flue and one of the side flues, is supported in forks, and can be thrown over either end flue by suitable hydraulic or electrical gearing. The hood, when seated, fits into the grooves of the water trough, thus forming a water seal.

When the valve is set, as shown in Fig. 24, the incoming gas or air enters through the top of the valve, as in the case of the butterfly, and passes down the left-hand flue to that regenerator. The waste products leaving the furnace through the right-hand regenerator rise into the hood and are conveyed to the stack.

This type of valve can be used for either gas or air reversal. One of the chief troubles connected with this valve is defective forks. When used on the gas circuit, tar is likely to accumulate in the water trough, causing the hood to seat badly. This trouble can, however, be alleviated by provision of inspection and clearing holes at the end of the various sections of the water trough.

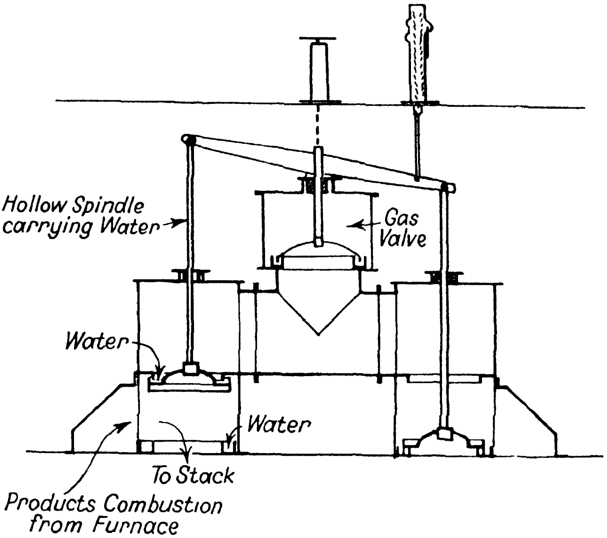


FIG. 25. Water-cooled gas reversing valve.

The Wellman valve is admirable for air reversal, especially on furnaces using enforced air.

Mushroom Valve

Valves of the mushroom type, the general principles of which are shown in Fig. 25, have been largely superseded owing to the warpings of the mushrooms and their seatings. Where these valves are still in use, water-cooled mushrooms and seatings are usually provided.

Lake Valve

The Lake valve is one of the most popular valves in this country for gas-reversal. It consists of a brick-lined hood, which is divided into two compartments, and rotates in a circular trough. The base of the valve is divided into four segments, one leading to the stack, one to the gas supply and the other two to the regenerators. This is surmounted by a casting, which forms a water seal with the hood. The outer rim of the hood is deeper than the central partition, so that during reversal the hood may be lifted clear of the central cross-section of the trough, without breaking the water seal at the circumference.

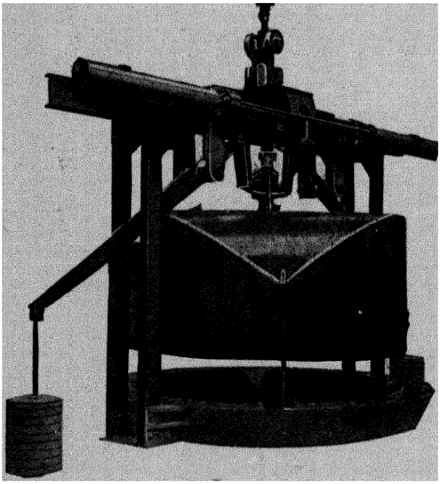


FIG. 26. Lake gas valve.

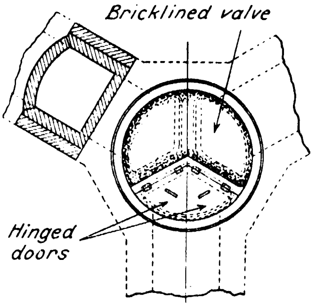


FIG. 27. Lake air reversing valve.

(*By courtesy of Messrs. Christmas & Walters.*)

The general design and construction of a Lake gas reversing valve, is shown in Fig. 26, whilst a modification of this valve for use on the air circuit is given in Fig. 27.

Normally the hood is lined with firebricks and provided the water seals or troughs are kept free from tar deposits, little or no trouble should be experienced. Inspection holes to facilitate cleaning should be provided at each end of the central troughs.

The reversing mechanism can be operated either by an electrically activated rack or hydraulic gearing. The arrangement for hydraulic operation is shown in Fig. 28. The tapered casting is moved by the hydraulic cylinder, causing the small wheel which is supporting the hood to travel upwards, thereby lifting the central partition of the hood clear of the water seals. When the hood is raised sufficiently, the V-shaped projection on the stem attaching the hood to the small wheel is engaged by a finger operating with the hydraulic piston, thereby throwing the hood on the opposite end.

It will be noted from the diagram of this valve, that the weight of the hood and its lining is counterbalanced by a system of weights and levers.

The correct setting of the valve is of supreme importance. The lift of the hood during reversal should be sufficient for the central partition to clear the water seal, but an excessive lift will result in loss of gas. The water seal should be deep enough to prevent any leakage of gas or any siphoning to take place.

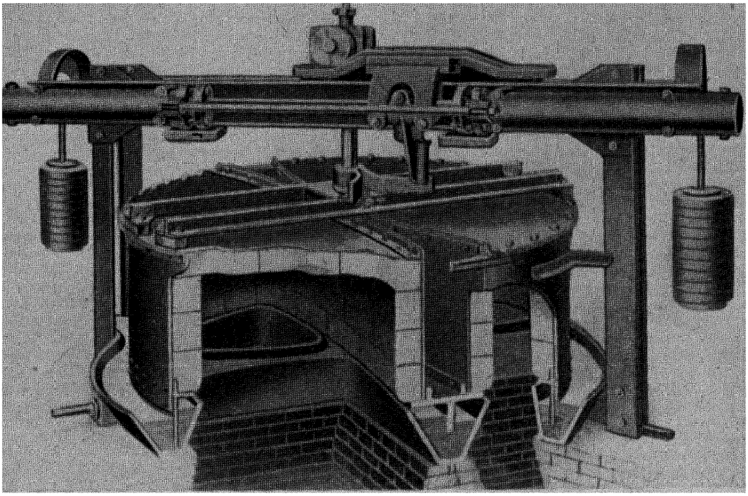


FIG. 28. Lake or Dyblie reversing gear.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

Damper Valves

During recent years damper valves, designed on the straight-through principle in which bends and sharp corners are avoided, have found great favour, especially in America. This type of valve is favoured on large-capacity furnaces. When properly designed and arranged, preferably provided with water-cooled dampers, they have proved very successful.

The air reversing valve, as shown diagrammatically in Fig. 29, consists of four haematite cast-iron seats which are bricked into the flues. The two lower seats, which allow the waste gases to pass to the stack, are water-cooled, but normally the upper seats, which act as air inlets, are effectively cooled by the incoming air. The dampers which close these openings are generally water-cooled. The dampers can be operated either hydraulically or electrically.

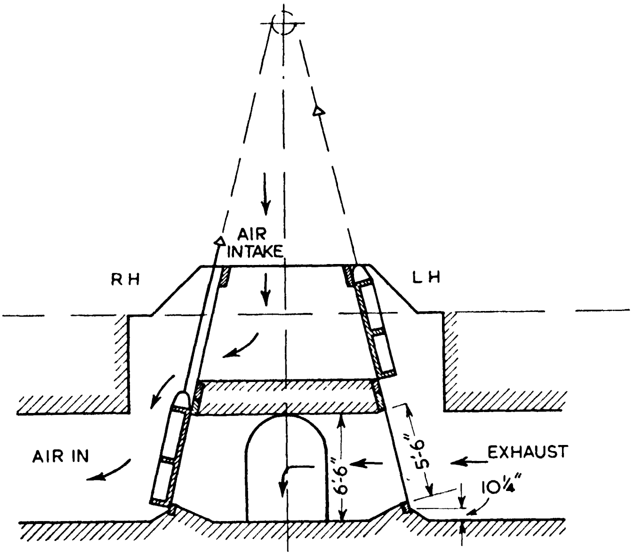


FIG. 29. Damper-type air valve.
(*By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd*)

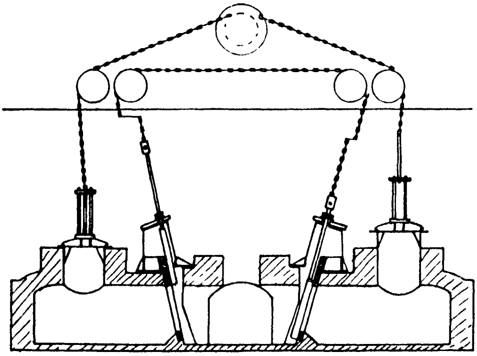


FIG. 30. Damper-type gas valve.
(*By courtesy of the Staffordshire Iron and Steel Institute*)

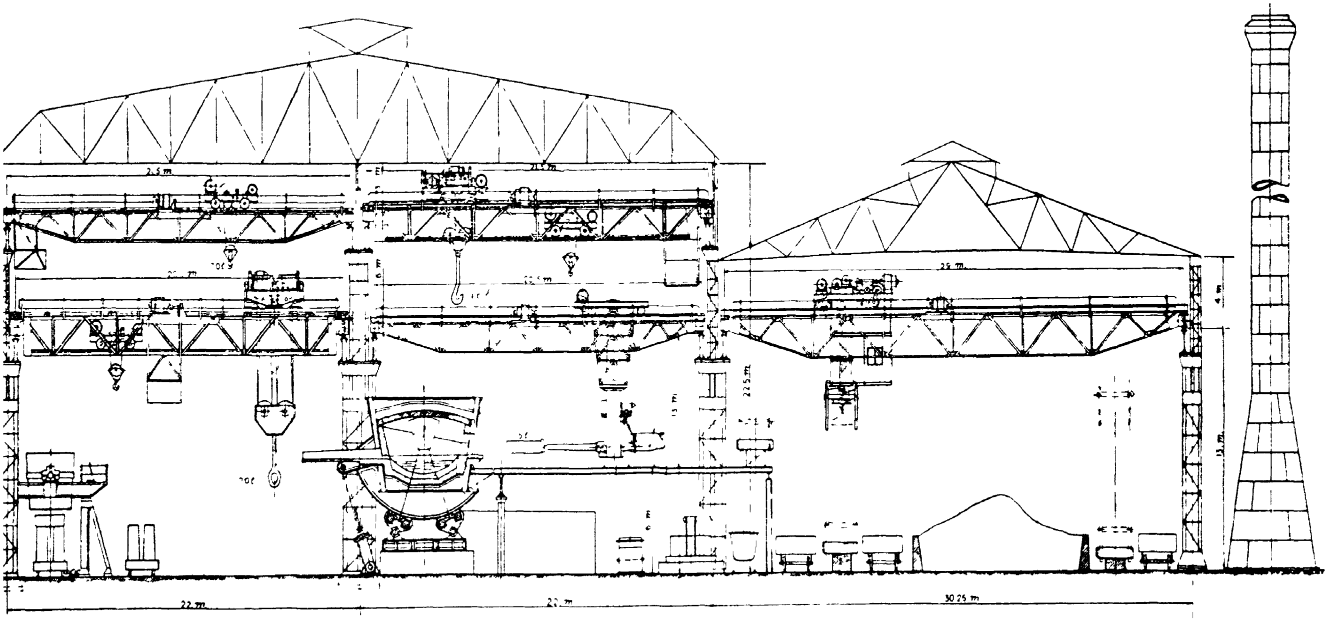


FIG. 31. Melting-shop layout, showing revolving overhead type charging machine and box-lifting device on magnet cranes.

(*By courtesy of the Iron and Steel Institute*)

The dampers are raised and lowered alternately. When one damper is raised, thereby closing the air inlet and allowing the waste products to pass to the stack, the other damper is lowered, closing the outlet to the stack flue and allowing air to flow into that regenerator. The slope of the dampers assures an air-tight junction with the seating.

When used for reversing the gas, the dampers are employed to regulate the flow to the stack flue only. The gas inlet is controlled by a water-sealed mushroom valve, either side of the system, as shown in Fig. 30. This type of valve is equally adaptable for use with either a mixture of coke-oven and blast-furnace gas or with producer-gas. Sometimes this mushroom inlet type of valve is also used on the air system.

To some extent the types of control and reversing valves are best selected in conjunction with the kind of fuel employed. This aspect is referred to in Chaper 5, when the various classes of open-hearth fuels are discussed.

Layout of Plant

The general layout of an open-hearth melting shop is shown in Figs. 31, 32 and 33.

It will be noted that in Figs. 31 and 32 the *stockyard* for raw materials is immediately behind the furnace platform. The raw materials are loaded either from stock or wagons into charging boxes, by magnet cranes. The charging boxes are lifted on to the furnace gantry by means of box-lifting cranes. In Fig. 31, the box-lifting crane is provided with an automatic device for engaging the boxes, thus removing the need of labour for hanging on the pans, as necessary in Fig. 32.

The arrangement of the stockyard adjacent to the furnace platform assures a steady and regular supply of raw materials to the furnaces, thus speeding up the charging operations. Unless the stockyard is of a wide span, however, stocking space is restricted. A careful layout of railroads in the stockyard is also necessary, otherwise delays are likely to occur when it is necessary to replace empty wagons.

When the stockyard is separated from the furnace platform, as in Fig. 33, the loaded pans are brought on to the platform on bogies. This method of supplying raw materials is naturally somewhat slower, and since locomotive service is necessary to shunt the bogies between the furnace platform and the stockyard it is more costly. The modern tendency, therefore, is to have the stockyard adjacent to the furnace platform.

Frequently auxiliary stocking bays are provided. In some cases the materials from these bays are run on to the furnace platform on bogies,

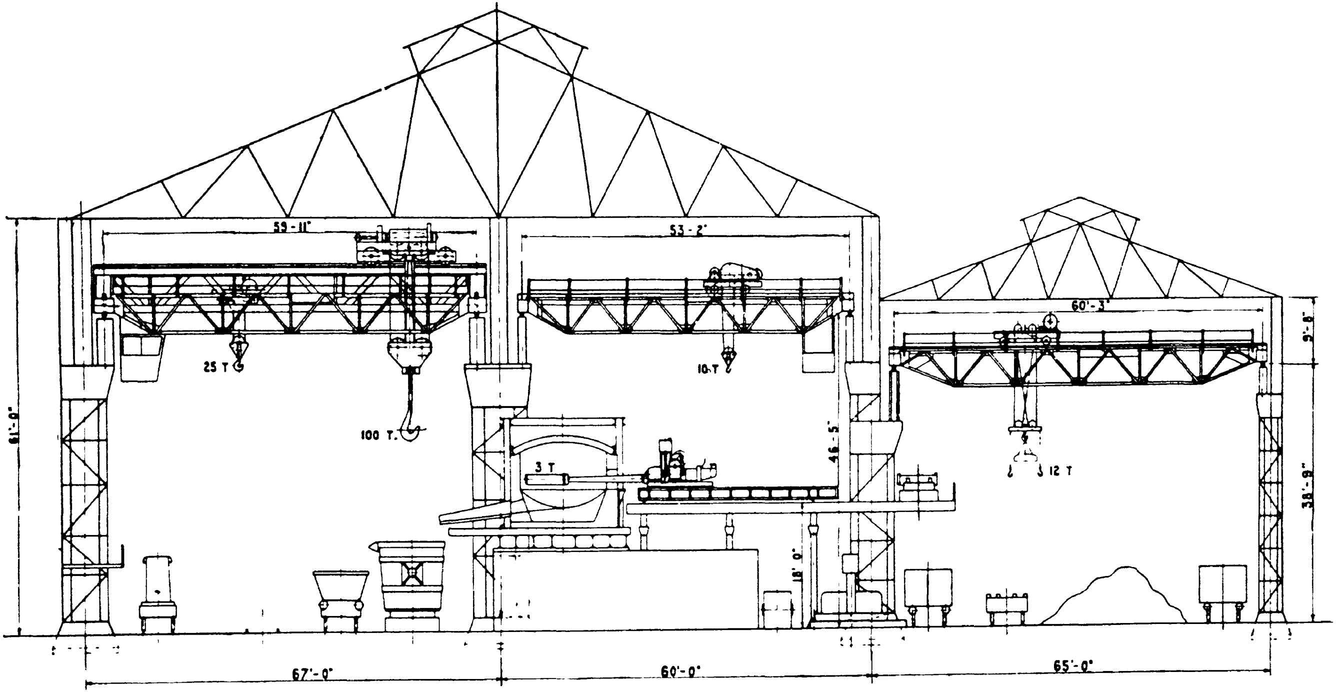


FIG. 32. Melting-shop layout, for small outputs, showing ground type revolving charger.
(*By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd*)

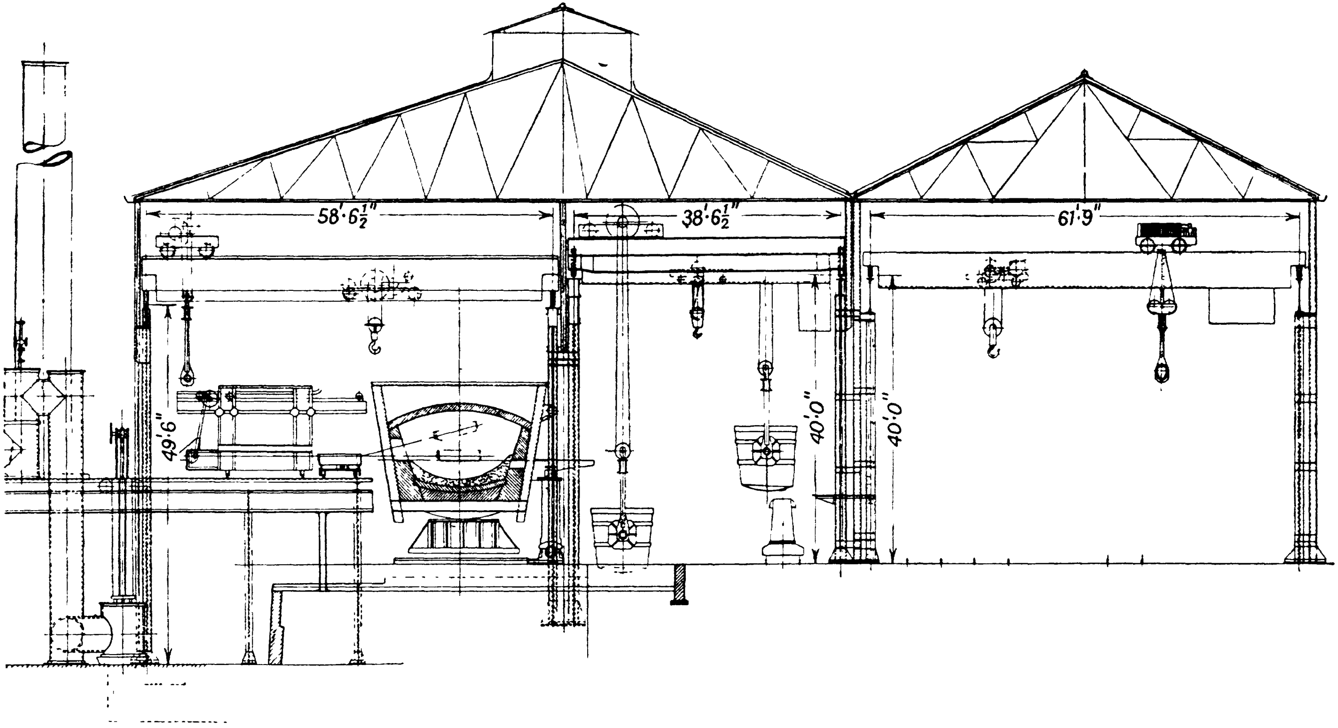


FIG. 33. Meltıng-shop layout, showing ground-type charger.

(*From " Symposium on Steelmaking ", by courtesy of the Iron and Steel Institute.*)

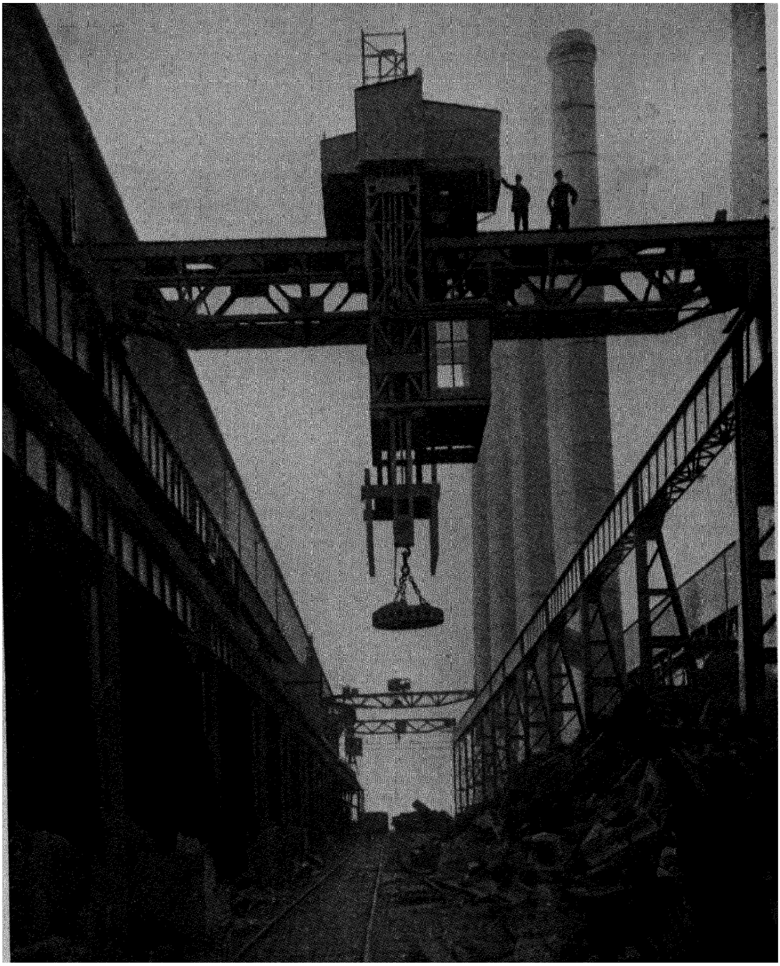


FIG. 34. Raw material stockyard and cranes.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.)

similar to that in Fig. 33, in other cases they are transferred, either by specially designed wagons or by bogies, to the main stockyard, and lifted onto the furnace gantry by the box-lifting cranes.

The charging boxes or pans vary in size, according to the size of the furnace doors and the capacity of the cranes employed. A fairly popular size is 6 ft. 6 in. long, 2 ft. 9 in. wide and 1 ft. 9 in. deep. A pan of this size will hold about 35 cwt. of pig iron. These charging boxes are shown in Fig. 35, and are usually built up of ½-in. steel plate around a steel casting, which is designed so as to allow the pan to be engaged by the ram of the charging machine.

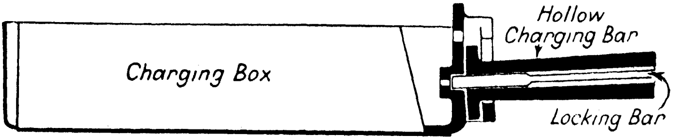


FIG. 35. Charging box or pan.

Naturally when using large quantities of scrap, especially the lighter varieties, the largest possible pan size is desirable to reduce the number of pans which must be handled.

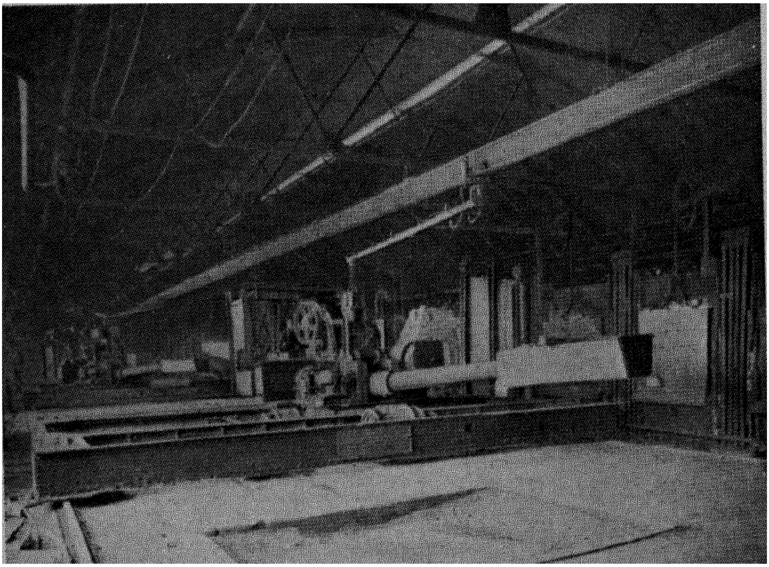


FIG. 36. Revolving-ground type charger.
(From "Practical Steelmaking", by W. Lister, by courtesy of the author.)

Three types of *charging machines* for handling these boxes are in use:

(1) Revolving ground type, as shown in Fig. 36.
(2) Revolving overhead type, as shown in Fig. 37.
(3) High ground type, as shown in Fig. 38.

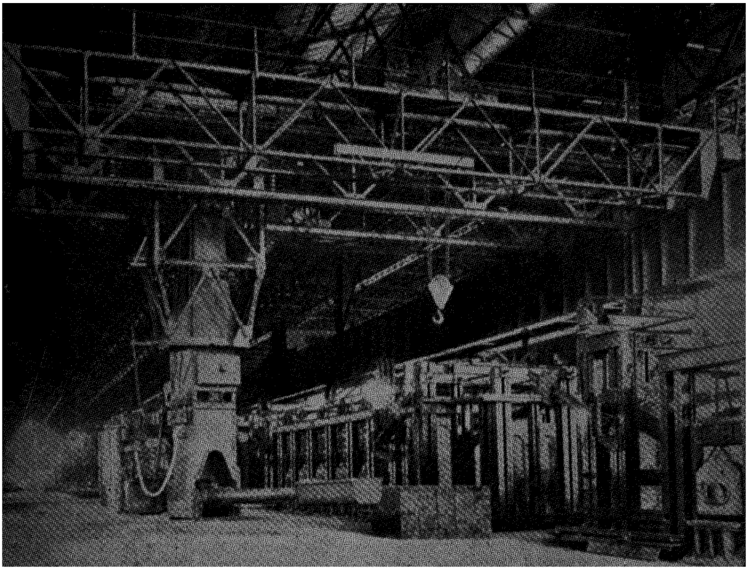


FIG. 37. Revolving overhead type charging machine.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

When the charging boxes are placed on a gantry from an adjacent stockyard, as in Figs. 31 and 32, one of the revolving types is employed. When the raw materials are brought up on to the stage on bogies

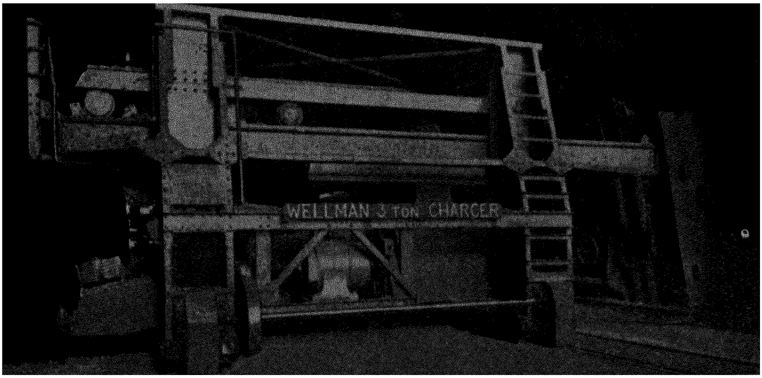


FIG. 38. A 3-ton high ground type charging machine.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.

immediately in front of the furnace, as in Fig. 33, a high ground type may be used.

The high ground type appears more robust, and requires, in general, less maintenance. The overhead revolving type, however, permits more manoeuvring. With the high ground type there is a tendency for the materials to be charged in heaps opposite each charging door. The revolving types of charging machine allow the materials to be spread more evenly in the furnace.

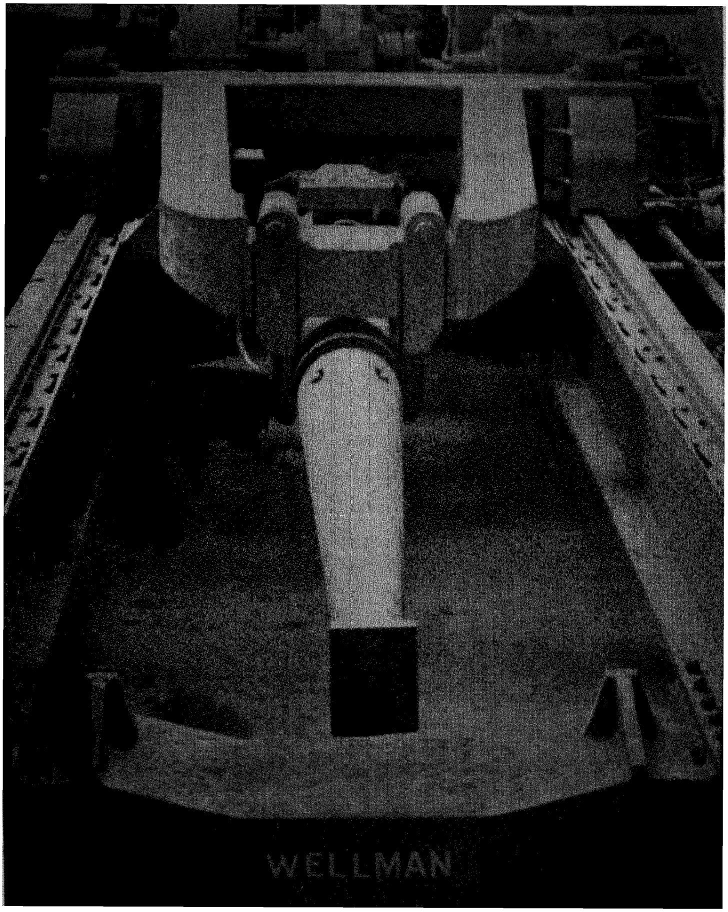


FIG. 39. Ram and locking bar mechanism.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.)

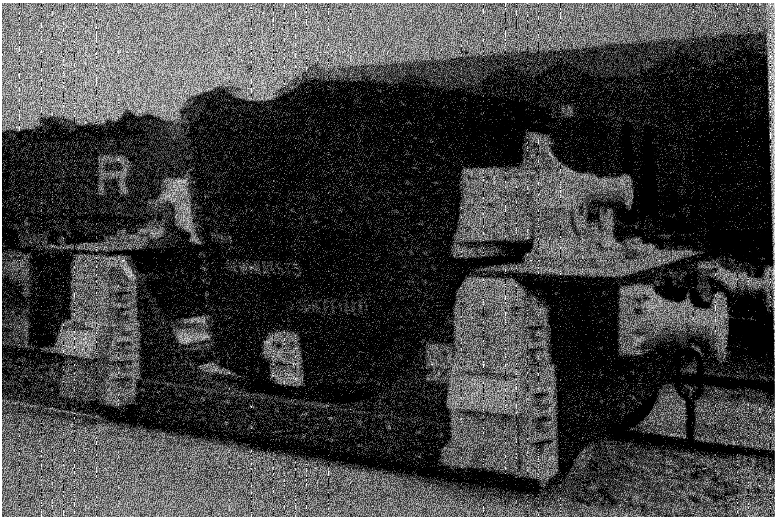


FIG. 40. Hot metal ladle.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

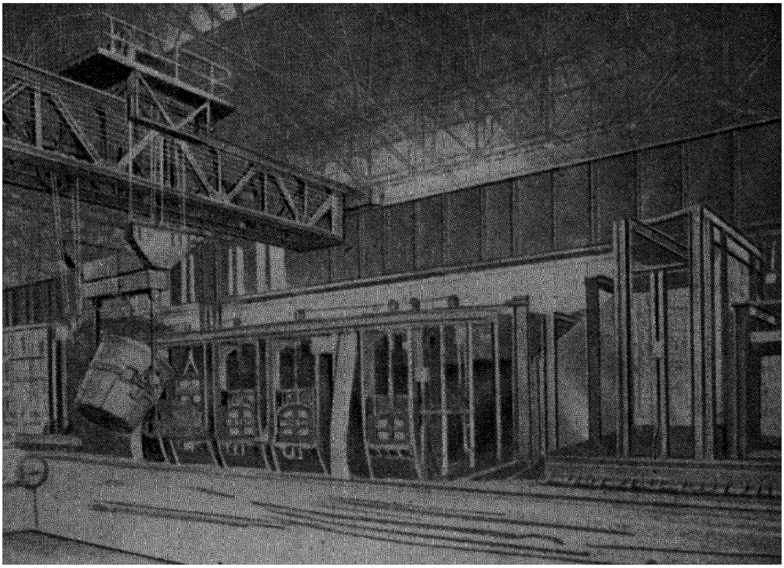


FIG. 41. Charging hot metal.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

The essential principle of each type is the same. The machine is fitted with a ram, which can be rotated. The end of the ram is inserted into the socket of the charging box and is secured by a locking bar, which runs down the centre of the ram. This device is shown diagrammatically in Fig. 39.

When hot metal is employed, it is transferred either direct from the blast furnace or from the mixer, in firebrick-lined ladles. The cranes for handling these ladles can be over either the furnace or casting bay. A chute is inserted into the furnace, through one of the doors, and the metal poured into the chute. This arrangement is shown in Fig. 40.

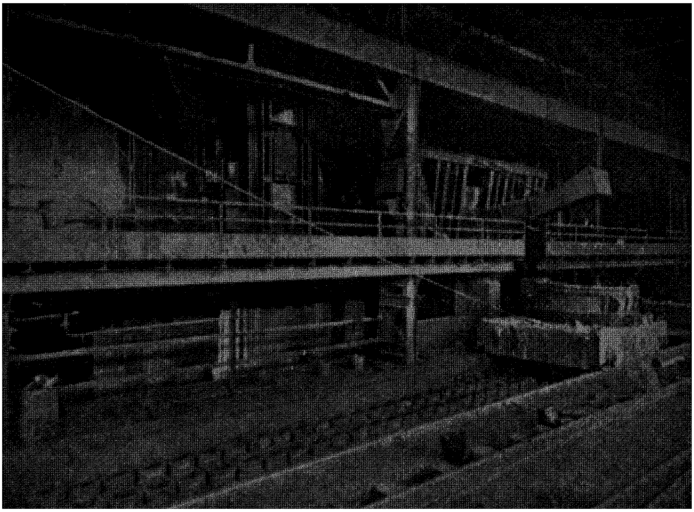


FIG. 42. Sunken pitside.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

Disposal of the Products

The general arrangement of the casting pit, which will be discussed in detail in Chapter 10, is shown in Figs. 31 to 33. Ample space in the casting bay is of paramount importance. In many works the moulds are prepared and set up in a separate bay, the casting bay being used only for the actual teeming of the metal. This method is very desirable in shops where different types and sizes of ingot moulds are employed.

Modern practice favours car-casting with the ladle held by an overhead crane. The crane carries the ladle over the ingot moulds, which are set on specially designed bogies. In some cases the ladle is placed in a stand and the bogies are moved under the ladle by means of a hydraulic or electrically operated rack.

In some old-fashioned shops sunken pits still persist. The general construction of a sunken pit is shown in Fig. 42. The slag overflows into slag pans, which are set at one end of the pit and separated by a low wall from that portion of the pit occupied by the ingot moulds. This type of pitside is costly to maintain, and if it runs parallel with the furnace, breakouts may be a serious problem, as the metal flows into this well and forms a massive lump, frequently fastening the ingot moulds together. These points are more fully reviewed, however, in Chapter 10.

In modern shops employing crane casting, the slag is allowed to overflow the ladle into slag ladles. When tilting furnaces are used, slag ladles are generally run underneath the furnace and the slag is poured off from the charging side of the furnace.

General Aspects

When designing a melting-shop layout, the flow of materials must be carefully considered if maximum outputs are to be maintained and restrictions avoided. Railroads in the stockyard and pitside should be so arranged that marshalling does not interfere with the whole operation of the bay. This can be done by the insertion of cross-overs, but if the cross-overs become too numerous, useful space is lost.

Modern design favours large overhead bunkers for handling lime and iron ore. This method enables these feed materials to be unloaded from wagons and stored in bulk in such a manner that the operation of a door allows them to be discharged in charging boxes, in which they are transferred to the furnaces.

The output of any melting shop is very largely a function of the speed with which raw materials, including feed materials, are sent up to the furnaces. Production may also be influenced by the rate at which the products and by-products are disposed of. Therefore the flow system requires careful planning.

CHAPTER 5

OPEN-HEARTH FUELS

Introduction

The fuel used originally for firing open-hearth furnaces, and even the chief fuel to-day for that purpose, is *producer-gas*. In some parts of America where supplies are available, natural gas is used in open-hearth furnaces.

In composite iron and steel works, where supplies of *coke-oven* and *blast-furnace gas* exist, these gases, or mixtures of them with or without producer-gas, are frequently employed in steel plants.

Probably the most recent development is the use of oil and tar-oils. The use of oil was first introduced in America, maybe due to abundant supplies at a low price, but during recent years the practice has been started in this country. Sometimes oil, tar or tar-oils are used as the only fuel, although at many plants it is a supplementary fuel. In some cases, they are used as an illuminant in conjunction with coke-oven gas. At other plants, they are employed to increase the flame intensity of lower calorific value fuels.

PRODUCER-GAS

Producer-gas is produced by the incomplete combustion of coal by a mixture of air and steam. Two main reactions are involved—air-carbon and steam-carbon reactions.

Air-Carbon Reaction

When air is blown through a column of incandescent carbon, the oxygen of the air combines with carbon, forming carbon dioxide, thus:

$$C + O_2 = CO_2$$

As the carbon dioxide thereby formed passes through further incandescent carbon, the carbon dioxide is decomposed with the formation of carbon monoxide:

$$CO_2 + C \rightleftharpoons 2CO$$

This is a reversible reaction which is readily influenced by temperature.

Rhead and Wheeler[1] advanced the theory that the first product of combustion of carbon was an unstable compound of carbon and oxygen of the unknown formula C_xO_y. Furthermore that this compound immediately decomposed forming carbon dioxide and carbon monoxide. The relative amounts of carbon dioxide and carbon monoxide are largely a function of temperature.

When carbon combines with oxygen, heat is evolved thus:

$$C + O_2 = CO_2 + 14{,}520 \text{ B.Th.U. per 1 lb. of carbon.}$$
$$2C + O_2 \rightleftharpoons 2CO + 4{,}410 \text{ B.Th.U. per 1 lb. of carbon.}$$

It therefore follows that when the CO formed from 1 lb. of carbon burns to CO_2, a further 10,090 B.Th.U. are evolved.[2] This means that the partial combustion of carbon, with the formation of carbon monoxide, by air in a gas producer increases the temperature of the fuel bed to the extent of 4,410 B.Th.U.s per lb. of carbon absorbed.

As already stated, the amount of carbon dioxide and carbon monoxide formed depend on the reaction:

$$CO_2 + C \rightleftharpoons 2CO.$$

As the temperature of the reaction zone increases, so the velocity of the reaction in the direction of the upper arrow increases. Wheeler[3] shows that when the temperature of the reaction exceeds 1,050 deg. C., the amount of carbon dioxide in equilibrium is negligible.

Wheeler also shows that:

(1) The lower the temperature of the fuel bed, the higher will be the carbon-dioxide content of the gas produced.
(2) At constant temperature, the carbon-dioxide content of the gas increases with the pressure, and therefore with the oxygen content of the air blast.

Discussing these reactions in a very able paper on Producer Practice, E. Wood[4] states that in order to obtain the maximum carbon-monoxide content in the gas, by the action of air on carbon:

(1) The temperature of the fuel bed should be high. That is, between 1,150 to 1,200 deg. C.
(2) The fuel bed should be porous and of small-size material.
(3) The blast pressure should be low, and consequently the speed of the blast through the fuel bed should be slow.
(4) The fuel bed should be sufficiently thick to allow the completion of the reactions.

Steam-Carbon Reaction

When steam is passed through incandescent carbon, it is decomposed with the formation of hydrogen and carbon monoxide, thus:

$$C + H_2O \rightleftharpoons H_2 + CO - 5{,}885 \text{ B.Th.U. per lb. of carbon.}$$

Therefore, the passage of steam through the producer cools down the fuel bed.

According to Taylor[5] at a temperature of 600 deg. C., the principal steam-carbon reaction is:

$$C + 2H_2O \rightarrow CO_2 + 2H_2.$$

As the temperature increases the amount of carbon dioxide decreases, until at a temperature of 1,000 deg. C., the reaction becomes

$$C + H_2O \rightarrow CO + H_2.$$

He further states that an excess of steam favours the formation of carbon dioxide.

Air-Steam-Carbon Reaction

Although a high temperature is desired in the fuel bed to produce the maximum carbon-monoxide content, an excessive temperature will cause the ash in the fuel to clinker. The temperature of the fuel bed can, however, be controlled by balancing the exothermic air reaction with the endothermic steam.

If the temperature of the fuel bed becomes too high, it may be reduced by the introduction of more steam in the air-steam blast. On the other hand the temperature can be increased by increasing the air/steam ratio.

Principle of a Gas Producer

A gas producer consists essentially of a steel shell, usually lined with firebricks. At the base of this shell, which generally stands in a water trough, is a cone or blast hood, through which the air/steam blast is admitted. In the roof, which covers the shell, is a hopper through which the raw coal is fed. The gas produced escapes through an off-take, normally at the side and near the roof of the producer. These principles are shown in Fig. 43.

In practice, many modifications in the design of gas producers are encountered, but the principle of them is fundamentally the same. The blast cone or hood is covered with ashes, which protect it from the heat of the combustion bed and also assist the distribution of the blast.

The solid material in the producer may be considered as forming two layers or beds—the ash bed and the fuel bed.

As indicated, the ash bed is of great importance. It has a twofold purpose—to protect the blast cone and to help the distribution of the blast. When starting up a producer, good clean ashes should be selected These ashes should be:

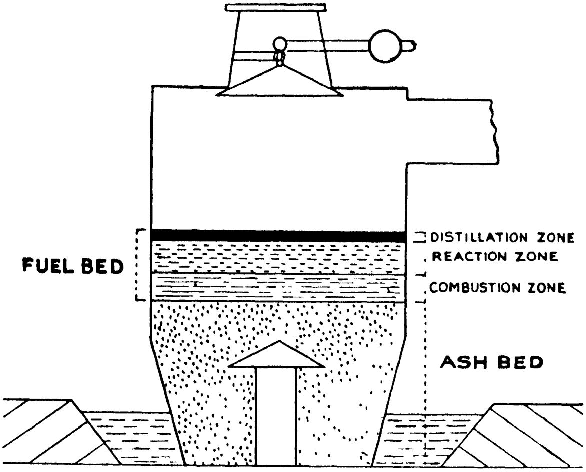


FIG. 43. Gas producer—zones of reaction.
(*By courtesy of the Staffordshire Iron and Steel Institute*)

(1) Well burnt. They should be free from any combustible matter, which may be set alight and thereby damage the blast cone and produce excessive heat in the base of the producer.

(2) Hard. They should be sufficiently hard or strong to resist the crushing action of the fuel column. If soft ashes are used, there is a likelihood of them being crushed to fines, thereby packing and causing bad distribution of the blast.

(3) Correct size. The ashes employed should be carefully sized. If they are too large, as pointed out by Wood,[4] this may lead to scaffolds being formed. If too fine, the free passage of the blast will be retarded. Fine dust may even choke the blast pipe.

Wood suggests that probably the most suitable size for these ashes is somewhere between the size of small and large nuts.

When making the ash bed in a new producer, the blast cone should be covered by several inches of ashes, and during the operation of the producer this covering must be maintained. In starting up a static producer, Wood[4] recommends that the ashes should be filled to a point about 18 inches above the blast cone. The bed should be levelled as the ashes are put into the producer.

After filling up the ash bed in a revolving producer, it is advisable to rotate the producer for several hours, to assist the settling of the ash bed. After this rotation it should be examined and any voids filled up. If the ashes have sunk too much, extra ashes should be added to adjust the height of the bed.

When the ash bed has been formed the producer is started up by lighting a coal fire on the ash bed and applying a light blast to kindle it. This fire is then fed with fuel, lightly at first, in the normal way. Gradually the blast pressure and the fuel feeds are increased until the producer is in normal operation.

The *fuel bed* may be divided into three zones:

(1) *Combustion Zone.* This zone lies next to the ash bed and is the hottest part of the producer. It is in this zone that most of the carbon monoxide is formed. Some of this carbon monoxide is converted to carbon dioxide by excess oxygen. The extent to which this takes place depends, as already stated, on the velocity of the blast and on the extent to which the fuel takes up oxygen.

(2) *Reaction Zone,* in which the carbon dioxide passing from the combustion zone reacts with further carbon to complete the reaction $CO_2 + C \rightleftharpoons CO$. As already shown, to maintain the maximum CO content in the gas, this zone must be maintained at a sufficiently high temperature.

(3) *Distillation Zone,* in which, as the name suggests, the volatile matter in the fuel is distilled.

The position of the ash and fuel beds, and of the three zones referred to, is shown in Fig. 43. It is essential in maintaining regular gas composition that the position and depth of these beds and zones are controlled.

Practical Operation of Producers

From what has been said it is obvious that the successful control of gas composition is dependent on:

(1) The correct depth of the various beds and zones of reaction.

(2) The correct temperature of these zones.

The best and quickest method, which is described in detail by Wood,[4] of determining the condition within a producer, is by means of test-rods. These rods are usually made of $\frac{5}{8}$-in. mild steel rounds, having a point at one end and a handle at the other. Their length is the distance from the poke-hole in the roof of the producer to the top of the blast cone. If inserted through the poke-hole in this way and left for a standard length of time, normally two or three minutes, on withdrawal they give a fairly accurate picture of the position and condition of the various beds and zones. In order to get a complete picture another type of rod is necessary. This is a rod of similar size with a small disk, which will rest on the top of the fuel bed, replacing the pointed end of the test-rod.

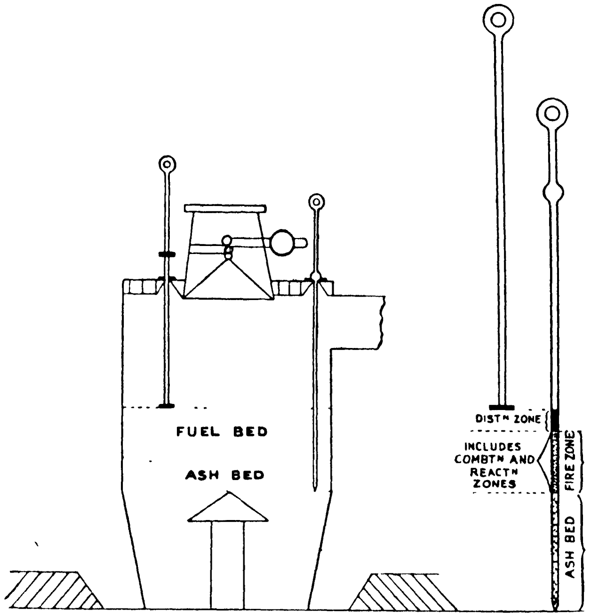


FIG. 44. Gas producer test rods.
(By courtesy of the Staffordshire Iron and Steel Institute.)

When the test-rod is withdrawn it should show a cold, or black-hot portion, which indicates the top portion of the ash bed. Then there should be a red or white-hot portion indicating the position and temperature of the combustion and reaction zones. The measuring rod will record the top of the fuel bed and the difference between this reading and the indicated position of the reaction zone will represent the distillation zone. This method, and the type of rods employed, is shown diagrammatically in Fig. 44.

If the ash bed becomes too thin the discharge of ashes must be arrested, or vice versa. If the fuel bed or distillation zone gets too thick, the feeding of raw coal must be retarded. A high blast pressure tends to reduce the thickness of the combustion and reaction zone. In order to obtain the most regular gas composition, the combustion and reaction zones should be at least six inches, preferably eight inches, thick. The thickness of the combustion zone is also affected by the saturation temperature of the blast, that is, the air/steam ratio.

In practice the combustion and reaction zones are frequently regarded as one, and referred to as the "*Fire Zone*". The temperature of the fire zone can be readily controlled by adjusting the *saturation temperature*, that is, the amount of steam introduced in the blast.

It is, however, impossible to lay down rigid rules as to the best saturation temperatures, which must vary with the type of coal employed and the operating condition of the producer. As pointed out by Wood,[4] at the works he discusses, two types of producers are employed, operating under different conditions, but using the same coal. At one plant the saturation temperature is 50 deg. C., whilst at the other it is 57 or 58 deg. C. Where the lower saturation temperature is employed the producers are never worked above a steam pressure of 25 or 30 lb. per sq. in., whereas the higher saturation temperatures are associated with blast pressures of 50 or more lb. per sq. in. The most suitable saturation temperature to meet the coal available and local conditions must be determined, and minor adjustments to it made according to the test-rods.

In order to get a maximum carbon-monoxide content in the gas, the fire zone should be maintained as high as possible, certainly in the neighbourhood of 1,000 deg. C. The actual temperature, however, will be dependent on the fusibility of the ash. If this temperature is exceeded, *clinker*, which will seriously interfere with the operation of the fire, will be formed. If the temperature of the fire zone becomes excessive, it may be checked by increasing the saturation temperature. The lowering of the saturation temperature will have the opposite effect. The approximate temperature of the fire zone may be judged from the test-rods.

The *top temperature* of the producer is also of importance, that is, the temperature of the gas leaving the producer. If this temperature runs too high, there is danger of the hydrocarbons being decomposed, giving rise to a thin, non-luminous gas. On the other hand, if the temperature of the outgoing gas is too low, there is a likelihood of tar deposition, resulting in the choking up of the gas mains and valves. This is particularly true when the producers are situated some distance from the furnaces in which the gas is to be used. The top temperature must be determined to meet local conditions. It can be adjusted by regulating the feeding of raw coal.

Normally the top temperature is best recorded by a base-metal thermo-couple inserted in the outlet of the producer. This thermo-couple is usually protected by a silica sheath.

The importance of *regular feeds* cannot be too strongly stressed, if constant gas composition and quality are to be maintained. Each feed immediately affects the amount of distillation gas evolved and the temperature of this zone. As already explained, an excessive rise in this temperature will result in the cracking of the heavy hydrocarbons and the production of a thin, non-luminous, or semi-luminous gas.

Whatever the method of feeding, *poking* or *agitation* of the fuel and fire zones is necessary. Each feed tends to collect in a cone-shaped pile, and requires levelling.

The blast tends to take the line of least resistance and if irregularities exist in the fire bed, *channels* will be formed. Unless these channels are removed by poking or agitation, two things are likely to happen:

(1) Intense combustion takes place in the vicinity of the channels, producing a very high temperature, resulting in the *formation of clinker.*

(2) Excess air passes through the channels burning some of the gas produced at the top of the producer, thereby altering its composition and calorific value.

Selection of Fuel

Although producer-gas can be made with the employment of a wide range of coals, the selection of fuel is of great importance if regular gas composition and operating conditions are to be maintained.

In general, bituminous coals, with a fairly high volatile content, are preferred, selected with due reference to the following properties:

(1) Caking and Swelling Properties. Strongly caking coals are undesirable for producer practice. They frequently cause channelling in the fuel bed, and thereby lead to clinker. Slight caking

properties give the best results, whilst high caking qualities result in reduced rates of gasification.

In districts where caking coals have to be employed, the operation of the producer can be improved by the judicious addition of non-caking coals, as pointed out by Wood.[4]

(2) Ash Content. The ash content should not be excessive. On mechanical producers Wood,[4] however, expresses a preference for a coal with 7 to 8 per cent. ash, rather than one with only 3 to 4 per cent. If the ash content is too high, it means that non-combustible matter is being paid for, and this material must later be disposed of at a cost. It also means that more cleaning is necessary in the case of static producers.

On the other hand, on mechanical producers, a very low ash content may render the maintenance of a good ash bed difficult.

(3) Fusion point of ash. Iron lowers the fusion point of the ash considerably. Any Fe_2O_3 present is reduced in the reducing atmosphere of the producer, with the formation of FeO. This lower oxide of iron readily combines with any silica present to form very fusible silicates.

In general, ashes consisting of silica and alumina have high fusion points, but lime and magnesia lower this temperature.

Sulphur is undesirable for two reasons:

(*a*) With over 1 per cent. sulphur in the ash, coals have a tendency to form clinker.

(*b*) The sulphur is carried over in the gas and may lead to sulphur pick-up by the metal in the furnace.

It is obvious that the lower the fusion point of the ash, the greater will be the likelihood of clinker formation.

(4) Moisture Content. The moisture content should be kept low. It is, however, better to contend with a slightly high moisture content, and use washed coal, than deal with the troublesome ash problems of an unwashed fuel.

(5) Volatile Matter. The volatile content does not generally affect the operation of the producer. The presence of a fair amount of volatile matter, however, is necessary if a luminous gas is to be produced.

(6) Calorific Value. The calorific value should be determined, as it

has its effect on economics and is required for determining the efficiency of producer practice.

(7) Size. The size of the fuel employed is of great importance. If too small it will tend to clog the producer, interfering with the free passage of the blast. If too large channelling is prone.

Wood[4] recommends ½ to 1½ in. nuts, although this size may be exceeded without undue trouble. Regular sizing is important. The percentage of fines should be watched. Soft coals, which show a tendency to break down into fines, are undesirable. Frequently stock coal, which has weathered, resulting in high proportions of fines, gives difficulties.

Classification of Producers

Producers may be divided into three classes:

(1) Static, which are hand fed, and the ashes are removed manually.

(2) Semi-mechanical, which have either mechanical feed, agitation, levelling or ash removal.

(3) Mechanical, in which the feeding, agitation, levelling and ash removal are all performed mechanically.

In general, static producers require a higher grade fuel for successful operation. Many claims for the use of low grade fuels have been made by the manufacturers of some mechanical producers, but the satisfactory operation of such machines demands a reasonable fuel. Fuels with an ash content over 15 per cent. can only be worked with good results on mechanical producers.

The labour requirements are much less on mechanical producers, but some of the economics are absorbed by the maintenance attendant on a mechanical device.

The more regular feeding, agitation and ash disposal of a mechanical producer tends to produce a more regular quality of gas than can be obtained from a static producer. However careful the control on a static plant, the human element prevents the same regularity.

Whatever type of producer is employed, ash discharge must be controlled, if loss of combustible matter is to be avoided. Regular analysis of the ash discharged should be available.

Normally the *rate of gasification* is higher on a mechanical rather than a static producer. Typical rates of gasification, as given by the Ministry of Fuel and Power,[6] are shown in Table XI.

TABLE XI

Rates of Gasification

lb./sq. ft./hearth Area per hour	Cwt. per hour (to nearest 0·5 cwt.)		
	8 ft. diameter	9 ft. diameter	10 ft. diameter
60	27·0	34·0	42·0
50	22·5	28·5	35·0
40	18·0	22·5	28·0
30	13·5	17·0	21·0
20	9·0	11·0	14·0
10	4·5	6·0	7·0

The rate of gasification is affected by the following factors, which are dealt with in detail in the Bulletin,[6] referred to above:

(1) Class of fuel used. In a mechanical producer a feebly-caking coal may be gasified at the rate of 50 lb./sq. ft. of hearth area per hour, whereas with a strongly-caking coal this rate will drop to about 30 lb. per sq. ft. of hearth area per hour. Even this rate will only be maintained satisfactorily under careful supervision.

(2) The size of the coal employed. Small coal will cause a drop in the rate of gasification.

(3) Type of producer. The design of the producer, especially the blower, will greatly affect this rate.

Static Producers

Static producers are still employed at many works with very satisfactory results. Their successful operation, however, demands careful supervision. Regular feeding, poking and cleaning is essential if regular quality and quantity of gas is to be supplied. Irregular feeding results in wide variation in gas composition. Insufficient poking tends to favour the formation of channels and allows clinker to accumulate, with corresponding deterioration in gas quality.

The typical construction of a static producer, which is self-explanatory, is shown in Fig. 45. It normally consists of a steel shell, built up of 1-in. steel plates, lined with firebricks. The structure is usually set in a concrete water bosh. It will be noted that inspection and poke holes are provided in the side of the shell to facilitate cleaning in the event of the fire becoming badly clinkered.

Normally the poking of the fire is accomplished by rods inserted through poke holes in the crown of the producer. These holes are also used for the test rods. They are closed by specially designed covers or balls, as shown in Fig. 46.

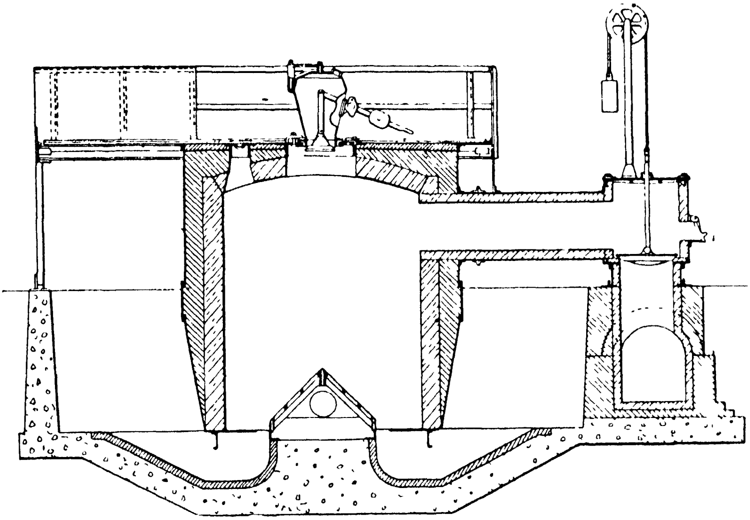
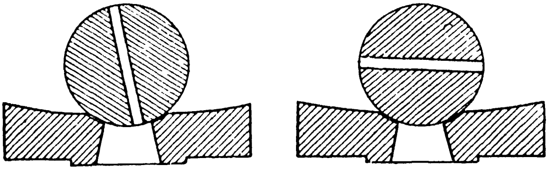


FIG. 45 Static gas producer

(From "Practical Steelmaking", by W. Lister, by courtesy of the author.)

The cleaning of the fire and the removal of ash requires very careful attention. There is always a tendency to remove too much ash at one time, thereby disturbing the depth of the ash bed. This, and the uneven removal of ashes, affects the distribution of the blast.



Ball in Position for Poking. Hole Closed after Poking.

FIG. 46. Poker-hole covers

(From "Practical Steelmaking", by W. Lister, by courtesy of the author.)

Semi-Mechanical Producers

At many plants, where static producers have been in use for many years, various mechanical devices have been incorporated in order to lower labour costs and assure a more regular supply of gas. Frequently these devices take the form of mechanical pokers or levellers, and/or mechanical ash dischargers. Probably the most favoured device is the mechanical feed.

The principles of all these devices are similar to those discussed in detail in connection with mechanical producers.

Mechanical Producers

In order to:

(1) Reduce labour costs.
(2) Obtain a higher rate of gasification.
(3) Employ a lower grade fuel.
(4) Obtain a more uniform quality of gas.

Various mechanical producers have been introduced. Many of these machines have been highly successful in achieving these objectives, but too much stress must not be laid on the use of lower-grade fuels. Even for mechanical producers, the careful selection of fuel is necessary.

The more important types of mechanical producers, together with their chief features and merits, may be summarized as follows:

The Wood Machine, the general construction of which is shown in Figs. 47 and 48, has given excellent results at many steel plants. In this producer, the top plate is fixed to the building and platform and it supports the coal feeder, gas outlet and the two water-cooled pokers. The shell of the producer revolves, normally once every half-hour, in a concrete water-sealed ash-bosh.

One of the chief features of the machine is that the agitating device consists of two water-cooled pokers. As shown in Fig. 48, these pokers are curved at the end and are cooled by water passing down a copper tube to the tip of the poker and returning to the top around the outside of that tube. In the case of failure, through leakage and so forth, these pokers can be quickly and easily replaced. One poker agitates the centre of the fire, whilst the other operates at the side. They rotate on their own axis at such a speed that they describe the path indicated in Fig. 49 during each revolution of the machine.

In the Wood machine the blast is admitted through a stepped blast hood (Fig. 47) which. it is claimed, ensures an even distribution of the blast over the whole fuel bed. The blast is supplied by a turbo-blower, which is largely replacing the old steam jet blower on producers (Fig. 50) especially when high rates of gasification are required.

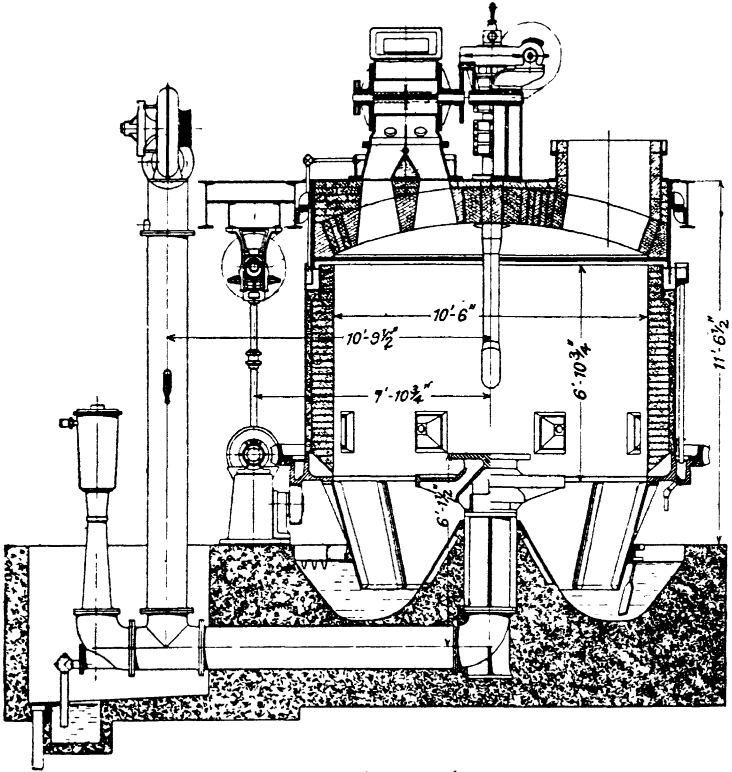


FIG 47. Wood gas machine.

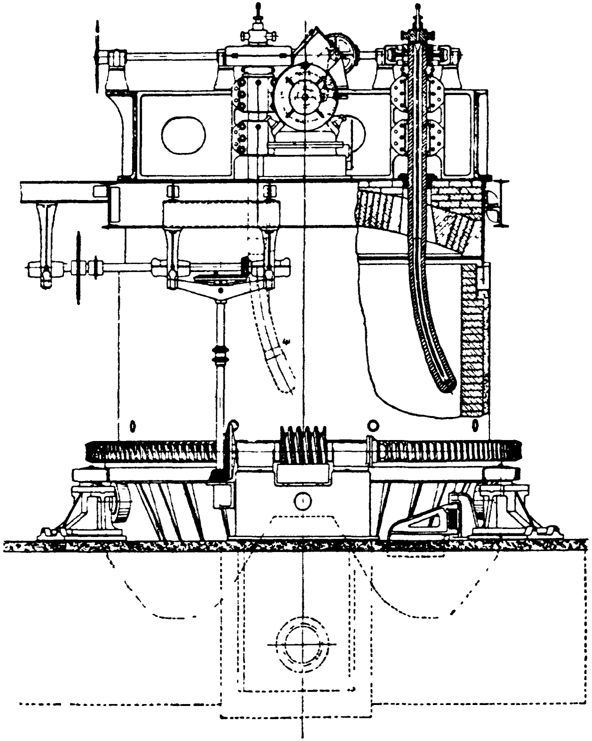


FIG. 48. Wood gas machine.

(From "Practical Steelmaking", by W. Lister, by courtesy of the author

The turbo-blower is simply a small steam turbine, which operates the fan, forcing the air into the producer. Frequently the exhaust steam from the turbo-blower is sufficient to "saturate" the air blast, but when higher saturation temperatures are required, addition "live" steam can be added to the blast prior to its admission to the producer.

The automatic ash discharge, as described by Lester,[2] consists of two ash ploughs, two scrapers and one sweep. The ploughs force the ashes from inside the ash pan to a position between the bosh and the outer edge of the pan. The scrapers, which are attached to the outside of the bosh, carry away any surplus ash to a point where it is automatically discharged by the sweeper into a pit or wagon. The ploughs are removable whereby the rate at which ashes are discharged can be regulated.

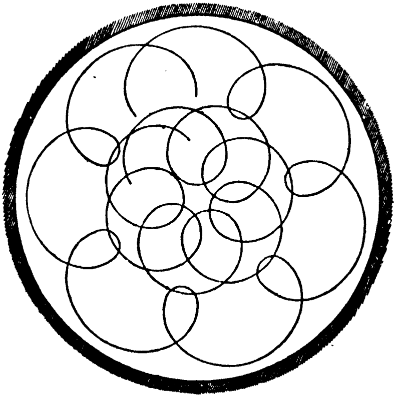


FIG. 49. Wood gas machine poker path.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

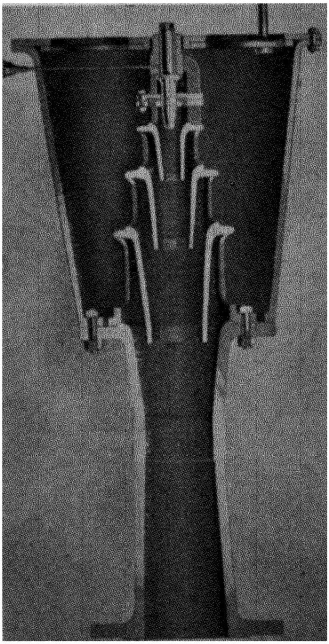


FIG. 50. Steam-jet blower.
(From " Practical Steelmaking", by W. Lister, by courtesy of the author.)

The Morgan Gas Producer, shown in Fig. 51, consists of a fixed top, as in the case of the Wood machine, but the shell and ash pan both revolve together. The ashes are removed by a spiral bar which gives uniform settling of the ash over the entire bed. When it is desired to remove ash, a lever is set, whereby the plough is held stationary during one complete revolution of the ash pan, and is then released. This causes the ashes to be discharged at a given point into a car or conveyor.

In the Morgan Producer, pokers are replaced by a water-cooled leveller which travels through the fire bed at approximately reaction zone level. This has two main virtues, namely, any voids or channels tend to be filled, and the efficiency of combustion is increased as the whole bed is gently disturbed resulting in the film of ash being rubbed off each nodule of coke thereby presenting a new surface for further combustion. The Morgan Producer is made in two sizes, 10 feet inside the brickwork with one leveller and 11 feet inside the brickwork with two levellers.

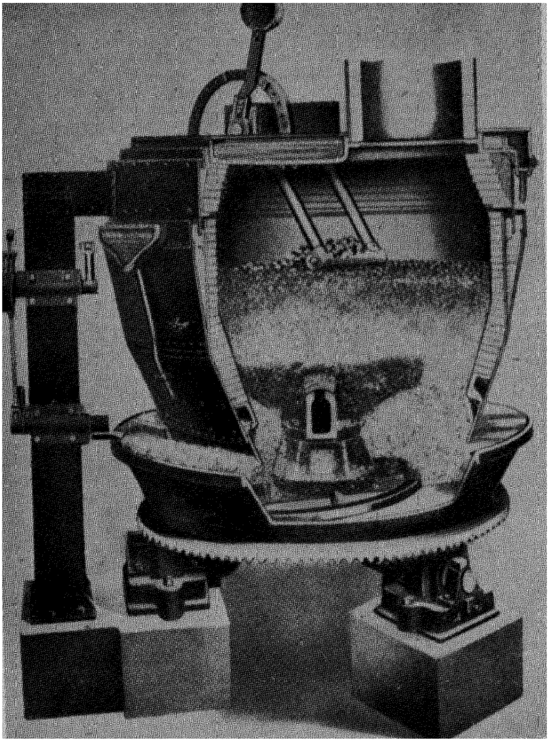


FIG. 51. Morgan gas machine.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

Whether levelling or poking is the better form of agitation appears to depend on the type of coal employed. As pointed out by Bailey,[7] when non-clinkering, free-burning coals are used, a leveller type machine gives excellent results, but when clinkering and coking coals are employed, poker-type machines are advocated.

Another feature of this type of producer is the construction of the lining, which is reduced in thickness towards the top, and water-cooled. This reduces the tendency for the formation of clinker. The water-cooling keeps the inner lining sufficiently cool to prevent the formation of clinker.

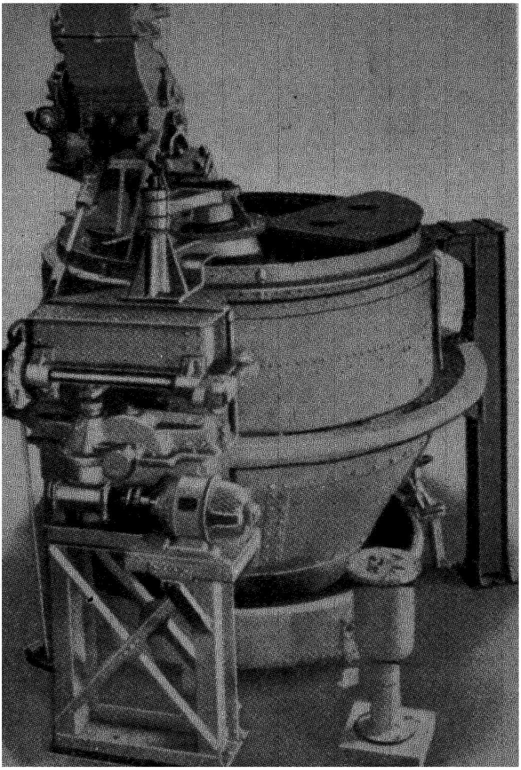


FIG. 52. Wellman gas machine.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

The Wellman Gas Producer, shown in Fig. 52, is built in three sizes: 8 ft., 10 ft., and 10 ft. 6 in. diameter inside the brick lining. Their normal rate of gasification is 32, 50 and 53 tons of coal per 24 hrs. respectively, but this rate is influenced by the type of fuel employed.

As with the Morgan machine, the brick-lined roof or dome, normally found in producers, is replaced by a water-cooled trough or pan. Providing adequate water for cooling purposes is available, this design avoids the serious delays which accompany any failure of a brick-lined roof.

The body or shell of the producer revolves in a clockwise direction. The entire producer is driven by one motor, which in the case of an 8-ft. diameter machine, is 3 h.p., and 5 h.p. for larger types. The shell of the producer revolves once every five minutes, and carries the fuel bed with it.

The blast hood, which is hexagonal, and the cast steel ash pan are fixed together and are independent of the shell. The ash pan is supported on large bearings and rotates with the shell through friction between the ashes in the shell and the pan. By operation of a hand lever, the rotation of this ash pan can be arrested when desired.

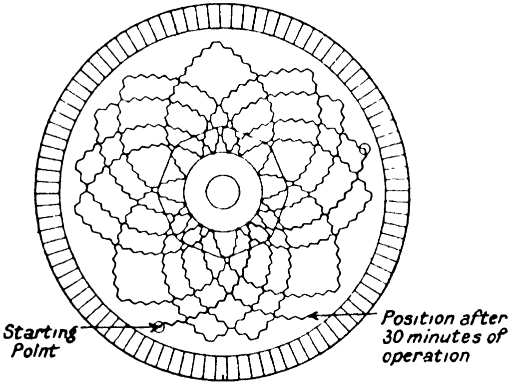


FIG. 53. Path of Wellman poker.
(*From " Practical Steelmaking ", by W. Lister, by courtesy of the author.*)

The effect of stopping the ash pan whilst the shell continues to rotate, is to loosen the ashes, thus rendering them more porous for the free and distributed ascent of the blast to the combustion zone. This stoppage, assisted by the hexagonal blast hood, also breaks up any clinker, and renders this type of machine excellent for gasifying coal of a clinkering nature.

The ashes are moved as they are formed, by a plough, which is lowered into the pan.

One important feature of the machine is the oscillating poker, which is inclined at an angle to the vertical, and describes the path indicated in Fig. 53. As it swings in its travel, upwards and downwards, from the centre towards the lining of the shell and backwards, it keeps the fuel bed loose and free, closes up incipient holes, maintains the fuel bed at a uniform density, prevents clinker forming on the walls of the producer and forces incipient clinker into the ash pan.

A producer of this type will require from 5 to 7 gallons of water per minute for cooling purposes.

The Chapman Producer, shown in Fig. 54, consists of a stationary shell and a revolving ash pan. The floating agitator, which is water-cooled, and the coal feeder are operated by one motor, whilst another motor revolves the ash pan to which the "ashing beam" is attached.

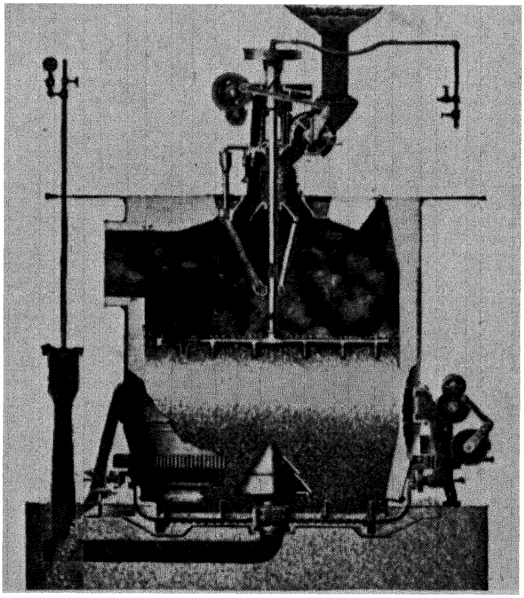


FIG. 54. Chapman gas machine.
(By courtesy of the Staffordshire Iron and Steel Institute.)

The Chapman agitator and coal feed can be fitted to any ordinary producer at a relatively low cost, thereby forming a medium between the hand-fired, hand-agitated producer, and the heavy type mechanical gas machines.

Control of Producers

Irrespective of the type of producer employed, a panel of control instruments is not only desirable, but essential to their efficient and economical operation. These instruments should be used intelligently, alongside the test-rods and gas analyses already mentioned. Normally such a panel will contain:

(1) Steam pressure gauge, which indicates the pressure at which the producer is being blown.

(2) Saturation temperature, which indicates the steam/air ratio.

(3) Top temperature recorder, which records the temperature of the gas leaving the producer. This temperature will assist in regulating the feed and in denoting any tendency for the cracking of the hydrocarbons, which impart luminosity to the gas.

(4) Blast pressure gauge, which is normally a U-tube giving a reading in inches water gauge. Abnormally high pressures would indicate the clogging of the fire due to clinker, mush or some other cause.

(5) Gas pressure gauge, which again is usually a U-tube giving a reading in inches.

At most plants the gas pressure is maintained at a constant figure, by means of a "Gas Pressure Regulator", such as the "Arca" (Figs. 55 and 56), "Chowning" or "Sinchroflow" (Fig. 57). These regulators, although they appear very complicated, give little trouble and are very reliable in practice. When a regulator is employed, the steam valve to each producer is left full open, and the regulator controls the steam pressure to all producers in the battery. As the pressure of gas in the main drops, so the steam pressure is increased, resulting in an increased quantity of gas being produced. An increase in the gas pressure in the main, automatically reduces the steam pressure at which the producers are blown.

The hydrogen content of the gas produced should be kept as low as possible. A gas rich in hydrogen has a tendency to propagate a short hot flame which is very detrimental to the brickwork. A high hydrogen frequently indicates undecomposed water vapour being carried forward in the gas.

This hydrogen results not only from the saturation temperature, that is, the air/steam ratio which can be controlled, but also from the variable moisture content of the fuel employed, and water leakage from the various cooling devices. A quick method of determining the water vapour content of the gas, is to ascertain the "Dew point". This may be done by allowing a small stream of gas to impinge on the cold surface of a beaker of water, when, at the dew point, the vapour will be deposited as beads of dew or moisture.

The test is commenced by using a beaker of warm water, in which a thermometer is inserted. The water is cooled in stages of about 1 deg. by the addition of cold water until dew is deposited. From this reading the amount of water vapour in the sample can be calculated, if the saturation pressure of water vapour at that temperature, and the barometric pressure for the day, is known, by means of the equation:

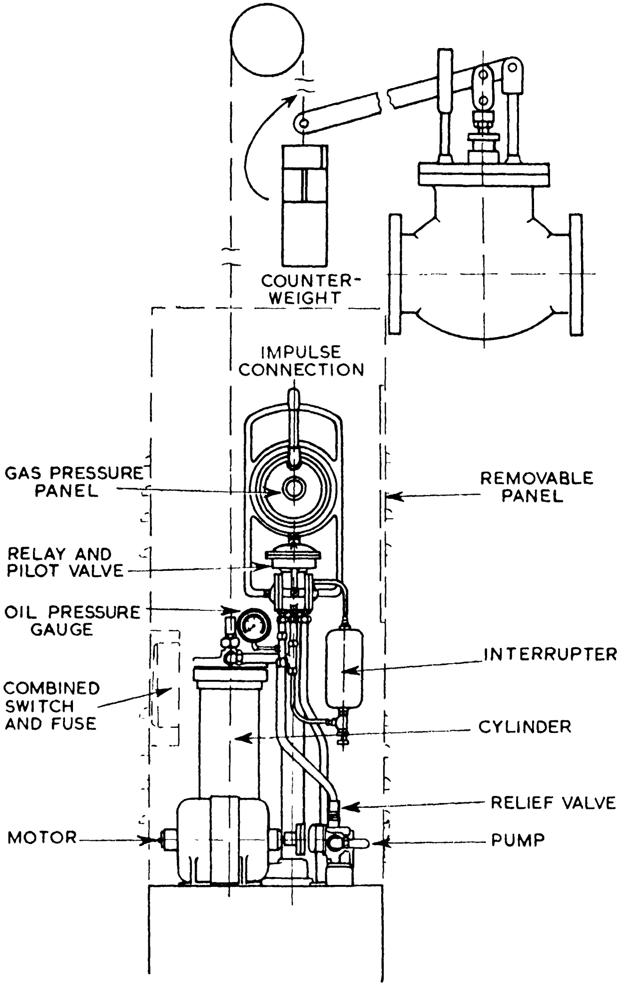


FIG. 55. Arka gas regulator.
(By courtesy of the International Construction Co., Ltd.)

Percentage of water vapour equals:

$$100 - 100\,\frac{\text{Barometric pressure} - \text{saturation pressure of water vapour}}{\text{Barometric pressure}}$$

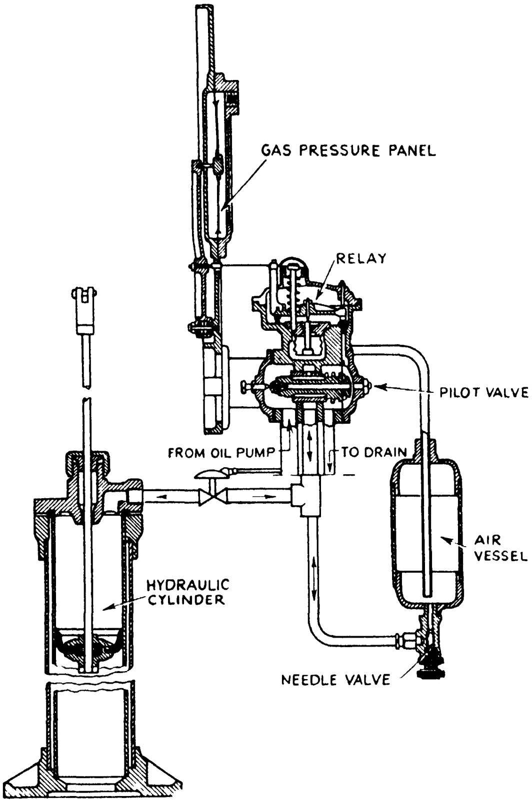


Fig. 56. Arka gas regulator.
(By courtesy of the International Construction Co., Ltd.)

In the example quoted by Wood,[4] the dew point was 35 deg. C., which is equivalent to a saturation pressure of 42 mm., whilst the barometric pressure for the day was 750 mm. Applying this to the equation, we have:

$$\begin{aligned}\text{Per cent. of water vapour in the gas} &= 100 - 100\,\frac{750 - 708}{750}\\ &= 100 - 94{\cdot}4\\ &= 5{\cdot}6 \text{ per cent.}\end{aligned}$$

Analysis of Producer-Gas

The analysis of producer-gas varies with the class of fuel employed, the type of producer used and the operating conditions. The operation of the producer should be so regulated to obtain a maximum carbon monoxide and a minimum carbon dioxide content.

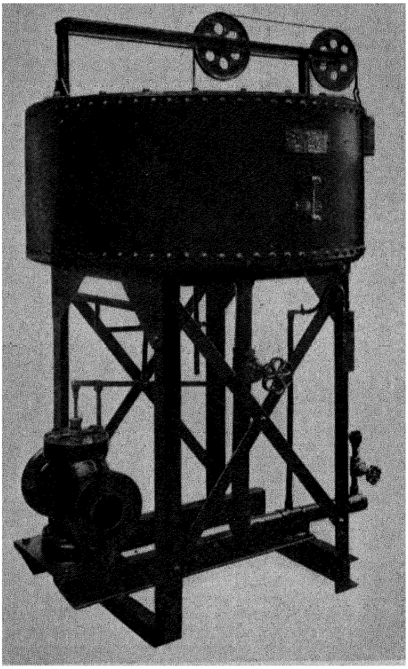


FIG. 57. Sinchroflow gas regulator.
(By courtesy of Messrs. Incandescent Heat Ltd.)

A typical analysis would approximate:

CO	24 to 26 per cent.	
CH_4	3 to 4 per cent.	
H	10 to 15 per cent.	Total combustibles
CO_2	4 to 6 per cent.	=37 to 45 per cent.
N_2	Approx 50 per cent.	

The calorific value will vary from 150 to 170 B.Th.Us. per cubic foot. One cubic foot of producer-gas, at 0 deg. C. and 760 mm. pressure will weigh approximately 0·072 lb. or alternatively 1 lb. of gas will occupy a volume of approximately 14 cu. ft.

The theoretical air required for the combustion of 1 cu. ft. of producer-gas is 1·20 cu. ft., but in practice a slight excess, bringing this requirement up to 1·25 cu. ft., is found necessary.

Layout of Producer-Plants

The actual layout of a producer-plant is governed by local conditions. In some works it is possible to empty the coal direct from bottom-door wagons, which are run up an incline, into the producer-hoppers, as shown in Fig. 58. At other plants, space does not permit this method, and the coal may be emptied from wagons by a "wagon tipper" into a sunken bunker, from where it is transported by grab crane or elevator to the producer-hoppers, as shown in Fig. 59.

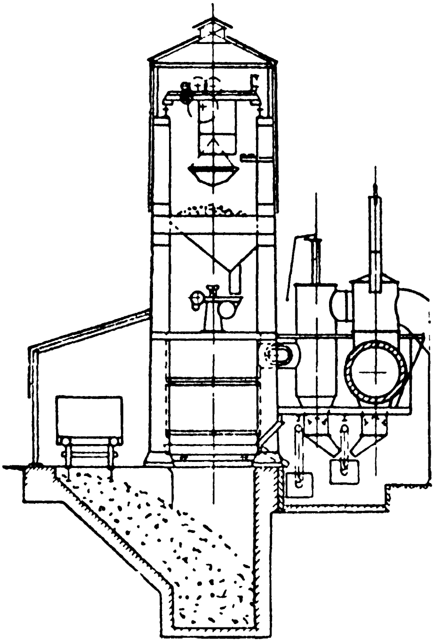


FIG. 58. Producer plant layout.

When designing a gas plant, considerable thought must be given to dust catchers, and to the size of the gas mains, which should have ample facilities for cleaning and burning out. It is desirable that a dust catcher is provided between each producer and the gas main. At the bottom of the dust catcher is an easily operated bell, which should be opened at frequent intervals for the removal of dust and tar.

A typical design and arrangement of dust catchers is shown in Fig. 60. This illustration also shows a water-cooled cut-off valve, which it is advisable to incorporate on each producer. This enables a producer, in the event of a breakdown or failure, to be quickly isolated from the main.

The gas mains should be of ample size to carry the quantity of gas necessary without involving excessive velocities and corresponding high pressures. Many plant engineers recommend 9 sq. ft. of gas main area per ton of fuel gasified per hour. In addition, an allowance of 20 per cent. to 25 per cent. must be made to compensate for tar and soot deposition. This figure, however, is purely empirical and must be amended to meet local conditions. Sudden changes in cross-sectional area or unnecessary changes in the direction of flow should be avoided. Changes in area or direction tend to increase the deposition of tar and soot.

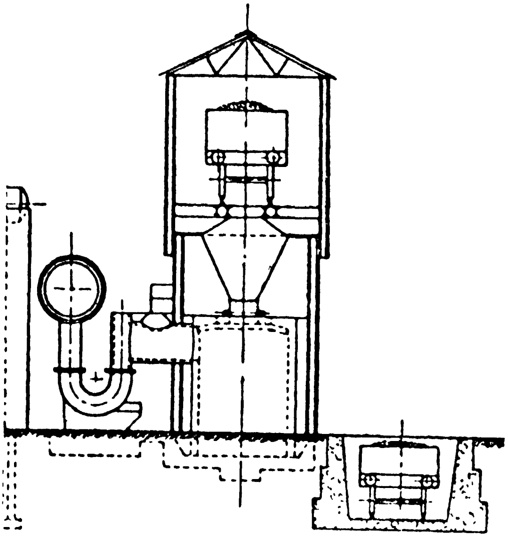


FIG. 59. Producer plant layout.

The insulation of gas mains results in considerable reduction of heat losses and lessens the tendency for tar deposition. A 4½ or 5 in. insulating course between the steel plates forming the main, and a 4½ in. dense firebrick lining, is highly recommended.

Facilities for burning out the gas mains at week-ends, or frequent

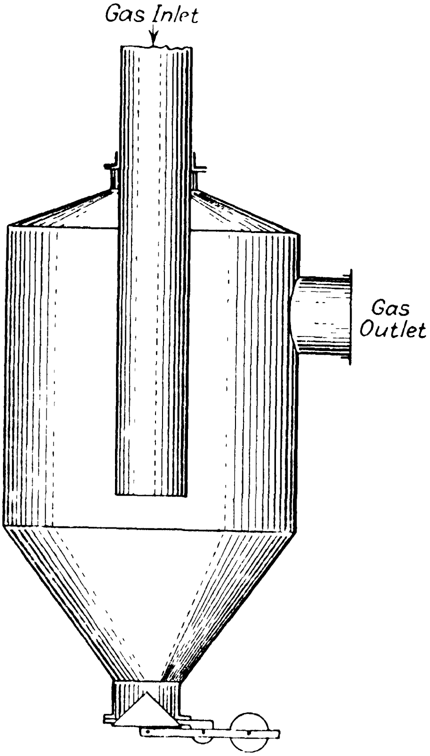


FIG. 60. Dustcatcher.

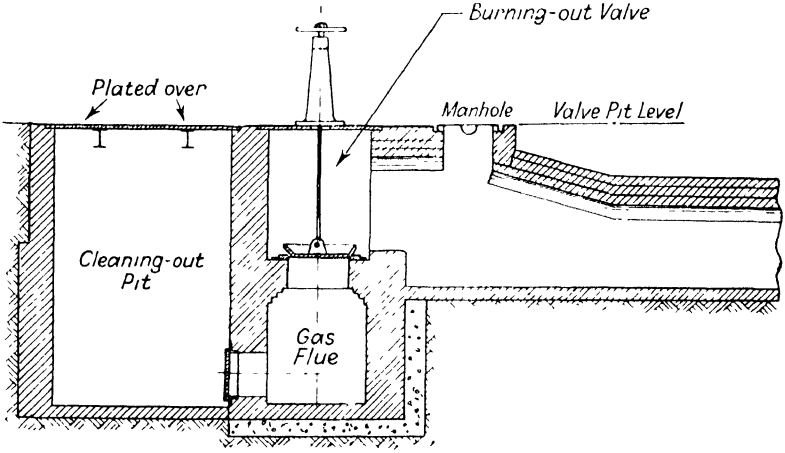


FIG. 61. Section through burning out valve and flue.

intervals, should be provided. Since the deposits are chiefly of a carbonaceous nature, a plentiful air supply and a good draught are essential to satisfactory burning out. When possible, the chimney of the furnace should be used to create the necessary draught, by means of a burning-out valve being inserted between the gas main and the stack. Typical designs of burning-out valves are given in Fig. 61. The actual procedure for the burning out of gas mains will depend on local conditions and can only be settled by careful study of the particular main. Some general principles, however, can be laid down. Having established a satisfactory draught at the furnace end of the main, air should be admitted at a point as near as possible to the producer cut-off valves. Normally the main should be sufficiently hot at this point to ignite the soot and carbonaceous matter. As the burning of the deposits proceeds, further doors or bells for the admission of air can be progressively opened, so that the whole main is gradually cleared.

The burning out can be accelerated by the use of compressed air or steam. When using compressed air, care should be taken to avoid overheating the main. Where changes of direction or cross-sectional area take place, large deposits are likely to accumulate, and agitation may be necessary to dislodge them.

Natural Gas

In America there are several places at which a rich natural gas is available. This gas is conveyed from the gas wells in pipes and admitted to the furnace, without regeneration, through pipes inserted into the ports. When natural gas is used, both regenerators are used for preheating the air.

The approximate composition of natural gas, as quoted by Boylston,[8] is:

	Per cent.	
CO	1·85	Total combustibles
CH_4	75·35	= 92·95 per cent.
C_xH_y	15·75	
CO_2	0·45	Inert Gases
N_2	6·60	= 7·05 per cent.

This gas has a calorific value of 1,100 B.Th.Us. per cu. ft.

The supply of natural gas, however, is diminishing, and steelworks have found it necessary to provide an alternative fuel in case of mergency.

Pulverized Coal

Pulverized coal has had a limited application to open-hearth furnaces in America, chiefly as an alternative or emergency fuel to natural gas.

Herndon[9] describes the use of pulverized coal at a plant consisting of four 50-ton and two 80-ton stationary open-hearth furnaces, all of which had basic bottoms and were of standard design. The only major modification in furnace design, was the opening out of the checkerwork in the regenerators, owing to the tendency for this to become choked. The slag pockets were also provided with removable boxes, which filled up in about 15 days.

The average analysis of the coal, which was ground to pass 80 per cent. through a 200-mesh screen, was: Volatile 36 per cent., fixed carbon 52 per cent., moisture under 1·25 per cent., ash 6 per cent. to 8 per cent. and sulphur under 1·25 per cent. Hermanns[10] suggests that the coal should have a high volatile content, low sulphur, and ash not exceeding 8 per cent.

According to Hermanns,[10] the success of pulverized coal firing depends on the state of pulverization, which should be carried far enough for 85 per cent. to pass through a 300-mesh screen if ideal combustion is to be obtained. If sulphur absorption by the bath is to be prevented, the flame must be so regulated that the sulphur is completely burnt before the flame reaches the bath.

The application of pulverized coal as a principal fuel for open-hearth furnaces is probably very limited, but the application of this fuel to metal melting furnaces, together with the design of the pulverization plant valves, burners and so forth, has been discussed in detail by Cookson.[11]

Mixed Gas Firing

At most composite iron and steel works considerable quantities of blast-furnace and coke-oven gas are available, and can be used for firing open-hearth furnaces. An excellent account of the efficient use of the various by-product gases arising in a composite works has been given recently by Turner and Gray.[12] Fig. 62, taken from their paper, gives an admirable summary of the fuel problems and distribution of a large works.

Typical analyses of blast-furnace and coke-oven gases, and mixtures of these gases together with their calorific value, are given in Table XII:

TABLE XII

ANALYSES OF BLAST-FURNACE, COKE-OVEN AND MIXED GASES[17]

	CO_2	O_2	C_nH_m	CO	CH_4	H_2	N_2	Calorific Value B.T.U/. Cu. ft.
Blast-Furnace	11·80	—	—	26·00	0·80	1·5	59·90	94
Coke-Oven . .	2·60	0·40	3·00	7·00	28·00	56·0	2·80	510
BF/CO = 2/1	8·33	0·13	1·00	20·33	8·66	20·0	41·53	240
BF/CO = 3/1	9·00	0·10	0·75	22·00	6·50	15·5	46·15	205

Both blast-furnace and normally produced coke-oven gas burn with a non-luminous flame, and under certain conditions this lack of luminosity affects the rate of heat transfer.

The problem of *heat transfer* has been discussed in several excellent papers, especially by Evan,[13] Pearson,[14] Putman,[15] Witting[16] and Wright and Pearson.[17] These works show that heat is transferred from the flame to the charge by convection and radiation.

During the melting down stage it is probable that convection is the more important factor. This transfer takes place by the direct contact of the flame and the solid materials. At later stages of the charge, when a slag covering exists, radiation is more important. It is, therefore, obvious that when using blast-furnace or coke-oven gas, or mixtures of these two gases, which produce a non-luminous flame, little disadvantage is experienced during the melting down. In the later stages of the charge, however, artificial luminosity is advantageous.

Pearson[14] points out that heat transfer by convection, which is actually direct conduction of heat from the hot gases to the solid (or liquid) surface by contact, is a function of velocity, since the thickness of the stagnant or slow-moving gaseous film adhering to the surface diminishes as the velocity of the main stream of hot gases increases. It is, therefore, important that port design is carefully considered in order to maintain sufficient velocity in the flame, thereby attaining maximum heat transfer.

As demonstrated by Evans,[13] heat transfer by radiation is a function of two factors:

(1) The temperature differential between the flame and the materials to be heated.

(2) The emissivity of the flame.

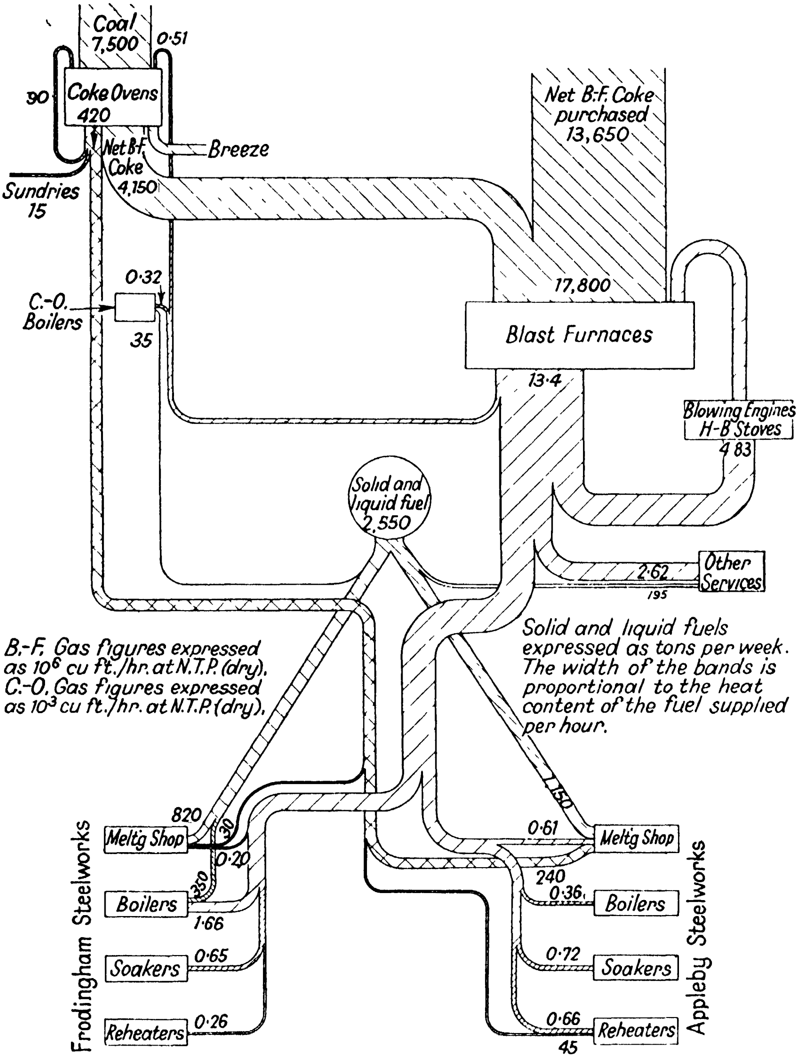


FIG. 62. Gas distribution of a large iron and steel works.
(*By courtesy of the Iron and Steel Institute*)

When solids are raised above the temperature of their surroundings, they radiate heat on all wave lengths. Gases, however, do not behave in the same manner. As shown by Wright and Pearson,[17] if the intensity of the radiation of burning gases is plotted against wave length, it is found that radiation is only emitted between certain wave lengths. These bands are of appreciable dimensions only with gases having three or more atoms in the molecule.

At high temperatures gases such as nitrogen, air, carbon monoxide or hydrogen have little ability to radiate heat. The combustion of blast-furnace gas, which consists chiefly of hydrogen and carbon monoxide, has therefore to be carried to completion before any appreciable benefit is obtained from radiation.

Luminous flames, however, give a greater emissivity. It has been shown that the luminosity of such flames are due to the presence of minute carbon particles. Wesemann[18] advances the theory that the luminosity varies with the fineness of the subdivision of the precipitated carbon. The presence of these carbon particles approximate "black body" conditions, whereby radiation proceeds on all wavelengths and such flames have a high emissivity. Under these conditions the heat transferred in B.Th.Us. per hour will be proportional to $(T_F)^4 - (T_s)^4$ where:

T_F = absolute temperature of the flame envelope.
T_s = absolute temperature of the solid (or liquid) surface.

The luminosity of producer-gas is due to the presence of fine carbon particles carried over in the gas. In the same way, coke-oven gas can be rendered luminous by decomposition of the hydrocarbons (methane, etc.). When coke-oven gas is pre-heated to 900 deg. C. or over, CH_4 is decomposed, resulting in the formation of carbon particles, which impart luminosity to the flame. The breakdown of ethane, naphthalene and benzol by pre-heating, all contribute to this luminosity.

This increased luminosity, however, following the cracking of the hydrocarbons, is accompanied by a decrease in the calorific value of the gas. In view of this fact many authorities recommend that coke-oven gas should be used without regeneration, and that luminosity should be imparted to the flame by an artificial luminant, such as tar or oil. This enables the total calorific value of the gas to be utilized and satisfactory heat transfer by radiation to be obtained. It also reduces thermal and other losses of coke-oven gas in the valves, brickwork and so forth.

The luminosity of a flame due to the presence of carbon particles can be destroyed if water vapour is present owing to the reaction

$$H_2O + C \rightleftharpoons CO + H_2$$

This reaction is a function of temperature and pressure. An increasing temperature increases the velocity of the reaction in the direction of the upper arrow. In the temperature range of 1,200 to 1,250 degs. C. the velocity from left to right is very rapid.

The reaction also accounts for the loss of "body" or luminosity which occurs so frequently in mixed or producer-gas flames, when leakages from water seals or valves occur.

Luminosity can also be imparted to the mixed gas flame by the addition of producer-gas, that is, the use of a "three gas mixture". This method, however, is not highly recommended, as it usually results in higher fuel costs due to the complication of control gear. As pointed out by Wright and Pearson,[17] when the three-gas mixture is employed, the phrase "the use of a minimum of producer-gas to give adequate luminosity" has no meaning since:

(1) Luminosity cannot be defined visually.

(2) The normal producer-gas mushroom control valve will not regulate to fine limits, and even if it is set, it has to be adjusted as the week proceeds, due to sooting up.

(3) There is no reason to suppose that any additional advantage from the hydrocarbons in the mixed gases results, i.e. the effect is purely additive.

It is, therefore, obvious that the most popular application of mixed gas firing takes the form of employing cold (unregenerated) coke-oven, or mixed gas, with tar or oil as a luminant.

The use of pulverized coal has been tried, especially in Germany, but it has not proved popular. This is due to the ash tending to choke up the regenerators.

The actual method of application depends on local conditions. At some plants the various gases are conveyed in separate mains to the open-hearth plant, and used direct from these mains in the furnaces. The installation of gas holders, however, is very desirable. The employment of gas holders enables a more regular setting of gas mixtures, and assures a uniform pressure of the gases. Normally the "dry" type of holder is employed. In this type of holder the gas is contained under a floating lubricated piston.

Without the use of a gas holder, wide fluctuations in gas pressure are likely to occur. In the blast-furnace gas system, any irregularity in the operation of the furnace will be felt. On the other hand, a holder will not only act as a buffer for pressure changes, due to coke-oven and furnace reversals but will even up the production and consumption of the gases.

When using blast-furnace and coke-oven gas, it is the usual practice to set the volume of blast-furnace gas at a fixed value, and to vary the quantity of coke-oven gas according to the heat input required. This method is advisable because:

(1) A smaller adjustment of the higher calorific value fuel brings about the desired results.

(2) A smaller volume change at the furnace is involved, thereby reducing any effect on the velocity of gas at the ports which would influence heat transfer.

Davies,[19] describing the use of mixed gas on a 200-ton tilting furnace, states that their normal practice is to set the blast-furnace gas at 200,000 cu. ft. per hr. During melting down the coke-oven gas averages 100,000 cu. ft. per hr., and is gradually reduced as the charge proceeds, so that at the finish of the heat it may be as low as 60,000 cu. ft. On the 80-ton fixed furnace during the melting stage, the blast-furnace gas is set at 120,000 cu. ft. per hr., and the coke-oven gas, which is reduced as required, at 60,000 cu. ft. The approximate heat consumption is 5,000,000 B.Th.Us. per ton of ingots produced on a tilting furnace, and is slightly less on the fixed furnaces.

The modification in furnace design, and the type of control and reversing gear used in connection with mixed gas firing, is discussed in the next chapter.

Oil Fuels

As already stated, tar and fuel oil is used frequently as an illuminant in connection with blast-furnace, coke-oven and mixed gas firing.

Describing one method of application, Evans[20] states that the tar is pumped from a main in which it constantly circulates, at a pressure of 15 lb. per sq. in., and at a temperature of 50 degs. C., into a burner, which consists of an ½-in. diameter water-cooled tube. Almost complete atomization, with the production of a luminous flame, is obtained by injecting steam at 100 lb. per sq. in. pressure into the burner.

At a plant discussed by Geary,[21] the tar is stored in a central heated tank, from which it is pumped through a steam-jacketed pipe-line to reservoirs attached to each furnace. It is delivered to the burners at a pressure of 60 to 80 lb. per sq. in. and at a temperature of 180 degs. Atomization is carried out by compressed air which is injected into the barrel of the burner at a similar pressure. The discharge of the tar mixture through a small orifice, approximately 2·5 mm. bore, produces a sufficiently fine degree of atomization.

TABLE XIII

CHARACTERISTICS OF OIL FUEL.

Type of Oil	Sp. Gr. at 60° F.	Viscosity at 100° F. Redwood	Flash Point	Cal. Value BTU/lb.	Sulphur Content	Remarks
Bunker. C.	0·99	6000 6500	150° F. Min.	18,250	2·7%	Principal Oil used in Open Hearth Furnaces in America
Mexican Fuel Oil ..	0·95	1500	150 F. Min	18,750	approx. 3·5%	
Coal Tar Fuel. 100 Creosote Grade B. ..	1·15	100	150° F Min.	16,500	1·0% MAX	Should be completely liquid on being slowly warmed to 100° F. (38° C).
C.T.F. 200	—	1500	150 F. Min.	16,250	1·0% MAX.	
C.T.F. 250	—	500 (at 135° F) T.V. at 30° C. 70–120 sec.	150° F. Min.	16,000	1·0% MAX.	
C.T.F. 300	—	Tar Viscometer (10 mm.) at 00° C. 40 80 sec.	150° F. Min.	16,250	1·0% MAX.	

The burner in this case consists of a 4-in. external diameter tube, and is carried in a 5-in. blast-furnace tuyere, set in the side wall of the combustion chamber. The burner is water-cooled and its tip projects about 6 in. into the furnace. A tar pump is attached to each burner.

During the last twenty-five years the use of oil as a principal fuel has made great progress in America. By 1940, according to Noor,[22] it had, owing to its low cost and the ease of handling, become the principal open-hearth fuel, with natural gas occupying the second place.

The chief advantage of oil fuel is the ease with which it can be transported, handled and applied in practice. It saves the erection of expensive producer plants. It also gives a higher efficiency. When coal is converted into producer-gas, even on good producer practice, approximately 60 per cent. of the total heat units in the coal are delivered to the furnace. When employing oil, about 95 per cent. of the total heat units in the oil are delivered to the furnace.

The fuel generally employed in America is "Bunker C" oil, the characteristics of which are given in Table XIII. It is, however, at times necessary to use other types of oil fuel, and this necessitates some modification in application. As shown by Seil,[23] "Bunker C" oil burns more slowly than the lighter oils. The variation in specific gravity also affects its ignition velocity.

The chief characteristics and properties of fuel oils, which affect their application, may be enumerated as follows:

Physical Tests

(1) Specific gravity, which affects ignition velocity and flame velocity.

(2) Flash point, that is, the temperature at which on being slowly heated, the oil evolves vapour in sufficient quantity that on the application of a flame a momentary flash is produced.

(3) Viscosity, which is the determination of the fluidity of the oil at certain temperatures. It may be measured in several ways, but usually by recording the time taken for a definite quantity of oil to pass through an orifice or short pipe under known temperature conditions.

Chemical Tests

(1) Calorific Value, which is determined normally in an ordinary "bomb" calorimeter.

(2) Sulphur content. At one time the sulphur content was more important than it is in modern practice, owing to the possibility

of sulphur pick-up in metallurgical operations. It has, however, been demonstrated that providing the sulphur is burnt before the flame touches the bath, the amount of sulphur absorbed may be neglected.

(3) Ash content. In certain classes of oil, where the ash content is appreciable, this figure is important, owing to troubles arising from checker deposits, etc.

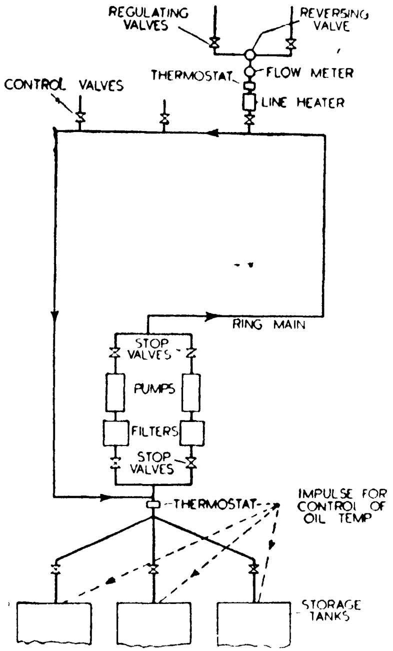


FIG. 63. Fuel oil supply system.
(By courtesy of " Metallurgia ")

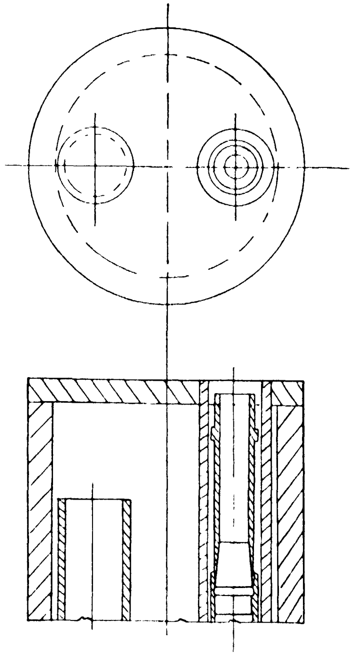


FIG. 64. Conventional oil burner.
(By courtesy of " Iron and Steel ".)

The normal practice is to store the fuel oil in bulk in central tanks, from which it is conveyed to reservoirs at each furnace. The pipe-lines from the storage tanks to the furnace reservoirs may have to be heated, but this depends on the nature of the oil used. The temperature necessary is a function of the viscosity of the oil. Some authorities[24] recommend a kinematic viscosity of 12·5 stokes as the maximum practical conditions for pumping, whilst a viscosity of 3·5 stokes is more suitable for free pump circulation.

The fuel oil must be heated before admission to the burners in order to obtain the correct viscosity to give satisfactory atomization. Some operators[24] recommend that a kinematic viscosity of approximately 0·25 stokes, which is equivalent to a Redwood 1 viscosity of 100 sec., is desirable for atomization. It is, however, a most important feature of fuel oil application that the correct temperature and/or viscosity for atomization is attained.

Atomization may be carried out with compressed air or steam. Steam is generally employed. The chief disadvantage in the use of steam is the water reaction, the effects of which have already been discussed.

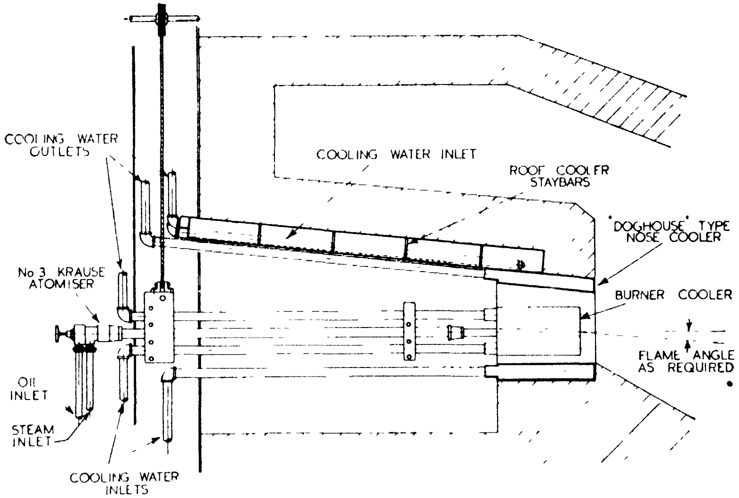


FIG. 65. Blaw-Knox oil burner.
(*By courtesy of "Metallurgia"*)

If the oil is atomized with compressed air at 1·5 to 2 atms. above atmospheric pressure, about 15 per cent. of the total air required for combustion is thereby supplied.

At most American works, where steam is used for atomization, the temperature of the oil and steam are very carefully controlled to suit the particular oil being employed. When using "Bunker C" oil, the oil temperature is normally maintained at 160 degs. F. At the plant described by Hay[25] the steam used for atomization is maintained in the mains at a pressure of 130 lb. per sq. in. and a total temperature of 450 degs. F. When charging scrap this steam is throttled to 110 or 120 lb. pressure, whilst after hot metal a pressure of 90 to 100 lb. is carried. On a $\frac{11}{16}$-in. dia. burner, using 500 gals. of oil per hr., the steam flow is 1,025 lb. per hour. This figure is equivalent to 2·05 lb. of steam per gal. of oil.

Burner design has occupied considerable space in American technical literature, but the conventional burner is generally adopted in preference to the more complicated designs. As pointed out by Fisher,[26] the conventional burner and mixer, shown in Fig. 65, in general use on most open-hearth plants is a simple piece of equipment. When compared with other methods of firing it may be regarded as a crude design. After many experiments, most operators prefer one gun at each end of the furnace rather than a multi-jet system.

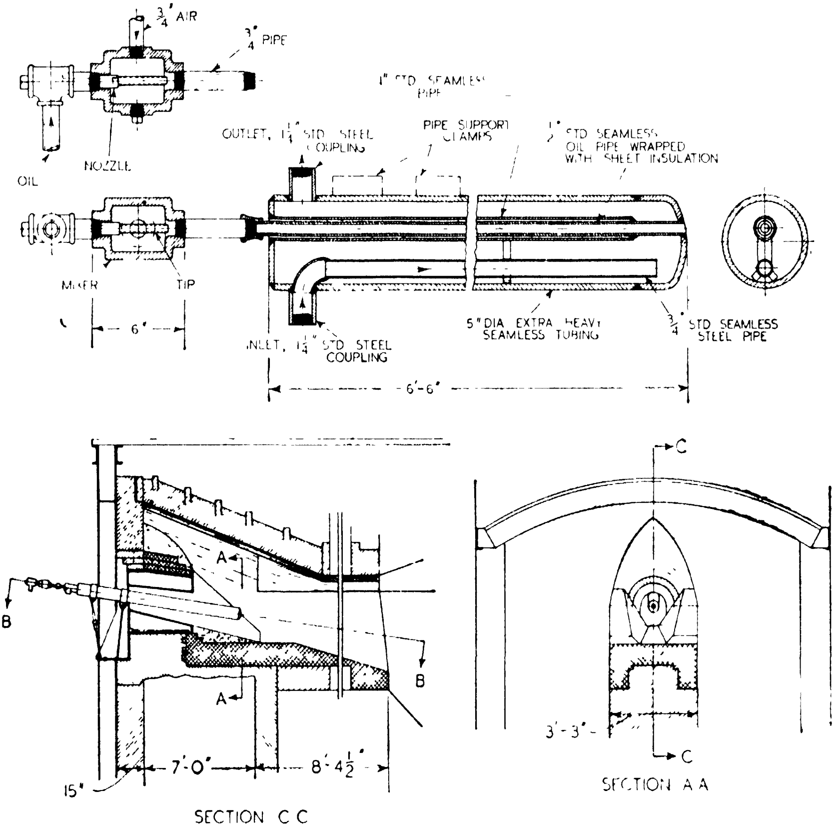


FIG. 66. Loftus burner.
(By courtesy of "Metallurgia")

These burners are designed to burn from 3 to 10 gals. of oil per min. in one gun. On furnaces of 80- to 100-ton capacity, a gun supplying 4½ gals. per min. is considered a fair average, whilst with 200- to 250-ton furnaces this average increases to about 6½ gals. per min. On large furnaces, the oil consumption may be as high as 9 gals. per min. during charging, and this is reduced gradually as the heat progresses. Towards the end of the heat the oil supply may be as low as 5 gals. per min. Some idea of fuel consumptions and rates are given in Table XIV, based on the work of Fisher.[26]

TABLE XIV

OIL FUEL PRACTICE

Group	No. of Furnaces	Tons Tapped	Max. Gals. of Oil used per Hour	Average % of Max Fuel maintained	Gals. of Oil per Ton of Ingots
A	9	201	550	82·5	21·04
B	5	186	500	89·6	22·26
C	7	166	460	90·2	22·73
D	5	162	425	91·3	24·02

Normally high fuel rates mean rapid and complete combustion in the furnace chamber. Low fuel rates frequently result in incomplete combustion in the furnace and unburnt vapours passing into the checkers. This results in checker failures and loss of heat units. If the best results are to be obtained, combustion must be controlled by the use of instruments.

Since heat transfer is affected by the temperature differential between the flame and the bath and by the distance between the heating media and the material to be heated, the control of flame temperature and position is of paramount importance. Discussing flame conditions, Fisher[26] suggests that three critical points in flame position should be observed:

(1) The lower part of the flame should strike the bath opposite or before the centre line of the first door.

(2) The top of the flame should be tangent to the line of sight from the wicket hole of the second door to the skew back line of the furnace.

(3) The flame should be sufficiently transparent at the wicket-hole of the third door for the refractory outlines of the back wall to be discerned.

These conditions are shown diagrammatically in Figs. 67 and 68.

Some open-hearth operators prefer a slightly longer flame than that recommended by Fisher.[26] In fact, the type of flame must vary with

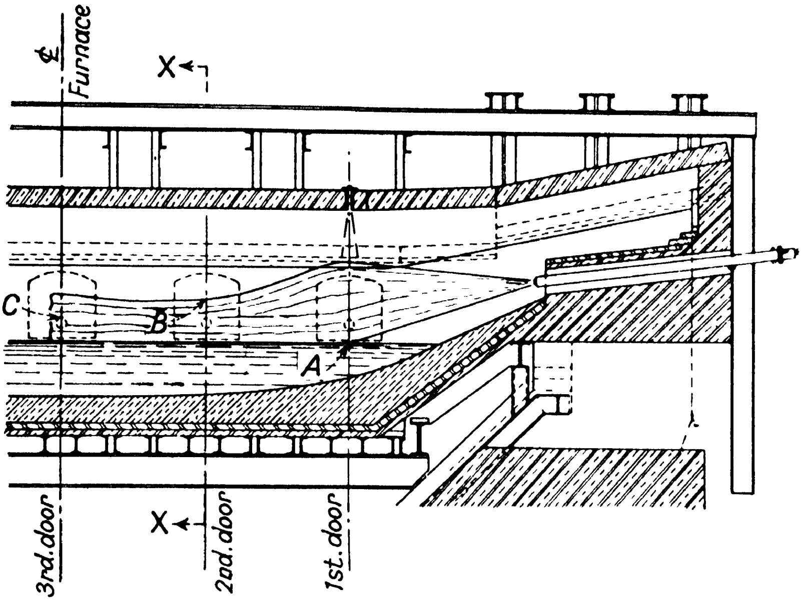


FIG. 67. Flame position in an oil-fired O. H. furnace. (*After Fisher by courtesy of the Am I.M.E*)

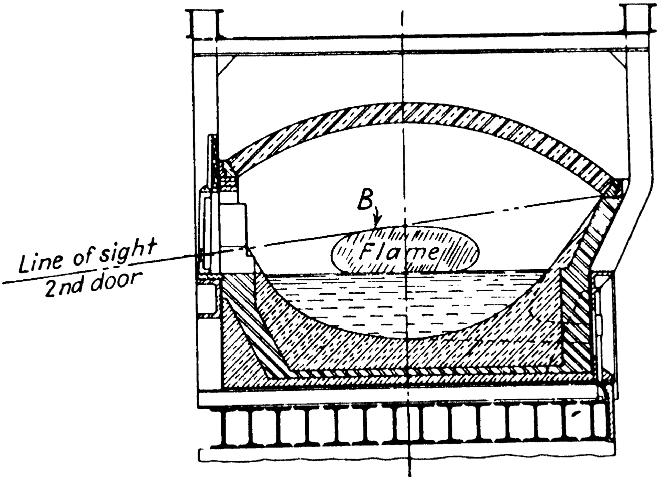


FIG. 68. Flame position in an oil-fired O. H. furnace. *After Fisher by courtesy of the Am.I M.E*)

the stage of the heat. A short hot flame is ideal for melting down, whereas a long luminous flame is more desirable during refining. This change in flame type may be controlled by adjustment of the fuel/air ratio and the atomizing steam pressure. The length of the flame may be decreased by increasing the steam pressure. The position and type of flame is also controlled by the design of the burner and its angle towards the metallic bath.

Colloidal Fuel

The use of colloidal fuel in open-hearth furnaces is discussed by Flagg.[27] This practice consists essentially of the addition of pulverized coal to ordinary oil fuel firing.

It is claimed that the addition of even 5 per cent. of coal to the oil fuel resulted in an immediate effect on the flame produced and some reduction in fuel consumption. Without coal the flame became non-luminous at the third door and the furnace became abnormally hot. With the addition of coal the flame could be seen over the entire length of the furnace and the roof was in general much cooler.

At the time of writing, however, the use of colloidal fuel is only in an experimental stage, and it is impossible to discuss such troubles as checker deposits and so forth, which are likely to be encountered.

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For further information on open-hearth fuels, the following works are also recommended:

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CHAPTER 6

THE OPEN-HEARTH FURNACE

When reviewing the operating data and dimensions of various modern furnaces, it is remarkable that such wide variations in design are capable of giving equally good results. As Hermanns[1] said in 1924, scientific principles for calculating the dimensions of open-hearth furnaces are very indefinite and many values used in arriving at them are very empirical.

If a steelworks is erected to produce a certain annual tonnage, the quantity required is divided amongst a number of furnaces. The number of furnaces decided will very largely control their individual capacity. One factor to be borne in mind is that as a general rule a large unit is more economical than a smaller one. Probably the most economical capacity varies from 150 to 200 tons.

Under certain conditions, however, small furnaces may be advocated. Small furnaces are more suitable for a jobbing trade. At works producing alloy and special steels of widely varying specifications, small units may be desirable.

Typical furnace dimensions are given in Tables XV and XVI representing both American and British practice.

Having fixed the capacity of the furnace, the first logical step is to determine the size of the hearth required to produce that tonnage. Reference to Table XVI shows considerable variation in hearth area per ton, whereas a more uniform value is obtained for American practice in Table XV. This figure, however, must vary according to the metallurgical load. The slag produced, for example, in the acid process occupies less space than that produced in the basic process. Therefore acid furnaces can be made 15 per cent. smaller than basic furnaces of the same capacity.[1]

The hearth area shown in Table XVI is the product of the length and width of the bath inside the dolomite or sand lining. The thickness of the brickwork and dolomite or sand lining must be taken into consideration when designing the hearth pan.

The ratio of the width to the length of the bath varies from 1 : 2 to 1 : 3. The larger value is usually used on larger furnaces. European

TABLE XV

DIMENSIONS OF OPEN-HEARTH FURNACES. AMERICAN PRACTICE (LONGENBECKER[2])

Furnace	Capacity Tons	Hearth			Area of Hearth per ton	Depth of Bath	Volume of Checkers to bottom tile		Volume of Checkers per ton*		Stack		H.P. of Waste Heat Boiler
		Length	Width	Area			Air Cu. Ft.	Gas Cu Ft	Air	Gas	Height	In Dia.	
		ft. in.	ft. in.	sq. ft.	sq. ft.	inches					ft in.	ft. in	
1	180	43 0	16 0	656	3·64	36	4,980	3,220	27·7	17·9	169 4	6 0	600
2	180	40 0	16 0	620	3·44	36	4,980	3,220	27·7	17·9	169 4	6 0	600
3	180	43 0	16 0	656	3·64	36	4,980	3,220	27·7	17·9	169 4	6 0	600
4	180	42 6	16 0	636	3·53	36	4,980	3,220	27·7	17·9	169 4	6 0	600
5	180	43 0	18 0	680	3·77	36	4,980	3,220	27·7	17·9	169 4	6 0	600
6	180	43 0	18 0	680	3·77	36	4,980	3,220	27·7	17·9	169 4	6 0	600
7	250	45 0	18 0	770	3·08	47	6,160	4,325	24·6	17·3	169 4	6 0	600
8	300	47 0	19 0	820	2·73	47	6,918	4,930	23·1	16·4	171 4	7 0	893
9	300	47 0	19 0	820	2·73	47	6,918	4,930	23·1	16·4	171 4	7 0	893
10	300	47 6	19 0	830	2·77	47	6,918	4,930	23·1	16·4	171 4	7 0	893
11	300	47 0	19 0	860	2·87	42	6,918	4,930	23·1	16·4	171 4	7 0	893
12	350	55 0	19 0	992	2·83	42	9,028	4,834	25·8	13·8	171 4	7 0	893

* Calculated by the Author.

TABLE XVI

OPEN-HEARTH FURNACE DIMENSIONS—BRITISH PRACTICE. (OPEN-HEARTH COMMITTEE[3])

Furnace	Capacity Tons	Hearth			Hearth Area per ton/Steel sq. ft.	Depth of Bath inches	Checker Dimensions in Feet						Process
							Height		Length		Width		
		Length ft.	Width ft.	Area ft.			Air	Gas	Air	Gas	Air	Gas	
1	42·6	28·00	11·50	284	6·65	9·25	17·25	17·25	12·00	12·00	8·50	6·00	Acid. Cold Charged Fixed.
4	80·6	37·50	12·50	412	5·11	12·00	22·50	22·50	16·70	16·70	9·25	7·50	Acid. Cold Charged Fixed.
9	61·8	28·50	10·00	242	3·92	15·70	10·50	10·50	20·00	20·00	9·50	6·00	Basic. Cold Charged Fixed.
15	100·0	37·00	14·00	440	4·40	14·00	20·50	20·50	20·50	20·50	12·00	8·00	Basic. Cold Charged Fixed.
20	106·0	39·33	14·00	468	4·41	13·90	21·75	21·75	17·00	17·00	10·75	7·00	Basic. Hot Metal Tilting.
24	121·2	35·00	10·50	335	2·76	23·50	9·75	9·75	22·00	22·00	8·75	6·25	Basic. Hot Metal Tilting.
29	250·0	45·00	13·00	533	2·12	30·50	18·50	18·50	19·50	19·50	11·67	7·33	Basic. Hot Metal Tilting.

TABLE XVII

OPEN-HEARTH FURNACE REGENERATORS

Source	Air Regenerators				Gas Regenerators				Capacity of Furnace	Remarks
	Length	Width	Depth	Volume per ton	Length	Width	Depth	Volume per ton		
	ft. in.	ft. in.	ft. in.	cu. ft.	ft. in.	ft. in.	ft. in.	cu. ft.	tons	
Davies[8] . .	18 8	9 9	16 8½	37·80	18 8	7 4½	16 8½	28·70	80	
Davies[8] . .	19 5	12 2	18 8½	19·76	19 5	9 0	16 8½	14·58	200	
Winter[4] . .	22 0	10 0	10 6	12·60	22 0	8 0	10 6	10·00	150	
Geary, Crooks and Drewery[9]	19 2	8 0	12 9	7·80	19 2	6 0	12 9	5·80	250	
Geary[10] . .	22 0	9 9	13 6	29·00	22 0	8 9	13 6	26·00	100	Producer Gas Fired
Geary[10] . .	22 0	9 9	13 6	29·00	22 0	9 9	13 6	26·00	100	Coke- oven Gas Fired

practice appears to favour 1 : 2·5 to 1 : 2·75. To some extent the length is controlled by the type of fuel employed. A fuel producing a short flame requires a shorter furnace than a fuel burning with a longer flame. The width of the furnace is limited somewhat by the distance at which repairs to the backwall can be performed. On large furnaces, where the width is considerable, this difficulty may be overcome by the use of a fettling machine.

The chief features or parts of an open-hearth furnace are shown diagrammatically in Fig. 22. This diagram is of a fixed type furnace.

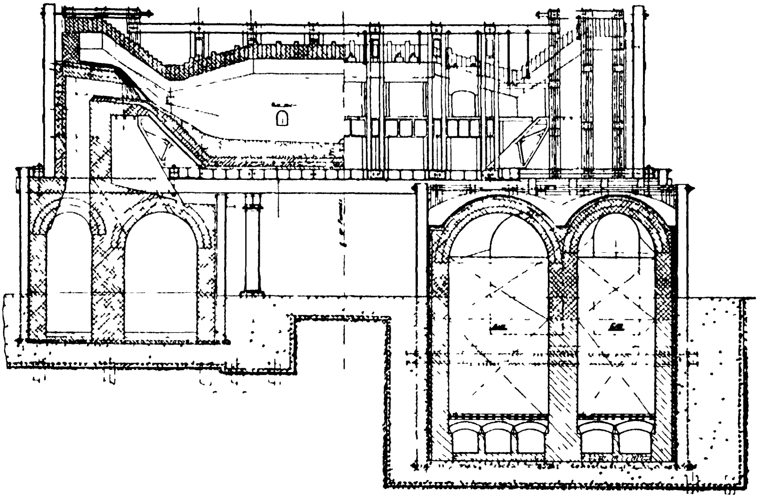


FIG. 69. A 90-ton fixed open-hearth furnace
(*By courtesy of the Iron and Steel Institute*)

Open-hearth furnaces can be divided broadly into two types: fixed and tilting furnaces. The general principles of construction and design are common to both types. The main differences are connected with the foundations, tilting gear and the port and furnace end construction. Minor modifications are also incorporated when special fuels are employed.

The Fixed Furnace

The general design of a 90-ton fixed open-hearth furnace is shown in Fig. 69. This particular furnace has been fully described by Winter.[4]

The *pan*, which forms the hearth of the furnace, is constructed of ½- to 1-in. steel plates riveted together and frequently reinforced by welding. In some cases this pan has a depression of about 5 in. in the vicinity of the tap hole. As pointed out by Reinartz,[5] this depression enables a greater factor of safety at this vulnerable point. The pan rests on a grillage or lattice of heavy joist or I beams, which is supported by massive concrete or masonry foundations. At each end of the pan, chill-boxes or castings of ample size are provided to facilitate the cooling of these parts. These castings allow a current of air to circulate.

In recent years much attention has been paid to strengthening the *superstructure* of all new furnaces. They have been rigidly bound together, heavy sections having replaced the old-fashioned tie-rods. The buckstays are frequently constructed of rolled steel slabs or laminated plates having a cross-section of 6 × 15 in. The strength and rigidity of the superstructure cannot be over-emphasized.

The design and dimensions of the *regenerators* play an important part in the successful operation of an open-hearth furnace. In spite of their importance, considerable difference of opinion regarding their size and design exists. According to Johnson,[6] Consett suggests that the total regenerator volume should be 70 cu. ft. per ton of steel melted, the air chamber being 10 per cent. larger than the gas. Campbell[7] recommends 50 to 100 cu. ft. per ton of steel melting capacity. Reference to Tables XV, XVI and XVII, however, shows wide variation in many cases. It is remarkable that such variable design and dimensions give equally good results at different works.

The efficiency of regenerators, however, is not merely the total volume per ton of capacity, but the effective regenerative or heating surface available. The heat transfer between the brickwork and the gases passing through the chambers is a function of the temperature differential and the time of contact between the brickwork and the gases. The time of contact is dependent on the velocity of the gases and the distance between the inlet and the outlet of the chambers. The effective distance would appear to be the diagonal between the uptake from the ports to the flue leading to the valve.

The effective heating or regenerative surface can be varied considerably within a given total volume, by altering the arrangement of the brickwork, or by the use of special shaped checker bricks. Clements[11] states that on a basis of half-hourly reversals, the only effective portion of the checker brick is that within one inch of the surface. Therefore, theoretically a two-inch brick would be ideal, but a brick of this dimension would be insufficiently strong. The standard 9 × 4½ × 3 or the 9 × 4½ × 2½ brick is preferred by most furnace builders. Special designs of checker bricks have been introduced, as shown in Fig. 70, with the object of increasing the effective regenerative surface, without unduly reducing the strength of the structure.

Loftus[12] shows that when 9 × $4\frac{1}{2}$ × 3 in. bricks are used, in such a way as to give 6 × 6 in. flues or openings, the effective heating surface is 2·82 sq. ft. per cu. ft. of checkerwork. The total weight of brickwork is 42·44 lb. per cu. ft. If 9 × $4\frac{1}{2}$ × $2\frac{1}{2}$ in. bricks are employed, still giving 6 × 6 in. flues, the total weight of checkerwork becomes 47 lb. per cu. ft. and the effective heating surface is increased to 3·49 sq. ft. per cu. ft. The effective heating surface may be increased to 7·44 sq. ft. per cu. ft. if Lermer patent checker bricks are used, but the flues are reduced to 5 × 5 in.

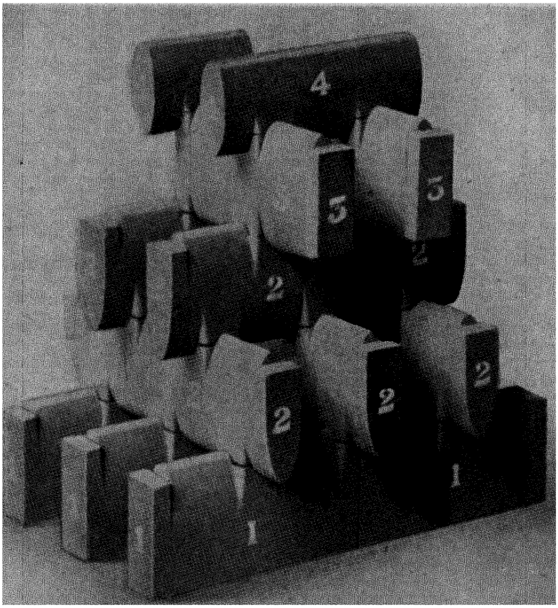


FIG. 70. Special checker-bricks.
By courtesy of the " British Steelmaker ".)

At several plants the effective heating surface has been considerably increased by reducing the size of the flues or openings. Although this reduction in flue size gives increased regenerative capacity of the chambers, the smaller openings are prone to become clogged by dust deposition. These depositions result in loss of draught, especially as the age of the checkers increases, resulting in decreased output per hour.

As pointed out by Robinson[13] the effective regenerative capacity of old-fashioned checkers can be increased in this way, but it is advisable on a new furnace to allow sufficient regenerative capacity with flues of sufficient size to reduce the tendency to clog. In practice, therefore, the type of bricks employed and the size of flues allowed are governed by local conditions and are usually the result of trial and error.

Many operators prefer to use the standard bricks owing to the higher salvage and the reduced cost of installing the checkers. At some plants the salvage value of ordinary bricks at the end of a furnace campaign is as high as 60 per cent. The salvage value of special shaped bricks is usually low.

The pillars and bearer blocks are normally high grade firebricks, of sufficient crushing strength to carry the weight of the checkerwork. The checker bricks may be firebricks throughout or firebricks in the lower courses with silica bricks in the top courses, which are subject to more heat. In recent years there has been a tendency to use semi-silica bricks in regenerators, but this matter is discussed in the chapter dealing with refractories.

According to Siegle,[14] the qualitative thermal efficiency of regenerators may be expressed by the formula:

$$n = \frac{t_1 - t_2}{t_3 - t_2}$$

where n = qualitative thermal efficiency.

t_1 = temperature of the outgoing heated fluid.

t_2 = temperature of the incoming fluid which requires heating.

t_3 = temperature of the incoming heating fluid.

Therefore the efficiency is NIL when $t_1 = t_2$.

Attempts to increase the regenerative capacity by staggering the checker-brickwork is referred to by Robinson.[13] This results in decreased velocity, with corresponding greater heat transfer, but there is an increased tendency for reduction in draught and dust deposition.

The ratio of the size of gas to air regeneration varies from 1 : 1 to 1 : 3. If the analysis of the gas used as a fuel is known, the relative volumes of the gas and air regenerators can be calculated from the quantity of air required to burn a unit volume of the gas.

During the operation of the furnace large quantities of dust, iron ore, scale and lime are carried over by the waste gases, and these tend to choke the regenerators. *Slag pockets* are usually inserted between the furnace ports and the regenerators to avoid this trouble. These slag pockets should be of ample size, so as to collect as much of the dust as possible and to assure a reasonable length of life. A section through the regenerator and slag pocket of a modern furnace indicating their relative sizes is shown in Fig. 71. This relationship is also shown in Fig. 69. Frequently the floor of the slag pocket will slope from the regenerator dividing wall towards the wicket of the slag pocket. This floor is usually

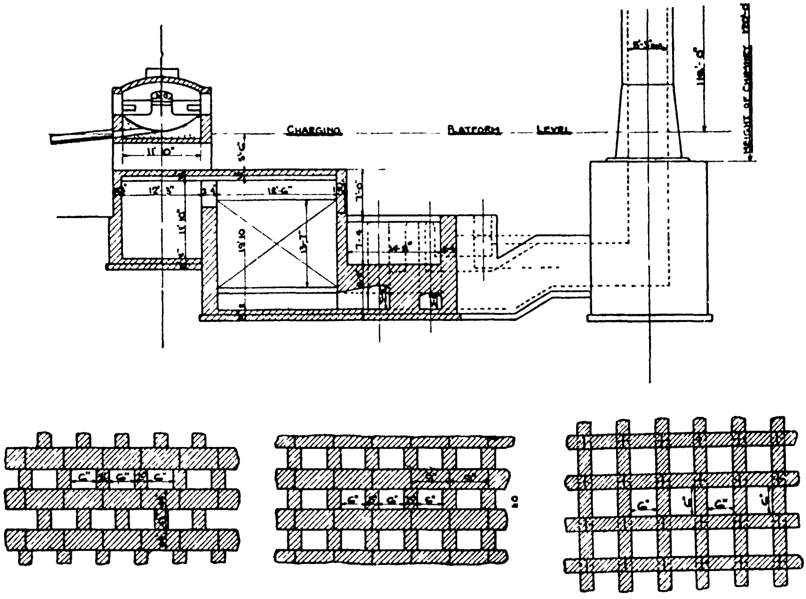


FIG. 71. Section through slag pocket and regenerator.

lined with ganister or loose sand to facilitate removal of slag at the end of a furnace campaign.

The regenerator and slag pockets are usually encased in $\frac{3}{8}$- or $\frac{1}{2}$-in. steel plates, securely bound together by binders of heavy section. Between the plates and the firebrick lining, which is usually 18 in. thick, a 4½-in. insulating course is frequently inserted.

The *flues* leading from the regenerators to the valves and from the valves to the stack should be of sufficient size to avoid any restriction to the flow. No precise formula has been laid down for ascertaining these dimensions, but according to Johnson[6] the cross-sectional area should be so proportioned that the hot gases have a velocity of between

TABLE XVIII

OPEN-HEARTH FURNACE PORTS AND UPTAKES

Source	Capacity of Furnace	Dimensions of Ports						Total Area of Uptakes in square feet	
		Gas			Air			Gas	Air
		Area	Length	Pitch	Area	Length	Pitch	Area	Area
		sq. ft.	ft. in		sq ft.	ft in.		sq. ft.	sq. ft.
Geary[10]	100	2·4	8 9	1 : 4·8	16·6	7 0	1 : 2·2	8·25	18·0
Drewery[15]	85	3·375	16 9	1 : 4·0	18·0	11 0	1 : 1·5	9·00	17·875
Open-Hearth Committee[3]	57	2·63	—	1 : 6·0	9·0	—	1 : 1·33	4·00	9·33
Open-Hearth Committee[3]	100	3·30	—	1 : 4·0	30·0	—	—	10·25	12·60
Open-Hearth Committee[3]	106	5·61	—	1 : 4·6	52·3	—	1 : 1·59	12·20	31·60

TABLE XIX

AVERAGE GAS AND AIR VELOCITY AT THE PORTS

Example	Process and Type of Furnace	Capacity	Average Time per Heat in Hours	Average Velocity at the Ports in ft per sec.		Ratio Gas Air Velocity
				Air	Gas	
1	Acid, Cold Charge, Fixed ..	42·6	14·75	28·90	51·05	1·78
2		80·6	11·17	14·00	71·00	5·08
3	Basic, Cold Charge, Fixed ..	114·3	15·00	13·85	71·00	5·18
4		61·8	11·75	15·00	88·00	5·72
5		100·0	10·25	13·08	89·50	6·87
6	Basic, Hot Metal, Fixed .. .	106·0	10·58	6·69	51·70	7·74
7	Basic, Hot Metal, Fixed .. .	54·6	8·25	9·80	74·00	7·55
8	Basic, Hot Metal, Tilter ..	121·2	15·70	23·20	72·30	3·11
9	Basic, Hot Metal, Tilter . .	251·0	17·41	32·20	97·70	3·03

6 to 10 ft. per sec. The flues for a 100-ton capacity furnace will frequently have a cross-sectional area of 20 to 22 sq. ft. In some cases the flues are encased so as to prevent gas leakage or air infiltration.

The *dampers* employed for regulating the draught or the flow of waste products through these flues may be of the dry alloy type or water-cooled castings. They should be capable of easy manipulation and encased so as to prevent air infiltration, and consequent loss of draught.

The *uptakes* leading from the slag pockets to the furnace ports should be designed to assure sufficient volume of air or gas being delivered to the ports to give the necessary velocity. Reference to Table XVIII will indicate the normal relationship of the area of the uptakes and ports. Usually the area of the uptake is much greater than that of the port in the case of the gas. The difference is not always large in the case of the air system. These uptakes should be as free as possible from sharp bends, as the hot gases tend to cause excessive erosion at such points.

The correct design of the furnace *ports* demands considerable attention. Much patient research has resulted in the evolution of several different types of ports. The essential features of the block or ports are such as will assure satisfactory length of life and good combustion conditions.

When designing the ports of an open-hearth furnace the following factors should be considered:

(1) The relative areas should be so dimensioned that satisfactory velocity is imparted to the flame.

(2) The angle of the air and gas ports should be arranged so that early and intimate mixing of the air and gas takes place, and that correct direction is imparted to the flame. Normally the inclination of the gas port varies from 12 deg. to 18 deg., whilst the angle of the air port will range from 30 deg. to 40 deg. The arch of the air port may be inclined at an angle up to 45 deg. The angle is usually greatest on small furnaces, where it is desired to obtain early combustion and to direct the flame on to the bath as soon as possible.

(3) The lateral convergence of the ports also assists the correct flame direction and velocity.

(4) As pointed out by Plumley,[16] the heat generated in the furnace depends on the amount of fuel burnt per unit of time. Therefore the maximum output will only be obtained when the three flowing streams of gas have reached their maximum. These three streams

are the gas, air and exhaust gases. It is obvious that the ports must be sufficiently large to admit the necessary quantities of gas or air, and yet not too large seriously to reduce the flame velocity.

In order to attain these conditions the following types of blocks have been developed:

(1) *Twin air ports*, as shown in Fig. 72, are still employed on some of the older furnaces. This block consists of a central gas port, with two converging air ports slightly above and to the side of it. A block of this type contains a considerable amount of brickwork and is fairly enduring, but it is not ideal for combustion. Combustion may be enhanced by careful attention to the relative pitch of the ports, and flame direction can be obtained by control of these angles and the angle of convergence. It is very difficult, however, to obtain intimate and early mixing of the air and gas.

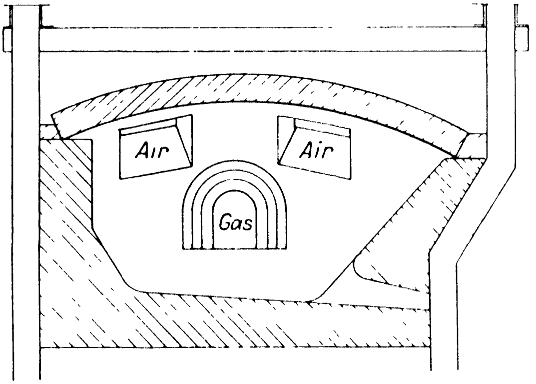


FIG. 72. Twin air ports

(2) The conventional fan port, Fig. 73, in which there is a single, inclined central gas port, with a wide air port immediately above it. The air port is inclined at an angle which favours early and intimate mixing and combustion. The port, correctly designed and dimensioned, will give ideal combustion and good flame direction.

Intimate mixing of the gas and air is favoured by a thin arch dividing the air and gas ports, but if the thickness of this arch is reduced too far, the structure will be weakened and the tendency for the arch to collapse under heat will be increased.

Flame temperature can be increased by a *pre-combustion chamber.* This pre-combustion chamber is the space between the nose of the gas port and the charge in the furnace. This space allows the gas and air to mix and enter into combustion before they meet the cold materials, thereby favouring the development of maximum flame temperature.

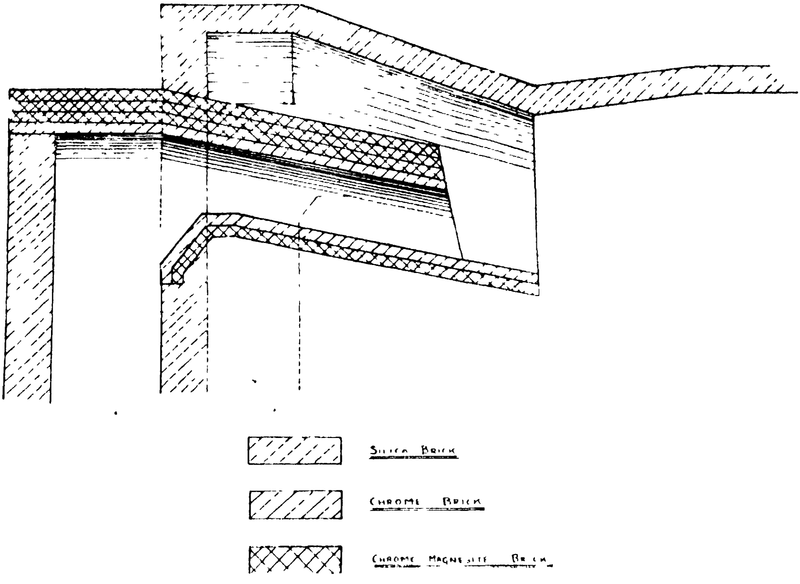


FIG 73 Conventional fan port
(By courtesy of the "British Steelmaker")

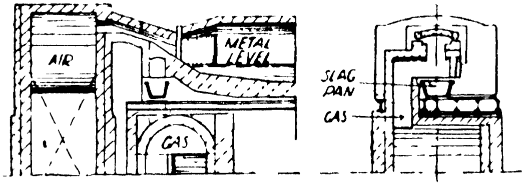


FIG 74. Moll port.
(By courtesy of the "British Steelmaker")

(3) *The Moll Port,* Fig. 74, originally developed in Germany, consists of an inclined air port, which gives intimate mixing and early combustion with the gas entering the chamber through a

short vertical uptake. Wood[17] states that this type of port gives good output and fuel economy, but large quantities of water are required for cooling purposes. Flame direction and rapid combustion are obtained by the angle of the air port and the relative size of the air port and uptake which gives the necessary velocity.

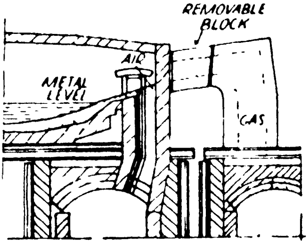


FIG. 75. Maerz port.
(*By courtesy of the " British Steelmaker"*)

(4) *The Maerz Port,* discussed by Killing,[18] is shown in Fig. 75. The chief feature of this block is the removable nose of the gas port. This portion of the block, which is subject to rapid wear and destruction, is designed to facilitate quick and easy removal and replacement. The gas uptake is arranged externally for cooling purposes. The angle of the gas port imparts good direction to the flame, and assures intimate mixing with the air required for combustion. This air enters the furnace through short vertical uptakes situated at either side of the gas port.

One great advantage of this design is that no water-cooling is necessary. Early and rapid combustion is claimed, and very high rates of production per unit of bath area have been attained.

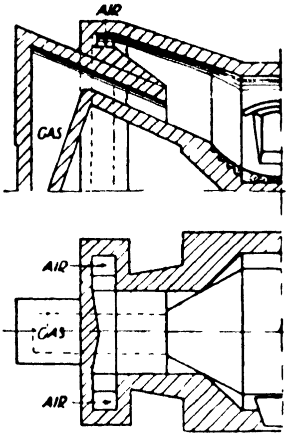


FIG. 76. Terni port.
(*By courtesy of the " British Steelmaker"*)

(5) *The Terni Port,* shown in Fig. 76, was first developed in Italy, but has been successfully applied in this country. It is a development of the conventional fan port, in which the angles of the air and gas ports favour intimate mixing of the air and gas with the production of a very hot flame. The different pitch of the two ports and the pre-combustion chamber, clearly shown in Fig. 76, assures the attainment of maximum flame temperature and imparts a turbulence to the flame, which gives rapid heat transfer.

Usually the air is supplied by a fan and this enables perfect control of flame velocity. Maximum air is used during the melting-down period, with the production of a short, hot, cutting-down flame. A reduction in the air pressure gives a longer radiating flame, more suitable for the refining operations.

The flame produced by the Terni port is severe on the block structure, but the application of water-cooling to the nose of the gas port considerably prolongs its life.

The advantages and construction of this type of block, together with operating data, has been given by Fiorelli.[19]

(6) *The Venturi Port* was developed in America but has found great favour in this country, owing to the high rates of production obtained. As shown in Fig. 77, this design consists of a short nearly horizontal gas port, which is extensively cooled at the nose.

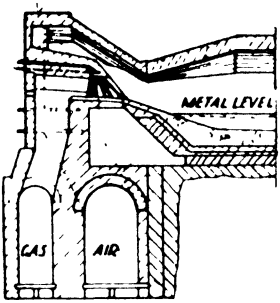


FIG. 77. Venturi port.

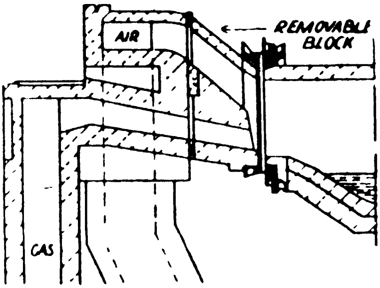


FIG. 78 Friedrick port.

(*By courtesy of the "British Steelmaker"*)

This port is situated immediately behind the "venturi" throat, which acts as a mixing and pre-combustion chamber.

On leaving the uptake the gas passes along the port and is intimately mixed with the air at the approach to the throat, resulting in the production of a very hot flame. The mixing of the air and gas is greatly enhanced by the angle of the air port and arch, whilst combustion is assisted by the pre-combustion chamber or throat. The convergence of the port gives velocity to the flame, whilst the opening out of the throat produces a venturi effect which favours heat transfer.

One disadvantage of this type of port construction is the large quantity of water required for cooling purposes.

(7) *The Friedrick Port* is another modification of the fan type port, with a removable nose. This design, clearly illustrated in Fig. 78, has been very successfully applied to many large tilting furnaces.

(8) *The Hoesch Port* was designed for use with cold coke-oven gas. It consists of a large converging air port into which coke-oven gas is supplied by a burner. Tar burners are also inserted. The general construction of this block is shown in Fig. 79.

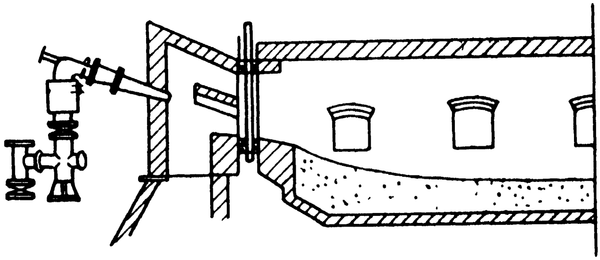


FIG. 79 Hoesch port.
(*By courtesy of the " British Steelmaker "*)

In recent years the operational life of furnace blocks has been considerably prolonged by water-cooling. This water-cooling is very extensive in some designs. It may take the form of steel pipes, 1½ to 2 in.

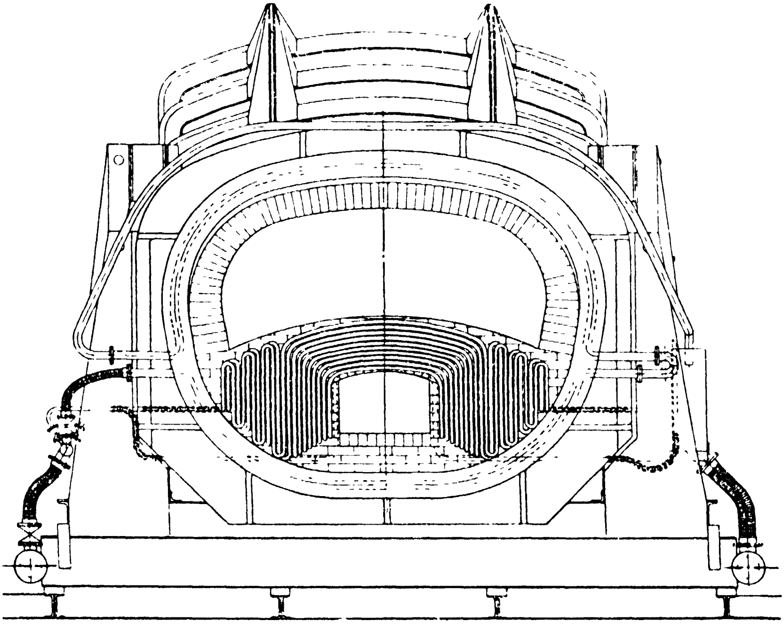


FIG. 80. Water-cooling of furnace ports.
(*By courtesy of the " Iron and Steel Institute ".*)

dia., through which water flows continuously. These pipes, which follow the shape of the port, are placed inside the brickwork of the gas arch. Sometimes "torpedo" tubes running from the rear to the nose of the port are employed. Steel chills are also used. Some methods of water-cooling are illustrated in Figs. 69, 77, 80 and 81.

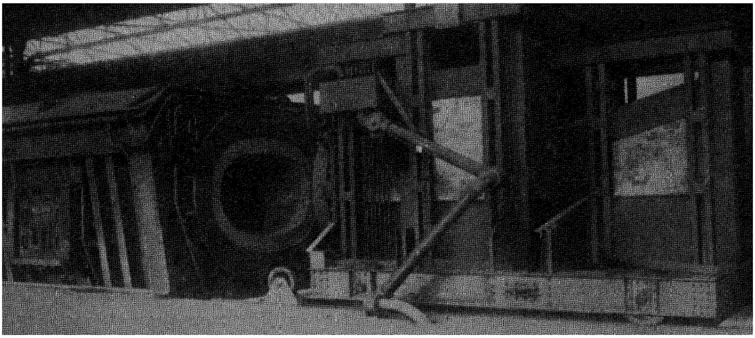


FIG. 81. Water-cooled ends of a tilting furnace.
(*By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.*)

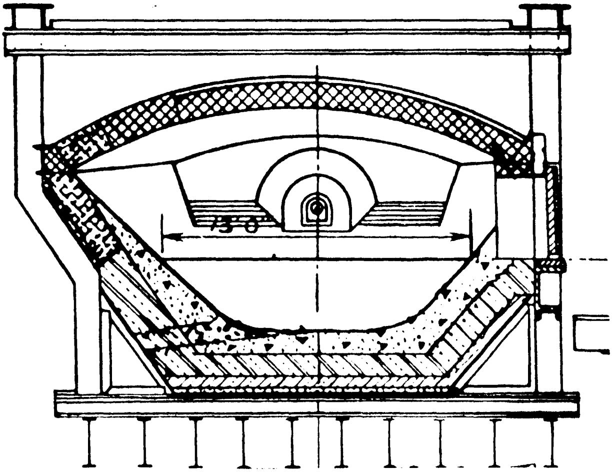


FIG. 82. Naismith sloping back wall.

On practically all modern furnaces the straight *back wall* has been replaced by some modification of the sloping back wall. This development has greatly facilitated the repair, by fettling, of this vulnerable part, and has thereby prolonged its life. The original Naismith sloping back wall was designed to slope outwards at an angle equal to the angle of repose of dolomite. As shown in Fig. 82, this design enabled the back wall to be fettled successfully.

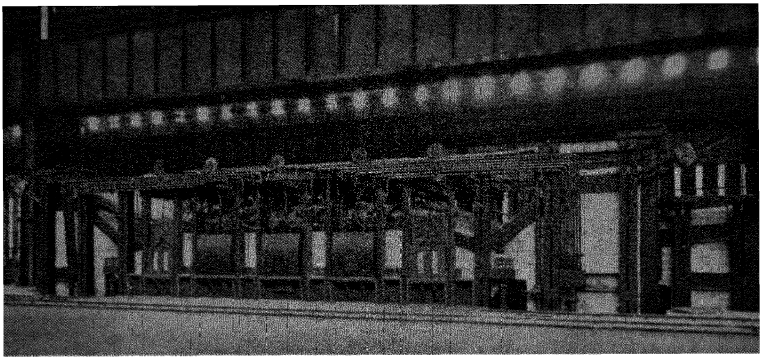


FIG. 83. Charging-side of a modern furnace.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.)

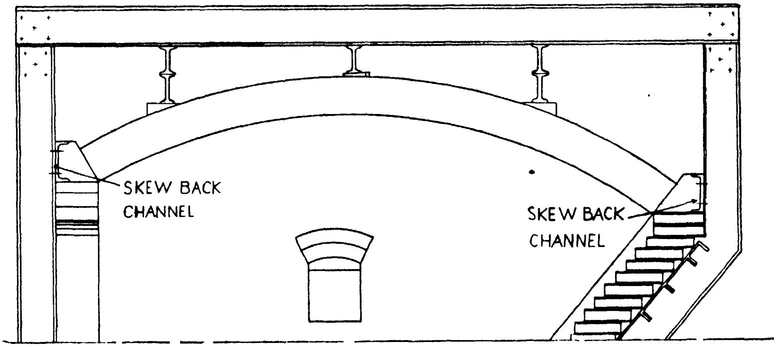


FIG. 84. Section through furnace showing skew-back.
(By courtesy of the "British Steelmaker".)

Water-cooled arches and door jambs have replaced much of the brickwork in the front walls of modern open-hearth furnaces. Water-cooled doors are also incorporated in the latest designs. A view of the charging side of a modern furnace erected by the Wellman Smith Owen Engineering Corporation Ltd. is shown in Fig. 83.

The furnace roof, which at the majority of works is still constructed of silica bricks, is sprung from a skew-back channel. This channel should be supported by the furnace structure and should be independent of the front or back wall. The skew-back brick, on which the roof is sprung, is held by this channel, which should have strength and rigidity. At plants where archless water-cooled door frames are installed, the lower web of the skew-back channel is also water-cooled.

The rise of the roof varies at different works. A fairly average and satisfactory rise appears to be 1·4 to 1·6 in. per ft. of span.

The roof thickness varies, but 12 in. appears to be the most favoured thickness. In some cases this thickness is increased to 15 or 18 in. As Reinartz[5] states, the roof span of a 150-ton furnace is considerable, and in order to obtain additional strength such a roof is frequently constructed of 15 or 18 in. bricks, and 18 or 22 in. reinforcing rib courses at intervals. At some works 15 in. bricks are used for a width of 5 or 6 ft. along the front and back walls, where erosion is most severe, whilst the central section of the roof consists of 12 in. bricks.

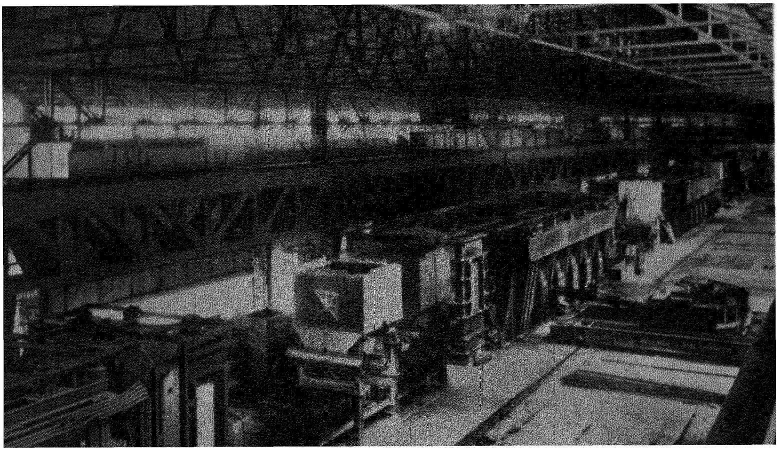


FIG. 85. A battery of fixed open-hearth furnaces.
(By courtesy of the International Construction Co., Ltd.)

The height of the skew-back channel above sill-plate or foreplate level varies, but normally this dimension will be 5 or 6 ft.

It is important that the roof is carefully packed when completed so that the expansion which takes place as it is brought up to heat can be controlled. Steel flats are generally laid across the roof laterally at intervals and over these three channels run longitudinally. One of these channels is placed in the centre of the roof, whilst the other two are equidistant on either side of it. All three channels are wedged to the cross-members with just sufficient clearance to allow the necessary expansion to take place evenly. Failure to control the rise may result in uneven lifting with subsequent collapse of the roof. This method of packing the roof can be seen in Fig. 85.

In recent years the introduction of chrome magnesite, commonly referred to as "basic", bricks has resulted in some modification to furnace design, especially when used in roof construction. These bricks, which will be discussed under steelworks' refractories, were originally employed in furnace ports, but later they were applied to the open-hearth roof. When basic bricks are used, the conventional method of roof construction has to be modified, as pointed out by Robertson,[20] for three reasons:

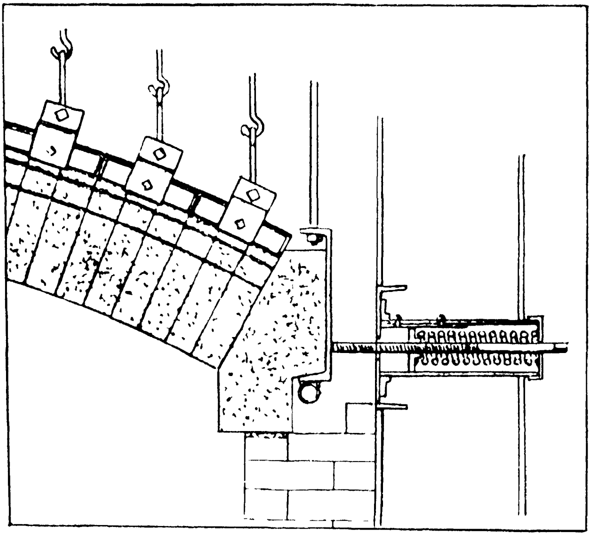


FIG 86. Suspended roof.
(*By courtesy of the " British Steelmaker "*)

(1) The weight of the brick is much greater than in the case of silica.

(2) At the temperature of operation, their crushing strength is much less than silica bricks.

(3) The continual expansion and contraction during the different phases of the process, which is much different from silica, presents a new problem.

These difficulties are overcome by replacing the conventional sprung arch by a completely suspended roof. The bricks are provided with either slots or projections, which are engaged by supports attached to the cross-members of the superstructure. A spring device is also incorporated at the skew-back, as shown in Fig. 86. The design and construction of the basic roof has been discussed in detail by Lindemuth.[12]

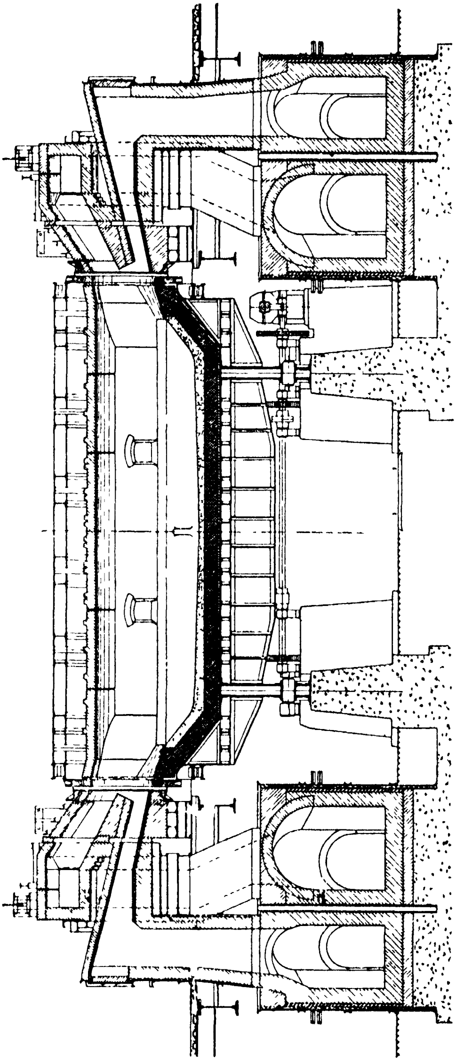


FIG. 87 A 200-ton tilting furnace—longitudinal section

(By courtesy of the Iron and Steel Institute)

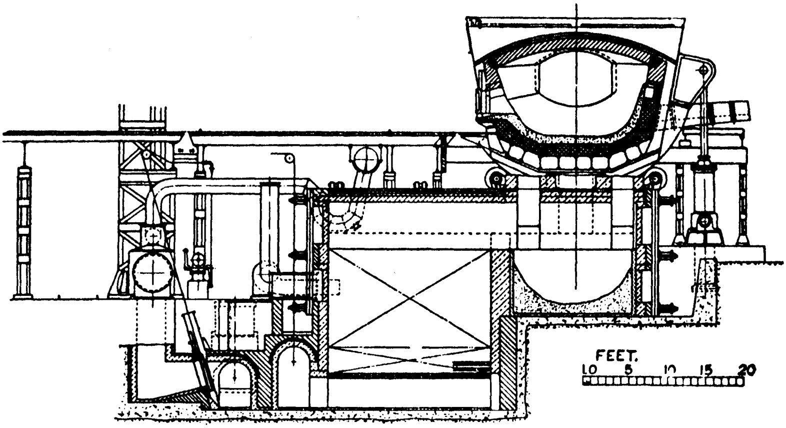


FIG. 88. 300-ton tilting furnace—cross-section.
(By courtesy of the Iron and Steel Institute.)

Tilting Furnaces

The fundamental differences in the construction of the fixed and tilting type of open-hearth furnaces are the foundations of the hearth and the design of the port and furnace ends.

The normal concrete pillars supporting the hearth are replaced by

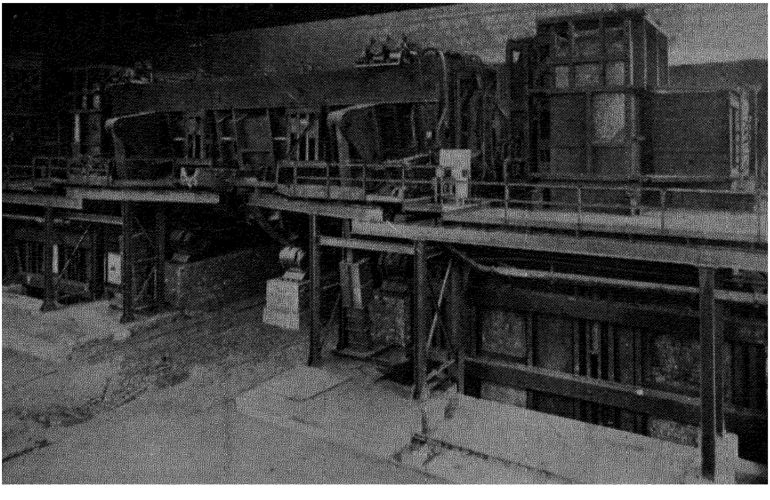


FIG. 89. Tapping-side of a 300-ton tilter.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.)

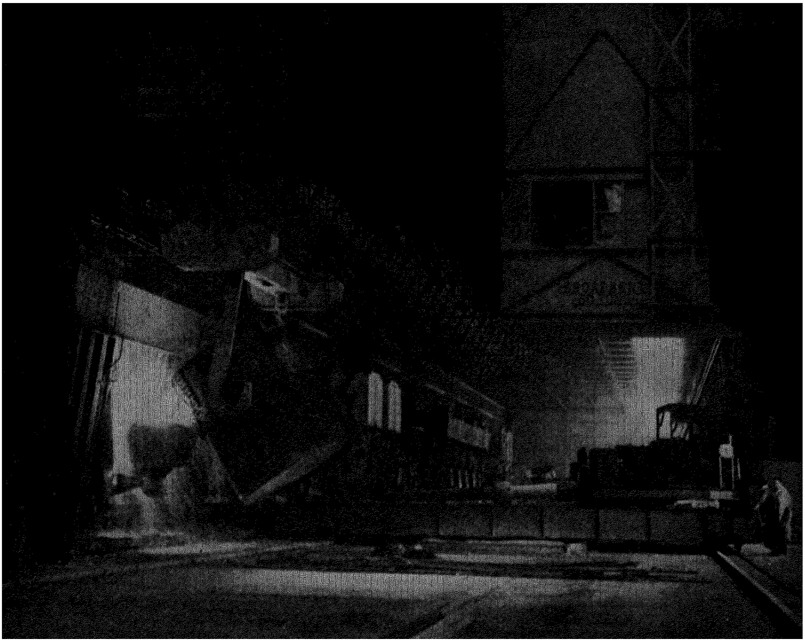


FIG. 90. Charging-side of a large tilting furnace.

(From "The Metallurgy of Deep Drawing and Pressing", by Dr. J. Dudley Jevons, by courtesy of the author.)

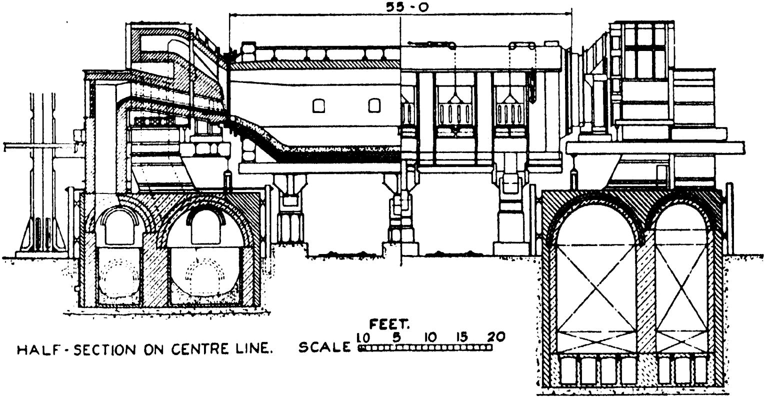


FIG. 91. A 300-ton open-hearth furnace.

(By courtesy of the Iron and Steel Institute.)

rollers or rockers on which the furnace rotates. These are shown in Figs. 87, 88 and 89. A furnace which rotates concentrically with the furnace ports is sometimes referred to as a tilting furnace, whereas if the furnace rotates eccentrically to the ports it is known as a rocking furnace. The chief advantage of concentric rotation is the fact that the gas can be kept on during slagging and tapping. It is important that the rollers provide ample support to the furnace structure. They should be of strong design, provided with rigid foundations, as they have a heavy load to carry.

The tilting gear may be operated electrically or hydraulically, and usually allows the furnace to be tilted about 10 deg. to the charging side for slagging, and through an angle of 20 to 30 deg. for tapping.

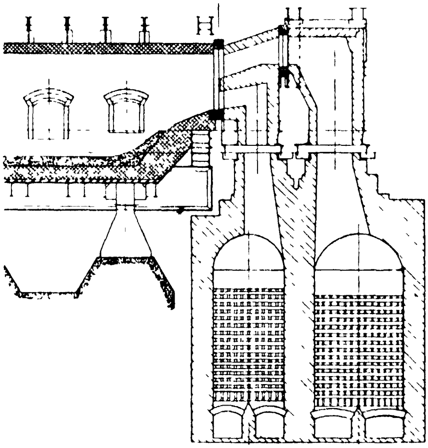


Fig. 92 Removable blocks for tilting furnaces.
(By courtesy of the Iron and Steel Institute.)

The chief feature of the furnace shown in Fig. 92 and described by Winter,[22] is the removable port-block. The port ends can be drawn back to allow the furnace to tilt during tapping and slagging. The port ends are lifted clear of the water-seals and drawn back by hydraulic power. Cast-iron water-cooled chills form the junction between the port uptakes and the uptakes leading from the slag pockets. Another type of removable port ends are shown in Fig. 93.

The tilting gear of the 300-ton furnace at Appleby-Frodingham[23,24] is operated electrically. Two rams are attached to the furnace rockers and are operated by means of racks and pinions. Two 80-h.p. motors are provided, and are arranged so that at any failure of one motor, the other motor can be brought into action immediately.

The hearth of this furnace is 45 ft. long and 15 ft. wide, giving a total area of 675 sq. ft. The body of the furnace is efficiently strengthened with vertical rolled steel buckstays, which are cross-connected over the top of the furnace by means of rolled steel sections and plates rigidly fastened to the vertical members. These are further strengthened by diagonal and longitudinal tie members, thereby ensuring rigidity and

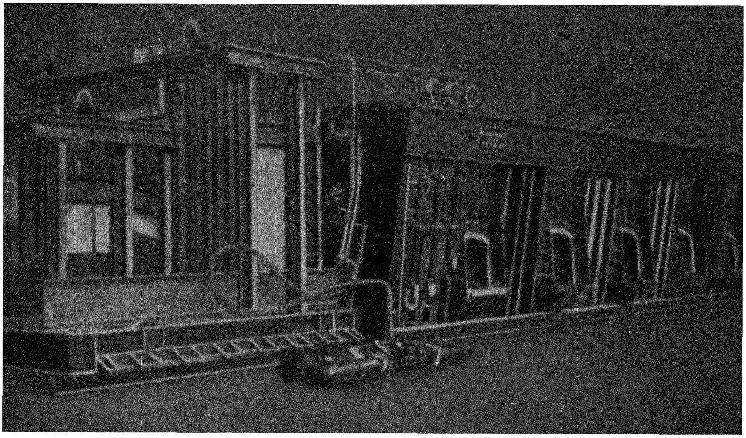
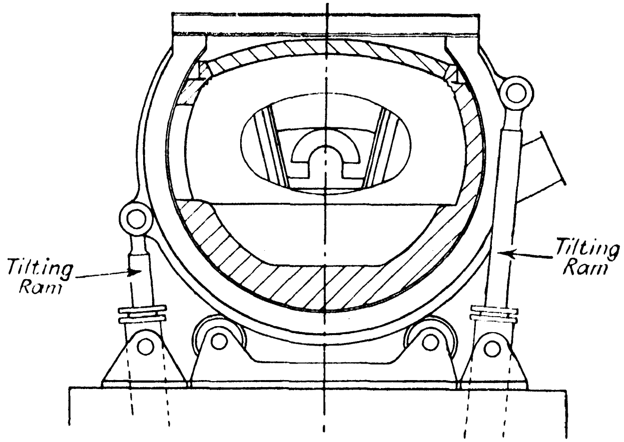


FIG. 93. Removable blocks.

(From "Practical Steelmaking", by W. Lister, by courtesy of the author.)



Section thro' Tilting Furnace

FIG. 94. Hydraulic tilting gear.

preventing distortion due to heat. The bottoms of the vertical members are joined securely to the rockers and the rolled steel sections, which extend under the body of the furnace. The bottom of the furnace is carried by heavy steel joists, which are supported by the rockers.

One rocker is immediately on the centre line of the furnace, whilst the other two are equidistant about the centre line and are placed sufficiently far apart to assure stability. These rockers operate on twelve large-diameter cast-steel rollers. Typical views of this furnace are shown in Figs. 88, 89, 90 and 91.

When the tilting gear is operated hydraulically, two cylinders are provided on either side of the furnace, as shown in Fig. 94. The cylinders are so designed that they move through a small angle as the furnace tilts.

On furnaces where the port ends are stationary, the space between the port end and furnace end must be regulated carefully to allow free movement during tilting and yet prevent excessive air infiltration. Frequently a loose ring, which can be moved towards the furnace end, is fitted over the port end ring. This reduces materially air infiltration.

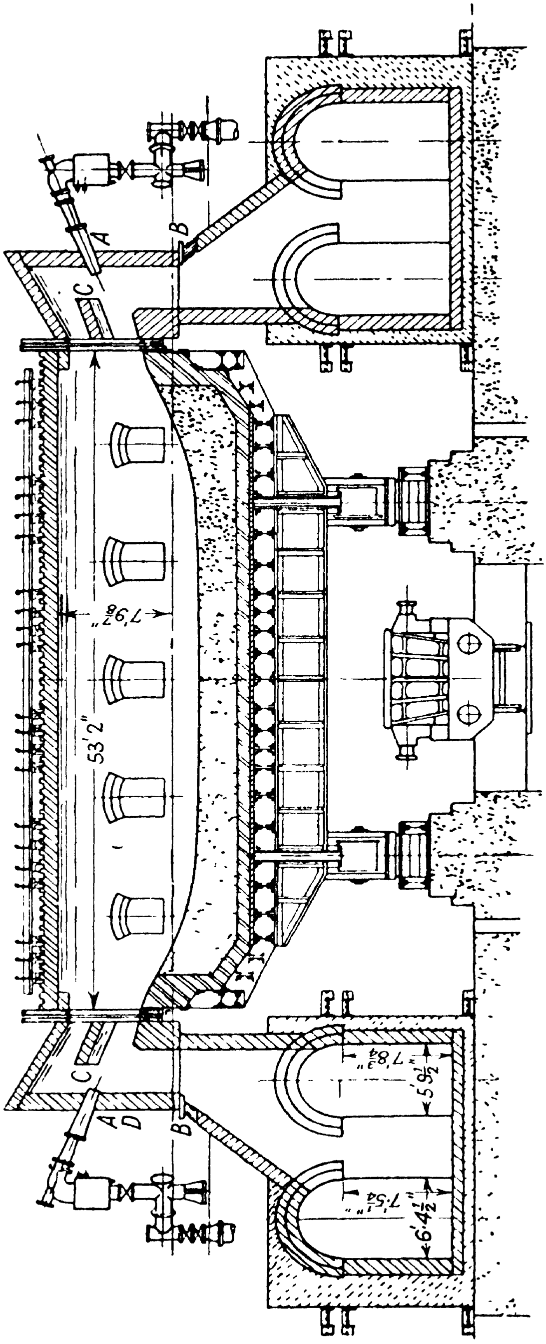
Mixed Gas Furnaces

When coke-oven gas, or mixed gas, is used as a fuel without an illuminant, the conventional type of furnace with air and gas regenerators is employed. In this case luminosity is imparted to the flame by decomposition of the hydrocarbons by pre-heating. Wesemann[25] shows that the decomposition of these hydrocarbons commences around 800 deg. C. and reaches its maximum velocity at 1,250 deg. C.

At the plant discussed by Davies,[8] the furnaces are operated with regenerator temperatures of 1,000 to 1,200 deg. C. for the gas, and 1,100 to 1,350 deg. C. for the air. If the moisture content of the gases is high, a slightly lower temperature, 900 to 1,100 deg. C., is employed in the gas regenerator. This lowering of the temperature reduces the velocity of the water-gas reaction and has given very satisfactory results.

When coke-oven or mixed gas is used with an illuminant, no gas regenerators are employed. On old furnaces which have been converted to this method of firing, the two regenerators are joined, and are connected with the air circuit, as shown in Fig. 95. In new furnaces designed for coke-oven gas, one large air regenerator is provided at each end of the furnace.

The coke-oven gas is admitted through burners (*A* in Fig. 95) inserted at the rear of the air port. Frequently a secondary arch (*C*), to deflect the flame downwards, is provided in the air port. The illuminant



A. Burner B. Flat Water-Cooler C. Secondary Arch

FIG 95 Coke-oven-gas fired open-hearth furnace.

(By courtesy of the Iron and Steel Institute) is normally introduced by inserting a long water-cooled tar or oil burner below the gas burners (as *D* in Fig. 95).

The usual type of water-sealed control and reversing valves employed with coke-oven gas is shown in Fig. 96. When mixed gases are employed the gases are generally mixed at a special valve before the gas proceeds to the reversing valves. This mixing valve may have two or three ports according to whether coke-oven and blast-furnace, or coke-oven, blast-furnace and producer-gas are being used.

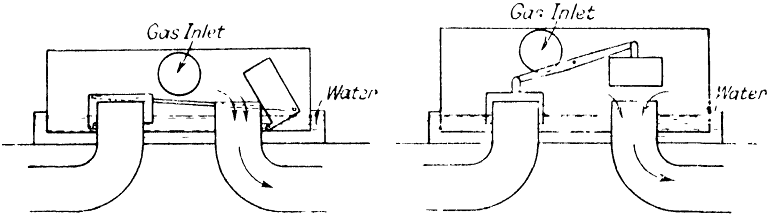


FIG. 96 Coke-oven-gas control and reversing valves

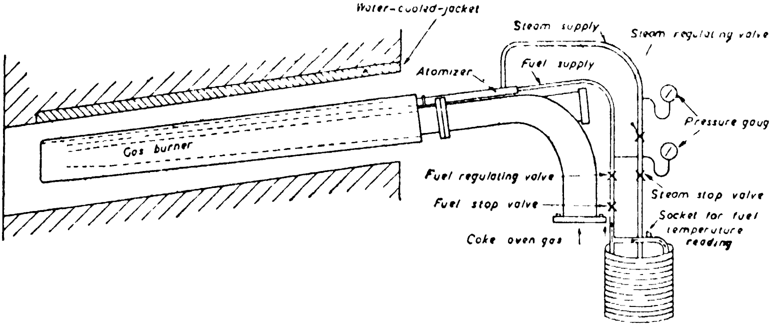


FIG. 97. Coke-oven-gas burner
(*By courtesy of the Iron and Steel Institute*)

Oil-Fired Open-Hearth Furnaces

A typical oil-fired open-hearth furnace, together with the control arrangement, is illustrated in Fig. 98. It will be noted that there are two air regenerators at either end of the furnace, whilst the oil is admitted by a burner, which has an adequate brickwork protection. The brickwork through which the burner projects is referred to as the "dog-house", and is so dimensioned as to allow a slight adjustment to the angle of the burner. Lateral adjustment of the burner is also desirable to correct flame direction. This sheath of brickwork provides a sight-hole through which the flame can be observed.

In smaller furnaces up to about 100-ton capacity one large air regenerator at either end of the furnace is provided, but on larger furnaces two regenerators at either end are favoured. On larger furnaces the span of the regenerator arch tends to become excessive and is thereby weakened. The provision of two smaller regenerators overcomes this disadvantage and in addition reduces the size and weight of the reversing mechanism.

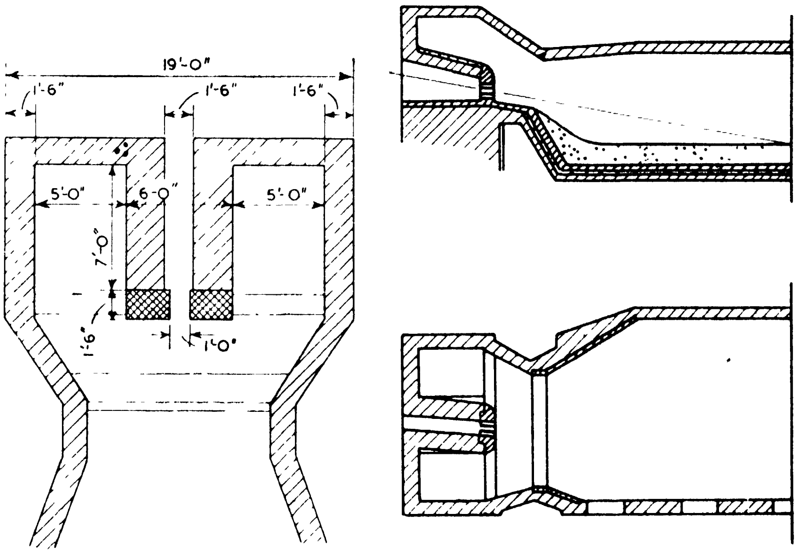


FIG. 98. Oil-fired open-hearth furnace.

So that the temperature of each regenerator can be controlled, each flue is equipped with a damper, by means of which the flow of the waste-products can be regulated.

Generally the air is supplied to the reversing valves by a common fan of adequate capacity. The amount of air can be controlled by means of vanes, or a butterfly damper, in the common duct leading from the fan before it forks to the two reversing valves.

The air port, which practically encircles the burner, is of the fan type and an adequate pre-combustion space is generally provided. This air port is connected with the air regenerators by two large uptakes on either side of the dog-house. The dimensions of these uptakes are of considerable importance, as they have to deal with all the products of combustion, without causing any restriction or involving excessive velocities which would increase erosion.

The normal method of oil flow is indicated in Fig. 64, consisting usually of a ring main leading from suitable storage tanks to reservoirs at each furnace. In many cases the oil flow is reversed by simple three-way L-type port valves, in which one leg is connected with the oil supply and the other two legs to either burner. This reversing device should be synchronized with a similiar arrangement for reversing the steam required for atomization.

In the case of the steam a by-pass is provided to allow a certain amount of steam to by-pass the reversing mechanism when on the exhaust end. This "dead-end" steam, which may amount to about 2 lb. per hr., is necessary to prevent carbon deposition in the burner, when on the exhaust end of the furnace.

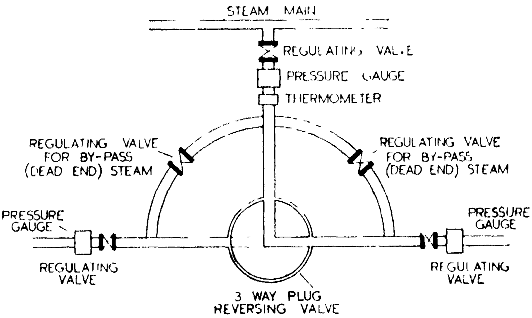


FIG. 99. Oil control valves.

Although the simple three-way valve has given very satisfactory results at many plants, other types of valves, similiar to those shown in Fig. 99, have been developed.

Some authorities recommend that oil-fired furnaces should be slightly longer than other designs, to assure complete combustion of the oil before the flame strikes the outgoing block. This feature, however, would appear to be a function of correct burner design and correct atomization.

The accuracy with which oil fuel can be applied merits the full application of instruments. The subject of instrumentation of open-hearth furnaces, however, is discussed in Chapter 14.

Reinartz[5] mentions that at some American plants using oil and gaseous fuel, the Isley system has been introduced. The usual valves are replaced by two "Venturi" stacks, one at either end of the furnace, as shown in Fig. 100. A large fan, operated by a 100-h.p. motor, is connected to each stack. The system is so arranged that on the incoming end the air is driven into the furnace under perfect control, whilst on the exit end the gases are pulled through the stack. Reversal is accomplished by changing the damper settings in the stacks.

It is claimed that this type of installation eliminates all air infiltration, avoids cooling around the valves, gives accurate control of operation and results in considerable economy.

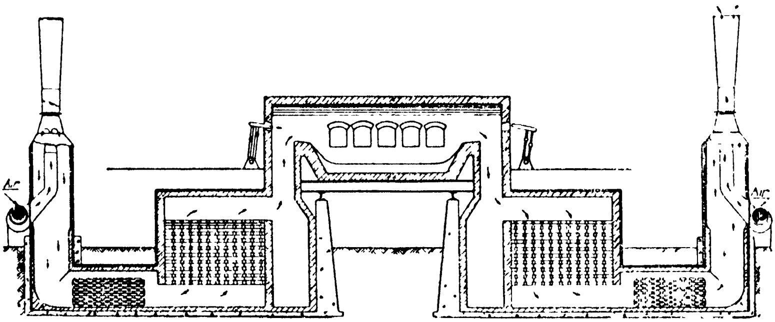


FIG. 100. Isley furnace
(By courtesy of the Iron and Steel Institute)

Furnace Stacks and Waste Heat Boilers

It is obvious from the work of Plumley[16] that however efficient port design and combustion may be, maximum output can only be obtained when the furnace flues and stack are capable of dealing effectively with the products of combustion. Any restriction to the flow of the waste gas must slow down the rate of production. It is, therefore, important that when dimensioning the furnace chimney, ample reserve of draught is obtained. On modern furnaces the stack will vary from 160 to 180 or more feet in height and 6 to 8 ft. in internal diameter, as shown in Table XV.

Hermanns[1] recommends that the area of the flue between the valves and the chimney should be equal to the areas of the air and gas flues. The area of the chimney at the mouth should be of the order of 500 sq. cm. per sq. m. of hearth area. He[1] suggests that the internal diameter of the chimney may be calculated from the formula:

$$d = \frac{F}{4}\,\text{m}$$

where F = Area of the hearth.
$m = L : B$
L = length of hearth.
B = width of hearth.
d = internal diameter of mouth of chimney.

As pointed out by Broome,[26] however, the open-hearth furnace provides one of the most satisfactory applications of waste heat boilers. He[26] shows that a 90-ton modern furnace producing 1,400 tons of steel per week, with a fuel consumption at the producers of 5 cwt. per ton of steel, is capable of generating 1,965,000 lb. of steam at 212 deg. F. with an efficient waste heat boiler. This is equivalent to a saving of 100 tons of coal per week.

One important factor is the selection of a boiler of adequate size and horse power. Under no consideration should boiler size or fan power restrict the flow of waste gases, otherwise the rate of output will fall. The author has known plants where the installation of waste-heat boilers, with adequate fan power, has actually increased the draught available and thereby added to higher rates of production.

The design, installation and operation of waste-heat boilers in conjunction with open-hearth furnaces has been thoroughly discussed by Webber.[27]

Normally the waste heat boilers employed are of the fire-tube type, in which the waste gases pass through tubes which are surrounded with water. These gases are drawn through the tubes by means of a fan on the exhaust end of the boiler, where they are discharged into a suitable stack. The general layout is shown in Fig. 101. Generally the waste heat boiler is placed on a by-pass flue between the reversing valves and the main stack. By a system of efficient dampers, either the natural draught of the main stack, or the induced draught of the waste heat boiler may be utilized.

Although the temperatures prevailing vary considerably, at the entrance to the waste heat boiler, temperatures of 550 to 600 deg. C. are quite normal. The temperature at the exhaust end of the boiler may average 180 to 250 deg. C. according to the efficiency of the boiler. The heat transfer and the draught may be adversely affected by deposits on the tubes. These deposits are prone to form when galvanized scrap is used, especially if any water leakage occurs in the valves or flues.

The design of waste heat boilers is a specialized engineering task, and it has been thoroughly discussed in a special report of the Iron and

Steel Institute.[27] In this report Webber[27] calculates the dimensions of a waste heat boiler for a 50-ton furnace, which he summarizes as follows:

Boiler length	16 ft.
Boiler diameter	7 ft. 6 in.
Number of tubes	339
External tube diameter	2 in.
Boiler—heating surface tubes	2,490 sq. ft.
Superheater	80 sq. ft.
Total	2,570 sq. ft.
Average steam output	3,300 lb. per hr.
Draught loss across boiler	1·85 w.g.
Average fan power.	14·5 kW.

Broome[26] gives the results and details of a series of trials on two waste heat boilers working in conjunction with two 90-ton open-hearth furnaces. The average evaporation per hour from and at 212 deg. F., was 16,000 to 20,000 lb. and the average power required for the fan was 41·5 to 56·9 kW. The variation in draught at different points in the system was:

At the furnace	1·0 to 1·2 in. w.g.
At boiler inlet	1·6 to 1·7 in. w.g.
At boiler outlet	4·8 to 6·0 in. w.g.

These figures show a considerable drop in draught across the boiler and thereby stress the need of ample fan power if unrestricted flow of waste gases from the furnace is to be maintained.

On the waste heat boiler quoted by Geary and others,[9] working in conjunction with a 250-ton tilting furnace, the total heating surface was 4,128 sq. ft. and the suction at the boiler outlet was 5·⅞ in. w.g. This boiler was provided with a 100 b.h.p. fan.

Insulation

Williams [28] states that complete insulation of open-hearth furnaces has given an increased roof life of 10 to 25 per cent. and a fuel economy of 10 to 15 per cent. Reinartz,[5] however, points out that the benefits of insulation appear to depend on the design of the furnace, the amount and kind of insulation used, the skill of application, and the personnel of the operating organization. He[5] states that estimates show that some plants which have applied insulation generally below the charging floor level have obtained a 5 per cent. increase in the melting rate and a 7 per cent. decrease in the fuel consumption. Insulation above the level of the charging floor, according to some engineers, has resulted in a 15 per cent. fuel economy, but no increased rate of output.

Modern practice appears to favour insulation below the charging floor level only. The insulation of the furnace roof is a debatable subject. There is much evidence to show that it results in increased refractory costs. It also tends to slow down the rate of production, because operatives are afraid of overheating their roof.

Slag pockets and regenerators are frequently insulated with 4½ in. of sil-o-cell brick permanently bricked into the walls of the chambers. Generally this insulating course is placed between the firebrick lining and the steel plates encasing the chambers.

The insulation of the furnace hearth has been tried at many plants, but over-insulation must be avoided. It has been found that too much insulation results in excessive fettling and bottom trouble.

Reinartz[29] states that many American plants have adopted a very successful method of insulating the flues to the valves and stack. This method consists of installing around the flue a 9-inch concrete wall containing insulating material, such as one part of Portland cement, 6 parts of insulating aggregate, ¼ to 1 in. in dia., and about 2 parts of insulating powder mixed with water.

Such materials as slag wool, asbestos, silocel powder and powdered alumina are frequently used as insulating materials in America.[29] Colloidal clay, sodium silicate, organic materials or asphalt are used as binders. These materials in general have a thermal conductivity about one-sixth that of a normal firebrick. One plant quoted by Reinartz[5] employs 1½ in. of silica sand and the 2 to 3 in. of granulated blast furnace slag on the top of the checker chamber roofs.

Clements[11] assesses the total heat lost in an open-hearth furnace as follows:

	Per cent.
(1) Radiation from bath and port ends	44·2
(2) Heat lost with gases to the chimney	36·2
(3) Radiation from the regenerators .. .	15·5
(4) Radiation from the flues	4·1

Drying and Gassing Up Furnaces

The method of drying and gassing up furnaces plays an important part in their operational life. The need of thorough and regulated drying and warming up of the furnace structure is dealt with in the chapter on steelworks refractories. Any tendency to rush the brickwork through critical temperature ranges may result in serious damage to brickwork and corresponding reduction of operational life.

When the furnace has been dried and is sufficiently warm for the stack to draught, it can be gassed. Usually a wood kindling fire is put into the furnace and the gas admitted. The gas control valve should be lifted at a steady and even rate. When the gas has once ignited, the pressure should not be allowed to drop until the flame is well established and fairly steady. A long smoking flame, which passes down the outgoing ports, thereby heating up the exit regenerators, should be maintained during this period.

Normally the first reversal of a new or recently repaired furnace will take place three to five hours after gassing. The actual time will depend on the manner in which the furnace stack is draughting. Subsequent reversals will be made more frequently until normal reversal periods of ten to twenty minutes are reached.

For the first three or four reversals, the valves will be held in a central position during the reversal, so as to allow the stack to draw all the gas out of the regenerators before it is admitted into the opposite end. At this stage the draught may be insufficient to draw the gases away quickly and if the valves are put straight over on to the opposite end, explosive mixtures in the slag pockets and regenerators may result.

An excellent method of drying and gassing new or repaired furnaces with coke-oven and mixed gas has been described by Davies.[8] When the brickwork is completed, a thermo-couple is fixed in the centre of the roof and then blast-furnace gas pipes are introduced through each door and through each slag pocket wicket. Owing to the fact that blast-furnace gas is easily extinguished, a small coke fire is started before the gas is turned in to the pipes. This coke fire is maintained to keep the flame alight. The reversing mechanism is kept centred thereby exhausting all flues to the stack.

It is endeavoured to raise the roof at a rate of 10 deg. C. per hr. until a temperature of 700 deg. C. is attained. The furnace is then ready for gassing. The gas pipes are then withdrawn from the slag pockets and the mains leading from the blast-furnace gas control valve to the reversing valves is thoroughly purged before the reversing valves are moved and gas admitted to the furnace. When the mains have been completely purged, some wood is thrown into the port, the chimney damper opened fully, and the reversing valves put on to the correct end. The blast-furnace gas will then issue from the port and burn brightly. When the flame has been established, the burners are withdrawn from the doors.

For the first three hours the blast-furnace gas is set at 250,000 cu. ft. per hr., by which time the furnace should have reached a temperature of 900 to 1,000 deg. C. Coke-oven gas at the rate of 50,000 cu. ft. per hr. is admitted and the blast-furnace gas is reduced to 200,000 cu. ft. per hr. About two hours later, five hours after gassing, the furnace is ready for the first reversal.

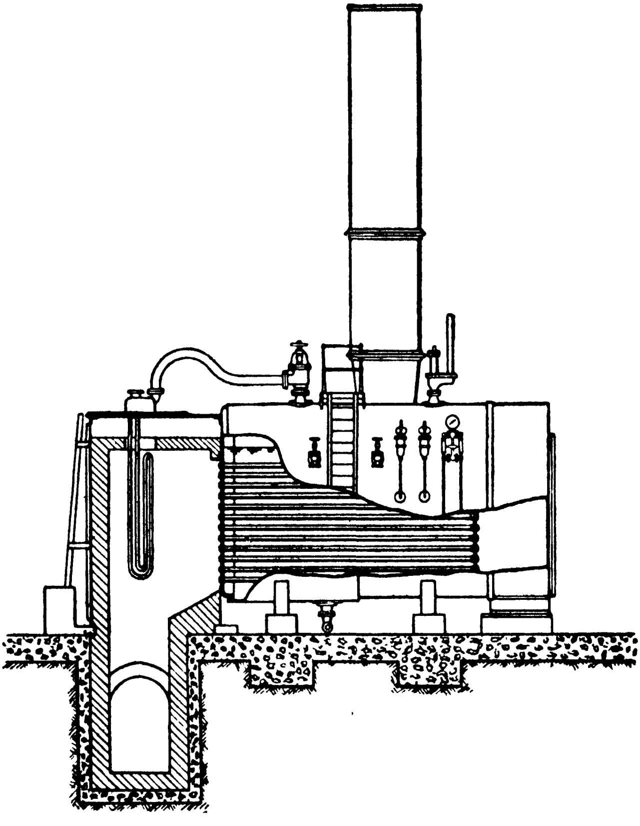


FIG. 101. Typical waste-heat boiler.

(By courtesy of " Iron and Steel ")

For the first two days it is advisable to take the coke-oven gas off during the reversals. Three or four minutes before the reversal is due, the coke-oven gas is shut off completely and the volume of blast-furnace gas increased. When the gas flow has been reversed the blast-furnace gas is reduced and the coke-oven gas admitted. The mixture of coke-oven and blast-furnace gas is gradually adjusted after the first twenty-four hours to its normal working proportions.

Metal Mixers

Mixers, which may be active or inactive, play an important role in modern melting-shop practice. In general, their design and construction is similar to modern tilting furnaces. Typical examples of inactive mixers are shown in Figs. 3 and 102, whilst Fig. 103 shows an active mixer.

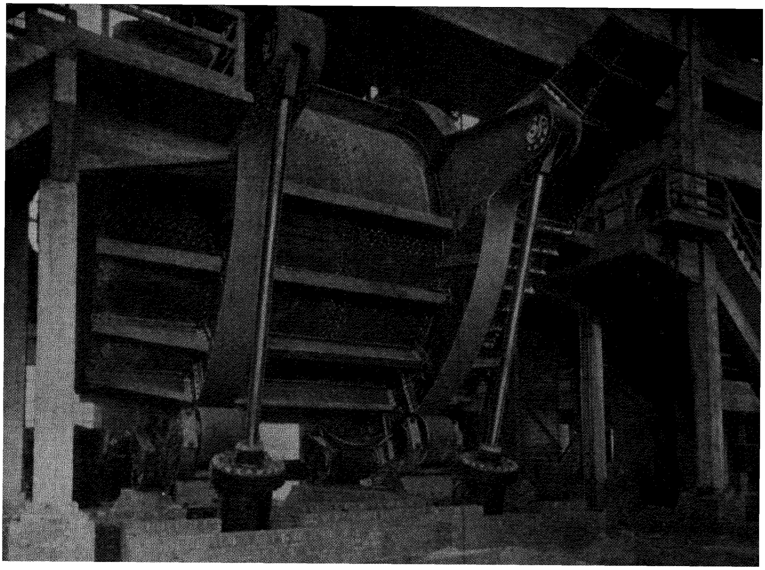


FIG. 102. A 700-ton inactive mixer.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.)

The capacity of mixers ranges from 250 to 1,000 tons, but active mixers do not generally exceed 400 tons capacity. The 400 ton active mixer, described by Geary and others,[9] has the following dimensions:

	ft.	in.
Length from block to block	43	0
Width inside lining	16	6
Depth of bath in centre	5	8
Air regenerators—length	19	2
width	8	0
depth	12	9
Gas regenerators—length	19	2
width	6	0
depth	12	9

This mixer is fired with producer- and coke-oven gas. The producer-gas is supplied by a producer which is isolated to the mixer, and the coke-oven gas is added at the mouth of the gas port by means of a water-cooled tube passing down the port.

Blast-furnace or blast-furnace and coke-oven gas are frequently used as the fuel.

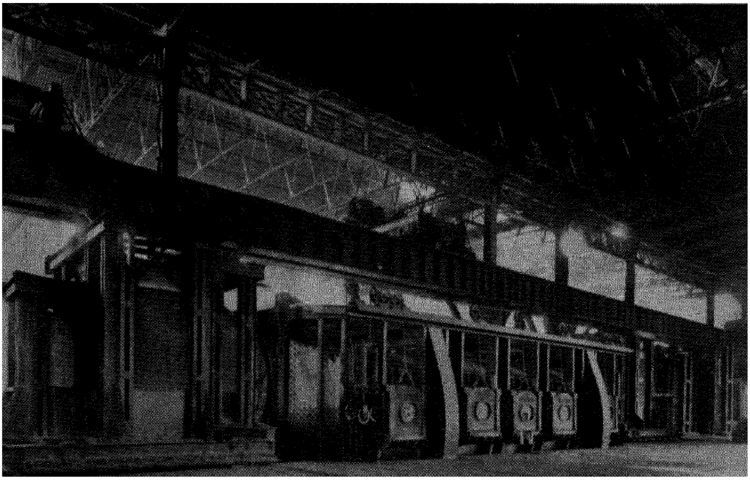


FIG. 103. A 400-ton active mixer.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.)

Refractory Materials

The refractory materials used in modern open-hearth construction will be discussed, with their properties, in Chapter 16. For the sake of completion, however, the various refractory materials used in different parts of an open-hearth furnace may be summarized as follows:

Slag pocket and regenerator linings ..	High grade firebricks.
Checkerwork	Semi-silica or firebricks.
Uptakes, blocks and roof	Generally silica bricks.

On many recently built basic furnaces, chrome-magnesite bricks have been employed in the furnace block. In a few cases, these basic bricks are used for roof construction.

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CHAPTER 7

THE ACID OPEN-HEARTH PROCESS

Preparation of the Hearth

Normally the pan forming the hearth is lined with silica bricks. This lining is generally 9 inches thick over the whole of the bottom and about 18 inches thick around the sides at slag level. An opening about 20 in. square, which is arched over with silica bricks, is left for the tap-hole. This brickwork is usually stepped around the sides to give some shape to the hearth.

Before preparing the working hearth, this brickwork is thoroughly dried. The working hearth or bottom may be prepared either by ramming a mixture of ganister and fireclay into position when cold, or by fusing into position a mixture of sand when the furnace is at top heat. The latter method appears to give by far the best results.

The tap-hole is always prepared by ramming a mixture of ganister and fireclay around a tapered plug, shaped like the desired tap-hole. When the tap-hole has been rammed solid, the plug can be withdrawn and the hole closed by inserting a piece of anthracite from inside the furnace, and backing up with a lump of fireclay from outside. Normally, according to Lester,[1] a mixture containing ten parts of ganister to one part of fireclay is used for the tap-hole.

When the bottom is rammed into position, a mixture of six parts of ganister and one part of fireclay is employed. The bottom is rammed to a minimum thickness of 9 in. and is gently sloped from all angles towards the tap-hole. The banks are rammed by means of formers or templates, so as to give a minimum thickness of four to six inches at the slag line. These banks are then sloped inwards.

Typical analyses of the ganister and fireclay used are given in Table XX.

TABLE XX[1]

	SiO_2	Al_2O_3	Oxide of Fe	CaO	MgO	Alkalis
Ganister .	95%	1·26	1·0 max	tr.	tr.	tr.
Fireclay .	66%	30·25	2·8	0·70	0·40	0·30

The fireclay acts as a flux, binding the more refractory ganister into a solid mass. After being thoroughly dried the furnace can be gassed.

When fusing in the bottom the brickwork is thoroughly dried, the ashes from the fires removed, and the tap-hole rammed before the furnace is gassed. At some works about half an inch of sand is spread evenly over the bottom and around the sides before gassing. This layer reduces the tendency for the silica bricks to spall. After the furnace has been brought up to a good working temperature, the fusing or burning in of the bottom can be commenced.

The sand mixture is then added in thin layers at a time, care being taken to assure that one layer is thoroughly fused before the next one is added. Satisfactory fritting of the bottom can be tested by means of a rabble. When a rabble is given a downward blow on to the bottom a little of the fused sand should adhere to it. An enduring bottom depends on good fusion and consolidation. It is imperative to give ample time for each layer to burn in, or continual trouble will result later.

Different grades of sand are used at different works, and typical analyses are given in Table XXI:

TABLE XXI

ANALYSES OF ACID-HEARTH SANDS

Type	SiO_2	Fe_2O_3	Al_2O_3	CaO	MgO	Alkalis	Authority
Sea Sand .	85·77	0·49	2·30	4·53	0·91	0·60	Lester[1]
Loam .. .	85·06	3·10	5·47	1·90	0·96	0·30	Lester[1]
Belgian Sand ..	98·40	0·44	0·52	0·11	- -	0·34	Ferguson[2]
Brown Sand	79·60	2·70	11·60	1·90	1·00		Ferguson[2]

The proportions of the various sands used vary according to the position in which they are employed. According to Lester[1] the first two or three layers may be ten parts of a second grade sand mixed with one part of loam. This is advisable owing to the almost-impossibility of getting sufficiently high temperatures to fuse a more refractory sand on the bottom layers and in the corners of the hearth. After about two inches of this mixture has been fused a more refractory sand is employed. The refractoriness is gradually increased as the height of the bottom proceeds, and is largely determined by the rabble test. If the bottom is too dry, extra loam is added. If the bottom becomes too sticky, the percentage of loam is decreased. Before adding the final layer the tap-hole is opened by means of a tap-hole drill (Fig. 104). This allows the necessary height of the bottom to be more accurately gauged.

The hearth described by Ferguson,[2] which gave an excellent life, was built up of Belgian and brown sand of the analysis shown in Table XXI. The ratio of brown to white sand varied from 1 to 4 in the lowest

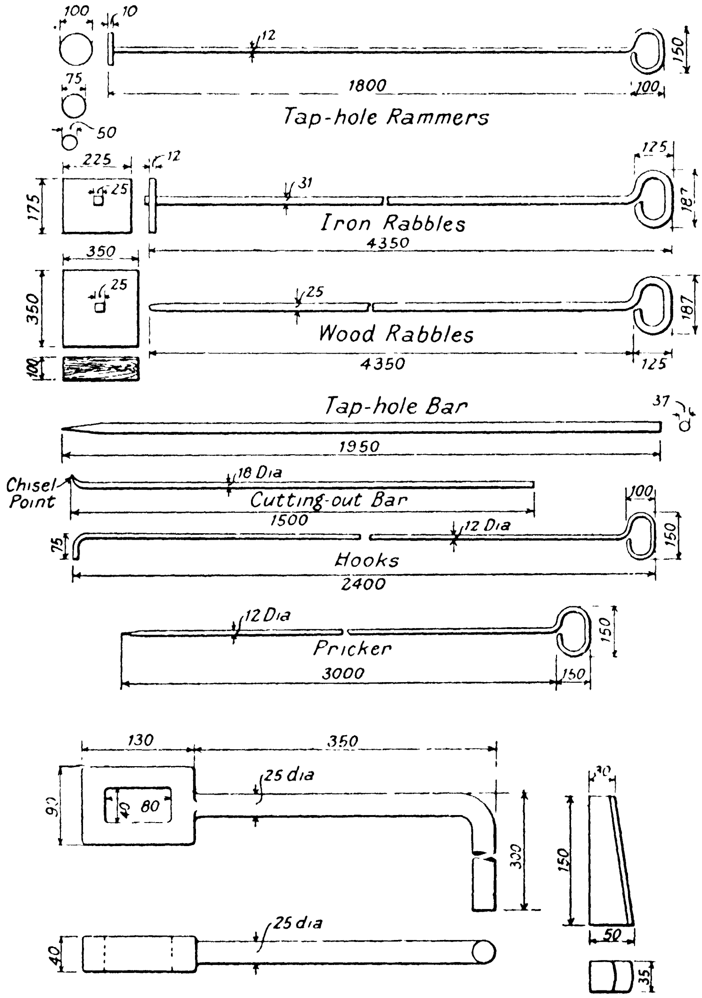

FIG. 104. Steelmaking tools.

(From "Practical Steelmaking," by W. Lister, by courtesy of the author

layer to a ratio of 1 to 7 in the final layer. The average composition of the sand used was 94 85 per cent. silica, 0·86 per cent. ferric oxide, 2·6 per cent. alumina, 0·44 per cent. lime, 0·19 per cent. magnesia and

0·28 per cent. alkalis. Any attempt to use a sand which corresponded to this average composition, however, proved unsatisfactory. It would appear that the brown sand forms a network around the silica grains of the silver sand. For the fusing of this 60-ton furnace hearth, 24¼ tons of sand were used, 19¾ tons of Belgian and 4½ tons of brown sand.

The durability of a bottom depends on thorough burning in and on the chemical purity and physical state of the silica grains in the silver sand. According to Ferguson,[2] the white sand should consist of clear-coloured, uniformly sized, rounded crystals. It should not contain angular irregularly shaped masses or fines. These will not build up into uniform layers on the furnace banks. He[2] recommends that the mechanical analysis should approximate:

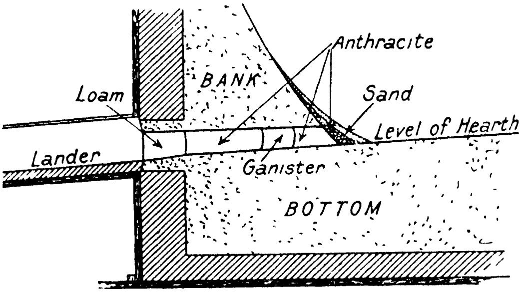

FIG. 105 Section through an acid open-hearth furnace.
(*From "Practical Steelmaking", by W. Lister, by courtesy of the author.*)

	Per cent.
Retained on—40-mesh sieve	4
75 ,, ,, ...	87
120 ,, ,, ...	8
160 ,, ,, ...	1
	100

Before charging, this bottom is consolidated by melting a quantity of acid slag and splashing this molten slag around the bath and banks. Any surplus slag will be tapped off through the tap-hole, but most of it will be absorbed.

One very important feature is the shape of the hearth. It must slope from all angles towards the tap-hole, so that a clear run off of metal and slag is obtained. A badly shaped bottom will result in loss of metallic yield due to metal being left in the furnace. If large quantities of slag and metal are left behind after tapping, excessive erosion of the hearth is likely to occur.

A typical section through an acid hearth is shown in Fig. 105.

Bottom Maintenance

The destructive agents, which attack an acid bottom, are the oxides of iron. Ferguson[2] states that the oxides of manganese are not destructive agents. The oxide of iron network required to cement the silica crystals gradually corrodes and ultimately fluxes away these primary grains, thus entailing constant renewal of the working face. Two different methods of hearth maintenance are practised.

Some operatives advocate the "dry off" method. Once or twice a week metal pockets are rabbled out and dried up with silver sand. When this drying up is completed, the gas is taken off and any metal-sand debris is removed through the furnace doors.

The other method consists of rabbling all metal pockets clear after each heat and fettling them with silver sand, which is thoroughly burnt in before recharging. The gas is kept on the furnace during this fettling and any metal-sand debris is left in the furnace. It is claimed that this method saves time and providing a good shape is maintained and the fettling is properly performed, good results are obtained.

The amount of sand used in the maintenance of the hearth varies widely at different plants. It may range from 40 to 90 lb. per ton, but 45 to 50 lb. per ton of ingots produced may be regarded as good average practice.

The method of closing an acid tap-hole after each heat is shown in Fig. 105.

Selection of Raw Materials

When selecting raw materials for the acid open-hearth process, the fundamental factor to be considered is the fact that sulphur and phosphorus cannot be eliminated. It is, therefore, essential that these elements are kept as low as possible. The chief raw materials employed are pig iron, steel scrap and iron ore. In addition, mill scale and limestone are used at many works.

For the selection of acid open-hearth pig iron, the following considerations must be taken into account:

Carbon. Not only should the total carbon be considered, but also whether it exists as graphitic or combined carbon. During the melting down period only graphitic carbon is oxidized. Combined carbon is not attacked in the presence of silicon and manganese. In controlling the carbon percentage at the "melt out" stage, it is, therefore, essential to standardize the combined carbon, which varies considerably in different brands of pig iron. Normally this element will vary from 3 per cent. to 4·2 per cent.

Silicon. Since silicon is not required to produce heat as in the Acid Bessemer Process, there is no object in having this element too high. On the other hand, there must be sufficient silicon present to give the slag volume required and prevent the oxidation of too much iron. A serious deficiency of silicon will result in the hearth being attacked.

The actual percentage desired varies at different works and by various operators. Lester[1] advocates 2·5 per cent. silicon. In the irons quoted by Ferguson[2] it ranges from 1·2 to 2·7 per cent., the average being 2·4 per cent.

Sulphur and Phosphorus. Since these elements cannot be eliminated, they must be kept as low as possible. The phosphorus should certainly not exceed 0·050 per cent., whilst the sulphur should be somewhat lower, say 0·040 per cent. maximum. This lower sulphur will counteract the tendency for the bath to absorb sulphur from the gas.

Manganese. Regarding the manganese content, there are two schools of thought. Lester[1] states that manganese should be kept as low as possible, owing to the fact that manganese oxide attacks the hearth. Ferguson,[2] however, suggests that manganese oxides should not be regarded as destructive agents, which attack the silica hearth.

Manganese retards the oxidation of combined carbon and thereby prolongs refining. On the other hand a reasonable percentage of manganese prevents over-oxidation and leads to the production of a higher quality of steel.

There appears to be abundant evidence in practice that a high residual manganese towards the end of refining improves the transverse tests and reduces the non-metallic inclusions. Some operators recommend a definite silicon-manganese ratio. A pig iron containing 1·0/1·5 per cent. silicon and 2·5/3·0 per cent. manganese has been found to give excellent results. In fact, at some works, charges are mixed synthetically to give a Si/Mn ratio of 1·0 : 1·5.

In spite of this desire for high manganese pig iron by many steel makers, large quantities of very high grade acid steels are produced from low manganese irons. Pig iron with manganese contents as low as 0·5 per cent. have been successfully employed in the production of high quality steels. The question of high and low manganese will be referred to again when discussing the reactions taking place during the process.

Steel scrap for the acid process must be low in sulphur and phosphorus. An upper limit of 0·04 per cent. is desirable. This scrap should also be free from rust. Not only does this oxide of iron give rise to irregular carbon melts, but it is frequently a source of hydrogen. Dawtry and others[3] regard "wet" rust as a dangerous source of hydrogen, which may play a role in the formation of hair-line cracks. Although unanimity on this point does not exist, it is highly desirable that all sources of hydrogen should be avoided.

The iron ores and limestone should be as free as possible from sulphur and phosphorus. Hydrate ores are not recommended owing to their hydrogen content.

The Process

The process may be conveniently divided into five stages:

Charging.
Melting.
The Melt Out.
The Boil.
The Finishing Operations.

Concerning the *method of charging*, there are two schools of thought. The American practice, which is followed at a few British works, is to charge all the scrap first and to finish with pig iron. It is claimed that this method enables the pig iron to be partially refined as it melts and drips over the practically molten steel scrap.

It appears logical, however, in view of chemical considerations to avoid scrap, especially light scrap, being placed on the hearth. Pieces of light scrap are prone to become oxidized and this oxide of iron attacks the silica of the hearth, sometimes causing serious erosion. The normal British practice is to charge the pig iron on the bottom and to finish the charging with scrap.

After the pig iron is charged, it is heated up before any scrap is charged. The scrap is then charged in stages. Many operators advocate the raw materials charged being brought to a condition of incipient fusion before the next lot is put into the furnace. Failure to observe this precaution frequently results in the charge melting out "cold", which leads to trouble later.

During *melting*, the maximum heat should be carried to assure a satisfactory melting temperature. As pointed out by Dawtry and others,[3] temperature control is of fundamental importance. If there is a sufficient reserve of heat at the "melt out", the subsequent refining should present little difficulty. Considerable skill, however, is required to correct the effect of cold melting. On the other hand, if the temperature becomes excessive, oxidation of the impurities will be retarded and the refining prolonged. The difference between "hot" and "cold" melting probably represents a very narrow temperature range. Dawtry and Co.[3] suggest that 1,520 deg. C. and under should be regarded as "cold" melting temperatures, whilst 1,570 deg. C. and over can be considered "hot" melting. The range preferred is 1,520 to 1,570 deg. C.

During the melting stage, much, if not most, of the graphitic carbon is oxidized. Some of the silicon and manganese is also removed by oxidation, and forms a slag with the iron oxide formed. These early slags will be comparatively low in silica averaging 40 to 45 per cent. The iron oxide content will depend on the amount of manganese in the charge. When using low manganese pig iron, these slags may contain 40 to 48 per cent. FeO. With high manganese pig iron, FeO is replaced by manganese oxide. Ferguson[2] gave two typical examples. With low manganese the slag showed 40·4 per cent. SiO_2, 46·5 per cent. FeO, 3·7 per cent. Fe_2O_3 and 6·6 per cent. MnO. The high manganese charge gave a slag containing 42·8 per cent. SiO_2, 26·8 per cent. FeO, 6·9 per cent. Fe_2O_3 and 17·4 per cent. MnO. The oxidation of combined carbon is prevented by the silicon and manganese present.

In order to ensure a satisfactory carbon "boil", the *melt out* carbon should exceed the "going on" or tapping carbon by 0·7 to 0·8 per cent.

When the charge is completely melted it is usually allowed to collect additional heat before ore feeding is commenced. Whether a charge is completely melted can be ascertained by pushing a rabble through the bath, when any unmelted material is encountered. At this stage the bath will normally be inactive. When working very low manganese charges, however, the silicon content of the bath may be low and there will be a strong tendency for the bath to boil vigorously. The presence of manganese retards the removal of silicon and thereby delays the carbon boil, which will not commence until the silicon is under 0·20 per cent.

At the "melt out" the slag should contain 48 to 52 per cent. SiO_2, whilst the FeO and MnO contents will depend on whether high or low manganese pig iron is charged. With high manganese pig iron the FeO may range from 20 to 25 per cent., with 15 to 20 per cent. MnO. Low manganese pig iron will give a slag on the high SiO_2 range, and the iron oxide may reach 30 to 35 per cent. Examples of high and low manganese charges at the "melt out", quoted by Ferguson,[2] are shown in Table XXII.

TABLE XXII

TYPICAL COMPOSITIONS AT THE "MELT OUT" STAGE[2]

	METAL			SLAG		
	C	Si	Mn	SiO_2	FeO	Mn
Low Manganese .	1·30	0·04	0·06	52·0	33·0	8·0
High Manganese . .	1·60	0·35	0·44	51·0	20·0	19·0

As soon as the bath has gathered sufficient heat, ore feeds commence. These feeds oxidize the silicon and manganese according to the reactions:

$$Mn + FeO \rightleftharpoons MnO + Fe$$
$$Si + 2FeO \rightleftharpoons 2Fe + SiO_2$$

These reactions are reversible and are readily affected by temperature. The oxidations of silicon and manganese are exothermic, the greater evolution of heat occurring in the case of silicon. Therefore, a rising temperature in the presence of a high manganese content tends to retard the oxidation of silicon and in extreme cases may result in the reduction of this element from the slag, thus:

$$SiO_2 + 2Mn \rightleftharpoons 2MnO + Si$$

This factor indicates the importance of temperature control.

The progress of a charge and the temperature prevailing can be measured by five methods:

(1) Direct pyrometric determinations.

(2) Regular analytical determinations of the FeO content of the slag.

(3) The rod test, which consists of dipping a mild steel rod into the bath and noting its appearance on withdrawal after a given time.

(4) The taking of small mussel-shaped samples. A small mussel-shaped sample spoon is dipped into the bath and a sample of the liquid metal withdrawn. Some of this metal is poured away and the bared surface is then covered with a thin coating of slag. It is withdrawn from the furnace and allowed to cool. When cold, the slag is knocked off and the exterior of the bead or

sample is observed. According to Dawtry[3] "good bath condition" is indicated if this sample shows a perfectly smooth surface, fully bellied on the underside and slightly so on the upper face.

(5) Viscosimeter test. The viscosimeter developed by Herty and his co-workers[4] has been successfully applied to the study of acid open-hearth slags and to the control of the process at many works. This viscosimeter, Fig. 106, consists of a split low-carbon steel mould, provided with a conical shaped well, into which the molten slag is poured, and a ¼-in. internal dia. tube, 10 to 12 in. long, along which the slag entering the well may flow. The viscosity of the slag is recorded as the distance in inches to which the slag flows along this tube before it solidifies.

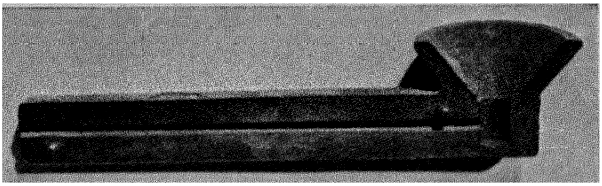

FIG. 106. Herty Viscosimeter.
(By courtesy of the Iron and Steel Institute.)

The viscosity of the slag measured in this way is a good indication of the SiO_2 content, and used with experience and intelligence is a useful guide in controlling the working of the charge.

As shown by Kramarov,[5] the inches of flow increase as the SiO_2 content of the slag decreases, and he gives the following examples:

SiO_2 Content of Slag	*Inches of Flow*
60 per cent.	1·57
57 ,,	3·00
54 ,,	3·93
50 ,,	4·72

As the ore feeds proceed, the silicon will be oxidized and the silicon content of the slag will increase. When the silicon has been reduced below 0·15 per cent. a vigorous *carbon boil* will commence. The oxidation of carbon is endothermic and therefore the maintenance of a vigorous boil demands a high temperature. If the furnace is over-fed with iron ore, the absorption of heat may be excessive and the vigour of the boil reduced. In extreme cases of over-feeding, the carbon may be unable to reduce the FeO, and the slag becomes thin, attacking the hearth. On the other hand, under-feeding results in a decrease in the FeO content of the slag, which becomes lighter in colour and viscous, and the reactions between metal and slag are retarded.

The maintenance of a vigorous boil is essential to satisfactory heat transfer and the elimination of non-metallics. The agitation of the bath by the carbon boil plays an important part in transferring heat through the bath. Houston[6] also suggests that a free boil is associated with the removal of gases like hydrogen and nitrogen present in the raw materials or absorbed during the melting period.

As the carbon boil progresses, the slag may thicken from time to time, due to the reduction of the FeO content by the carbon reaction:

$$C + FeO \rightleftharpoons Fe + CO$$

This slag may be thinned by limestone additions instead of ore feeds. The lime replaces FeO in its combination with SiO_2, thus:

$$FeO\ SiO_2 + CaO \rightarrow CaO\ SiO_2 + FeO$$
$$FeO + C \rightleftharpoons Fe + CO$$

In this way limestone assists the carbon boil by liberating FeO from the slag for the work of oxidation. This also results in a lower total FeO content in the slag and a lower state of oxidation of the metallic bath. The progress of a typical acid heat, taken from the work of Dawtry and others,[3] is given in Table XXIII. The author has inserted approximate temperatures at the various stages. At some works, however, limestone is not employed. The carbon boil is maintained by judicious feeds of iron ore.

Houston[5] recommends that lumpy ore, 4 to 6 in. in size, should be used to initiate the boil. This type of ore does not dissolve in the slag, but makes direct contact with the metal. Fine ore, which first dissolves in the slag, can be used when the charge is boiling. The use of fine ore may permit a more delicate adjustment of slag conditions towards the end of the boil.

When the carbon has been reduced to practically the desired limits, the *finishing operations* begin. Two different opinions are held regarding the best practice to be adopted during the final stages of the boil, prior to the charge being tapped. At many plants, the final stages of the process are controlled so as to give a passive condition, whilst other

TABLE XXIII

CHANGES IN METAL AND SLAG COMPOSITION

	When completely melted	After 1 hr.	After 2 hrs.	After 3 hrs	After 4 hrs	After 5 hrs.	After 6 hrs before adding finishings
Approx. Temperature, degrees C.	1560	1600	1620	1630	1630	1630	1640
				METAL			
Carbon, per cent.	1·30	1·18	1·04	0·70	0·48	0·32	0·26
Silicon, per cent.	0·55	0·48	0·08	0·06	0·07	0·07	0·07
Manganese, per cent	0·24	0·20	0·06	0·08	0·09	0·08	0·07
Oxygen, per cent. (Herty Method)	0·004	0·004	0·008	0·012	0·015	0·016	0·018
				SLAG			
SiO_2, per cent.	48·00	48·00	54·0	58·0	58·0	59·5	59·5
FeO, per cent.	27·00	27·00	18·0	12·0	11·0	11·2	11·0
MnO, per cent.	15·00	14·50	12·0	11·5	11·0	11·0	10·5
CaO, per cent.	0·50	0·50	6·0	12·0	13·0	13·5	13·8

operators advocate finishing with a fluid slag and de-oxidizing the bath with silico-manganese or similar de-oxidizers.

The first method, and one frequently advocated in this country, consists of regulating the bath conditions so that the metal and slag are passive at the end of the process. Oxide and limestone feeds are reduced and carbon attacks the FeO content of the slag, which is thereby reduced. The extent to which the FeO content of the slag is reduced, depends on the temperature and viscosity of the slag. The reduction of the FeO, accompanied by an increase in the SiO_2, causes the slag to become more viscous. It is obvious, therefore, that the higher the temperature prevailing, the lower the FeO content may become. Towards the end of this stage, silicon is actually reduced from the slag by the action of carbon and iron, thus:

$$SiO_2 + 2C \rightleftharpoons Si + 2CO$$

This reduction of FeO in the slag is accompanied by a reduction in the FeO dissolved in the metal, providing the slag is sufficiently fluid to allow this transfer. According to McCance,[7] the relationship of FeO in the metal and slag may be expressed:

$$\frac{\text{FeO in metal}}{\text{FeO in slag}} = L_{FeO}$$

Where L_{FeO} is a constant depending for all practical purposes on temperature only.

This method possesses the advantages of reducing the amount of de-oxidizers used, and the de-oxidation products arising therefrom; and gives complete control of the final bath conditions, enabling the increased accuracy in meeting the specifications required. It does, however, prolong the refining time, thereby reducing the rate of output and increasing the period of contact between hearth and metal, frequently resulting in greater erosion of the hearth and more silicate inclusions in the steel from the hearth.

The reduced activity of the bath and the viscous slags employed, hinder the removal of non-metallic inclusions in the steel. According to Stokes' Law, a small sphere falling under the action of gravity through a viscous medium reaches a constant velocity equal to:

$$V = \frac{2gr^2 (d_1 - d_2)}{9n}$$

where v = velocity in cm./sec.
g = 981 cm./sec./sec.
r = radius of the sphere or particle in cm.
d_1 = density of the liquid gm./c.c.
d_2 = density of sphere gm./c.c.
n = viscosity of the liquid (poise).

It therefore follows that the more viscous the slag, the greater will be the likelihood of inclusion being trapped in the metal.

The viscous slag may also suppress reactions which have not reached equilibrium, and these reactions may resume in the ladle, or even the ingot mould, with unsatisfactory results. The passive condition of the bath, as pointed out by Ferguson[2], does not necessarily denote true equilibria.

Herty and Jacobs[8] recommend the employment of fluid slags and suitable de-oxidizers. The more fluid slag and the greater activity of the bath, in conformance with Stokes' Law, favours the removal of non-metallic inclusions. These conditions also remove the possibility of suppressed reactions, which are not in true equilibrium. The more active bath conditions tend to increase the rate of production and reduce the time of hearth/metal contact. This method, however, results in a greater amount of dissolved FeO in the metal; it requires an increased quantity of de-oxidizers to remove this FeO and introduces the problem of removing the resultant products of de-oxidation.

In order to assist the removal of non-metallic inclusions, Herty[9] recommends the use of silico-manganese alloys. He shows that the use of silico-manganese alloys gives a much cleaner steel than when ferro-manganese and ferro-silicon are used separately. If the best results are to be obtained, the Mn/Si ratio must be between 3 : 1 and 5 : 1. The manganese present gives rise to MnO, which fluxes the more refractory silica or silicates. It has also been shown that the silicate particles formed by de-oxidation with this double alloy are larger in size, and therefore in accordance with Stokes' Law are more easily removed. The addition of silico-manganese, or other de-oxidizers, arrests the reduction of carbon.

Large tonnages of high grade steels are produced annually by both these methods, but there appear many reasons to apply a middle course. The author favours control of the bath towards the end of the refining so as to reduce the FeO content of the slag, without impairing the fluidity of the slag to such an extent as to arrest the removal of non-metallic inclusion and to suppress reactions which have not attained true equilibrium. The final de-oxidation can then be achieved by the use of silico-manganese. The modification of the methods of both schools of thought tends to preserve most of the advantages of each practice and minimize the disadvantages.

The control of manganese throughout the charge assists the reduction of the amount of dissolved FeO in the metal at the final stages. As shown by Maurer and Bischof[10] the dissolved FeO is controlled by the reaction:

$$\mathrm{FeO} + \mathrm{Mn} \rightleftharpoons \mathrm{MnO} + \mathrm{Fe}$$

and the velocity of this reaction depends on the formula:

$$K = \frac{(\mathrm{FeO})\ \mathrm{Mn}}{(\mathrm{MnO})} = \frac{(\mathrm{Fe})\ \mathrm{Mn}}{(\mathrm{Mn})}$$

The (FeO) and (MnO) refer to the concentration in the slag, whilst the Mn represents the manganese in the metal. They demonstrate that the MnO/FeO ratio increases with the percentage of manganese.

When high manganese pig iron is employed, MnO replaces FeO in the slag, resulting in a lower dissolved FeO content in the metal. Kalling and Rudberg[11] discuss similar advantages gained by the use of manganese ore instead of iron ore. They give an example of as much as 8 to 10 per cent. of MnO being built up in the slag by the use of manganese ore. This results in a lower FeO content and a considerable economy of alloys.

In the practice described by Houston,[6] the charge consisted of 50 per cent. pig iron and 50 per cent. special scrap, and was balanced so as to give a silicon content of 1·1 to 1·3 per cent. and a manganese content of 0·95 to 1·05 per cent. Any deficiency of silicon or manganese is adjusted by ferro-silicon addition during charging, or ferro-manganese addition just before the charge is melted level.

Some authorities advocate "*pigging back*" before tapping. Houston[6] quotes an incident where the "pigging back" of a large furnace to the extent of 20 points of carbon gave very beneficial results. Pig iron and finishings amounting to 5 per cent. of the charged weight were dropped straight into the bath. On the other hand, the application of a similar practice on a smaller furnace produced very unsatisfactory results. It is suggested in this case that the chilling effect of the cold additions was responsible for the poor results. Maybe such additions should be heated before being added to the furnace.

By correct control of the bath conditions, including the temperature and slag viscosity, silicon and manganese can be reduced from the slag, thereby rendering alloy additions unnecessary. The physico-chemical considerations underlying this practice—commonly referred to as the Swedish process—are thoroughly discussed by McCance.[7]

The fundamental considerations in obtaining correct conditions for tapping the charge, appear to be:

(1) The lowest possible FeO content in the slag, consistent with reasonable fluidity.

(2) True equilibrium between metal and slag, thus avoiding suppressed reactions being activated in the ladle or ingot moulds.

(3) The avoidance of non-metallic inclusions, either by attention to slag viscosity and bath activity, or by the use of suitable de-oxidizers.

In the acid process the *alloy additions* necessary to bring the metal to the required specification are generally added to the bath. If the bath has been properly controlled there is no loss of the silicon addition, but some loss of manganese is normal. The manganese loss will depend on bath conditions and the alloy used. According to Houston,[6] when using ferro-manganese immediately after the last carbon result has been received, prior to tapping, a loss of 20 per cent. can be expected. If silico-manganese is added to the bath at an earlier stage, the loss of manganese may approximate 30 to 35 per cent. The time factor is important.

The carbon content of the steel may be adjusted either by spiegel and/or pig iron additions to the bath, or by anthracite coal addition to the ladle. If anthracite coal is used, 14 lb. of anthracite will raise 40 tons of steel 0·01 per cent. in carbon. Some carbon of course is introduced in the ferro-manganese, and some other alloy additions, and this increment must be taken into account.

For example, on a 50-ton heat, if the specification requires C 0·42/0·47, Si 0·25/0·30 and Mn 0·70/0·80, whilst the final bath analysis is C 0·40, Si 0·05 and Mn 0·08, the additions necessary would be:

	C	Si	Mn
Specification . .	0·45	0·275	0·75
Bath analysis . .	0·40	0·050	0·08
To be added	0·05	0·225	0·67

Silicon

$$\frac{0{\cdot}225 \times 1000}{75} = 3 \text{ cwt.}$$

That is, 3 cwt. of 75 per cent. ferro-silicon to the bath, assuming no loss of silicon.

Manganese

When using 80 per cent. ferro-manganese containing 6 per cent. carbon, and assuming 25 per cent. loss of manganese:

$$\frac{0{\cdot}67 \times 1000}{60} = 11{\cdot}2 \text{ cwt. to the bath.}$$

Carbon

The ferro-manganese addition will give:

$$\frac{11{\cdot}2 \times 6{\cdot}0}{1000} = 0{\cdot}067 \text{ per cent. carbon.}$$

Therefore the final position will be:

	C	Si	Mn
Bath analysis	0·40	0·05	0·08
3 cwt. 75 per cent. Fe Sil to bath	—	0·225	—
11¼ cwt. 80 per cent. Fe Mn to bath	0·067	—	0·67
Finish analysis should be	0·467	0·275	0·75

When these additions to the bath have had time to melt and become uniformly distributed, the furnace can be tapped. In the case of a fixed furnace the tap-hole is opened by driving a pointed bar through the tap-hole. When this bar has penetrated into the furnace, it is knocked back and the stream of metal follows the bar and runs down the short launder into the ladle. Just before the furnace is tapped, a bath sample can be analysed for carbon and manganese. Any adjustment can then be made by addition of anthracite coal and/or ferro-manganese to the ladle.

In the production of *alloy steels* various alloys are employed. For nickel steels, metallic nickel is usually charged into the furnace and no loss occurs. Nickel may also be added by the use of nickel scrap or nickel-ferrous ore. Vanadium is normally added in the form of ferro-vanadium containing 32 to 42 per cent. vanadium, 3 to 5 per cent. silicon and 0·5 to 3·0 per cent. carbon. This alloy is generally added in the ladle, and should be kept away from any slag. A 10 per cent. loss of vanadium takes place.

In the manufacture of molybdenum steels, either calcium molybdate (40 per cent. Mo and 25 per cent. CaO) or ferro molybdenum (50 to 60 per cent. Mo, 1·5 per cent. Si and 2·0 per cent. C) may be employed. No loss of molybdenum is experienced. It is generally added to the furnace shortly after the charge is melted.

Chromium is normally added in the form of ferro-chromium. There are two grades of ferro-chromium—high and low carbon. These approximate the following composition.

	Cr	C	Si
	Per cent.	*Per cent.*	*Per cent.*
High C ferro-chromium ..	60–75	4–8	2–3
Low C ferro-chromium ..	60–75	0·1–2	1 max.

The addition is made to the furnace after the steel is in condition for tapping, and a loss of 5 per cent. of chromium is usually encountered.

Korber and Oelsen[12] show on a laboratory scale that chromium can be reduced from the slag. According to their work, with 25 per cent. FeO in the slag, the partition of chromium between slag and metal is about 8 to 1. With 2·25 per cent. Cr in the slag, the residual chromium in the metal is approximately 0·28 per cent. A high MnO content in the slag favours the recovery of chromium. Fritterer[13] shows that chromium will be reduced from the slag at the same time as silicon. These points are of interest when employing chromium scrap with the object of alloy conservation.

A typical example of the working of an acid open-hearth charge is shown in Table XXIII. It is probable that ore feeds commenced an hour after melting, and these were followed by feeds of limestone. It will be noted that the carbon drop (boil) did not begin until the silicon was under 0·20 per cent., after which a steady drop is observed. After three hours refining the carbon drop continues, whilst the silicon remains practically constant. A slight return of manganese takes place during these stages.

An important feature of the acid process is the *control of the slag*. In Table XXIII it will be noted that the SiO_2 content gradually increases as the heat progresses. This increase in SiO_2 is accompanied by a thickening of the slag. The drop in the basic content (FeO, MnO and CaO) of the slag which accompanies this change in viscosity will be observed. When melted, the total bases are 42·5 per cent. and little change occurs until the carbon boil begins. Three hours after melting, the $MnO + FeO + CaO$ have decreased to 35·5 per cent. and the final slag analysis, before the alloy additions, is approximately the same figure. During this period, much of the FeO has been replaced by CaO.

The decrease in the FeO content of the slag is indicated by its appearance. The colour of the slag changes from black to brown, and as the FeO falls still further it becomes lighter. The actual tint of the final slag is affected by the MnO present. Houston[6] recommends that the refining slag should be glossy, brownish black on the surface with a clean greenish fracture, showing no signs of black flecks of oxide. He adds that it should become lighter as refining proceeds until it has a dark brown surface, with a paler green fracture. He suggests that the analysis of the finishing slag, prior to the finishing additions being made, should be within the range:

	Per cent.	
SiO_2 ..	58 to 60	Total Bases:
FeO ..	12 to 15	36 to 38 per cent.
MnO ..	12 to 16	
CaO ..	8 to 10	

It will be noted that the SiO_2 content of the slag given by Reinartz[14] in Table XXIV, which shows the history of another acid open-hearth heat, is slightly higher.

Discussing the constituents of acid open-hearth slags, White[15] shows that if little or no lime is used, 95 per cent. of the total constituents are FeO, MnO and SiO_2. Such slags approximate closely the simple ternary system, $FeO-MnO-SiO_2$. Finishing slags of this type usually contain 50 to 55 per cent. SiO_2, and are thereby practically saturated with SiO_2 at the temperature prevailing at that stage. As he[15] points out, acid slags can be considered as increasing continually in viscosity as the silica content increases, and the indications are that a condition of limiting viscosity will be reached at, or at least near to, the saturation point.

When lime is add up to 7·8 per cent. CaO, it enters the ordinary slag silicates as a constituent. When CaO exceeds this value, voglite appears.

The saturation concentration of SiO_2 in ternary slags consisting of FeO, MnO and SiO_2 at steelmaking temperatures, is about 50 per cent. Korber and Oelsen[16] show that this figure is not materially affected by variations in the FeO/MnO ratio. The presence of CaO, however,

TABLE XXIV

HISTORY OF AN ACID OPEN-HEARTH HEAT

	Slag						Metal		
	CaO	SiO_2	FeO	Al_2O_3	MnO	MgO	C	Mn	Si
	Per Cent	Per Cent	Per Cent	Per Cent	Per Cent	Per Cent.	Per Cent	Per Cent	Per Cent.
10 40 a.m. Melted .	3·40	51·90	28·86	1·82	12·22	1·65	1·34	0·73	0·275
10.55 a.m.–12 30 p m. added ore									
12.30 p m. added lime									
1. 0 p.m. .. .	5·30	58·70	19·40	2·08	12·50	1·72	—	—	—
1.30 p m ..	8·20	60·95	13·64	2·15	12·76	1·80	0·65	0·07	0·02
2. 0 p m.	—	—	—	—	—	—	0·56	0 07	0·02
Added Mn and Si ..	—	—	—	—	—	—	—	0·80	0·16
2.40 p.m.	9·10	61·16	12·43	2·25	13·00	1·85	—	—	—
Final Analysis	—	—	—	—	—	—	0·57	0·73	0·25

increases this value. At 1,600 deg. C. Korber[17] shows that the saturation value is progressively increased as the percentage of CaO is raised, thus:

CaO per cent.	Nil	12	18	34 per cent.
SiO_2 ,,	48/52	57	60	63 ,,

The constitution and properties of acid open-hearth slags are very ably reviewed by White[15] to whose work reference should be made for more detailed information.

Uses of Acid Open-Hearth Steels

Steels produced by the acid open-hearth process are eminently suitable for tyres, axles and high grade work. If correctly made they should be reasonably free from non-metallic inclusions and possess a low oxide and gas content. Since alloy additions can be made to the bath, the process is favoured for the production of alloy steels, where great uniformity is required.

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CHAPTER 8

THE BASIC OPEN-HEARTH PROCESS

Preparation of the Hearth

Normally, the pan forming the working hearth is lined with magnesite or dolomite bricks. Prior to the War, magnesite bricks were used exclusively, but the restricted supply of these, and the considerable development in the manufacture of stabilized dolomite bricks resulted in the adoption of dolomite on an extensive scale. There is, however, still considerable difference of opinion regarding the respective merits and demerits of each type of brick. In general it may be said that magnesite appears to be more reliable. Many operators who employ dolomite bricks for most of the hearth always use magnesite bricks in the more vulnerable area around the tap hole.

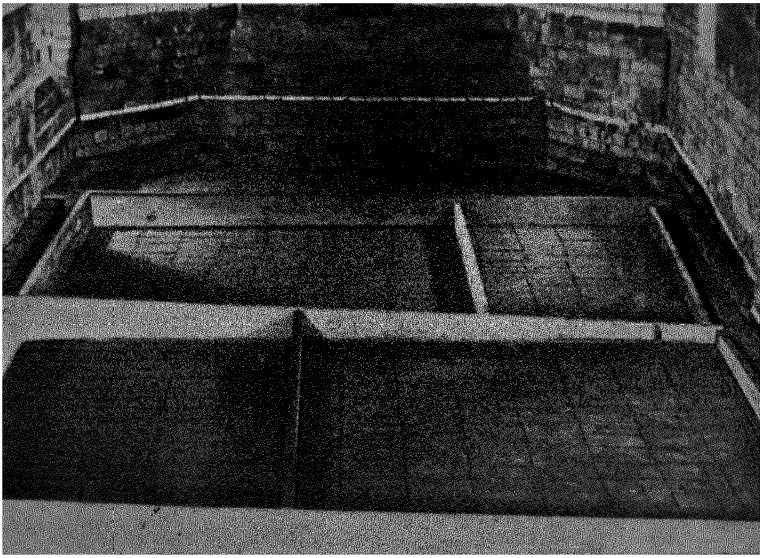

FIG. 107. Ramming of a basic open hearth.

The dolomite or magnesite brick lining is usually 12 to 14 inches thick, and in some cases a 3-inch layer of firebricks is placed between the pan and the first layer of the basic bricks. Firebricks are also frequently employed in the corners of the pan to assist the shaping of the lining, as shown in Fig. 107.

Before the working hearth can be superimposed on this brick lining, the brickwork must be thoroughly dried. During the drying process care must be taken to see that the expansion joints are covered with sheet metal or the like, to avoid ashes or dirt filling them, especially when coal fires are employed for drying. If insufficiently dried, the steam evolved when the furnace is gassed may cause the brickwork to lift very badly.

The brickwork may also lift as a result of insufficient allowance for expansion. The expansion of magnesite and dolomite bricks is approximately 0·35 in. per yard, therefore in a furnace 38 ft. long, 4½ in. must be allowed for expansion. This is accomplished by leaving 3 or 4 1 in. or 1½ in. spaces laterally. Similar expansion joints or spaces are allowed in the width of the furnace. The expansion joints in the various layers of brickwork should be staggered.

When the brickwork is thoroughly dried and any ashes removed, the working hearth may be prepared by:

(1) Fritting on (*a*) magnesite.
    (*b*) dolomite.
(2) Ramming with dolomite and tar.

Whichever method is used the *tap hole* is always prepared before the furnace is gassed. A wooden plug, having the shape of the desired tap hole, is inserted through the arch left in the brickwork and tarred dolomite is rammed thoroughly around this plug. It is most important that this tap hole is the right height and has a good slope away from the furnace. The procedure of preparing the tap hole is illustrated in Fig. 108.

Recently a dolomite cement called doloset has been introduced for the preparation of basic open-hearth furnace tap holes. This cement is mixed with water and is rammed around the former as indicated above. This material forms an excellent tap hole and this method of preparation is rapidly becoming a recognized practice. This particular application of doloset, together with other uses, is shown in Fig. 110.

In America a similar preparation made of magnesite is frequently employed. Its method of application is similar.

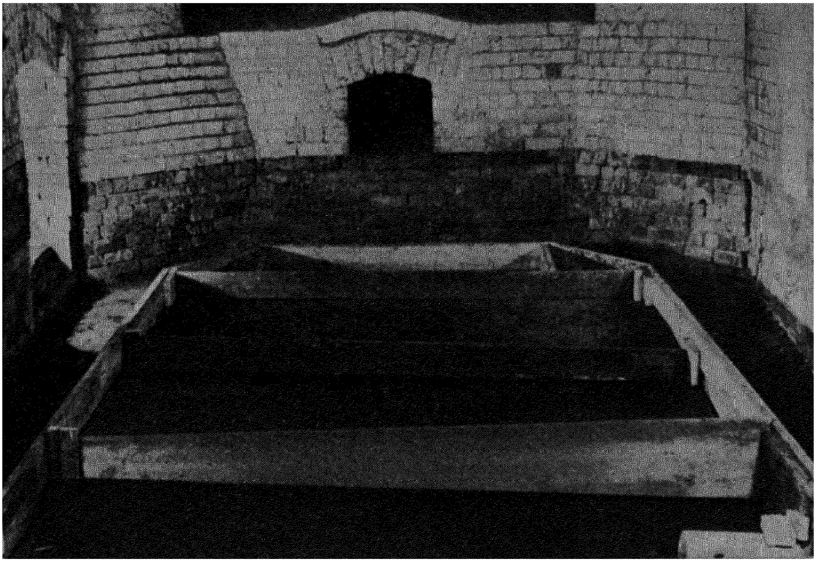

FIG. 108. Ramming a basic hearth.

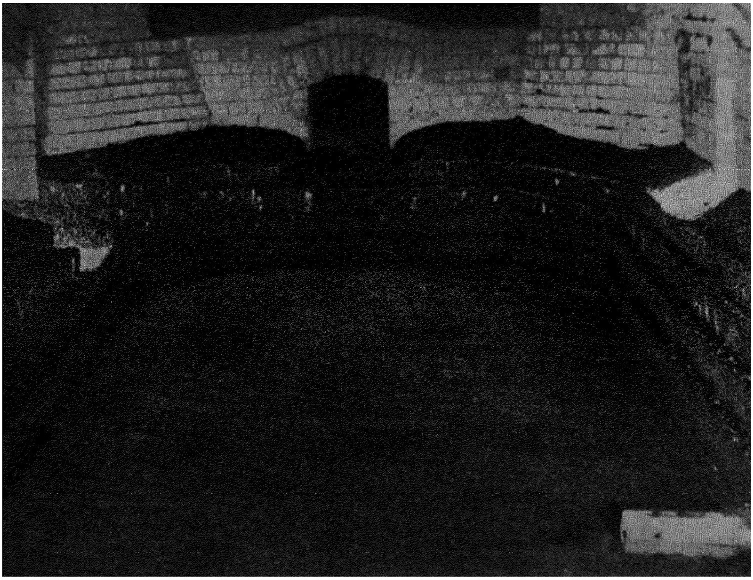

FIG. 109. Ramming a basic hearth.

When the hearth is *rammed into position* cold, tarred dolomite is employed, and this is rammed in stages around suitable formers. It is imperative that the dolomite should be rammed thoroughly, if a compact hearth is to be attained.

In recent years considerable work has been carried out on the question of grading dolomite, so as to obtain a maximum packing density. This subject is discussed in Chapter 3, page 35. The fundamental principles discussed there are equally applicable to basic open-hearth furnaces.

The normal procedure is to ram the bottom first, in layers of about three inches at a time. As this ramming proceeds the bottom is sloped towards the tap hole. When the requisite thickness, which is usually eight to ten inches in front of the tap hole, has been rammed into position, the tap hole is prepared as already described. The ramming of the ends, front and back wall then proceeds in stages. This practice is shown in Figs. 107 to 110.

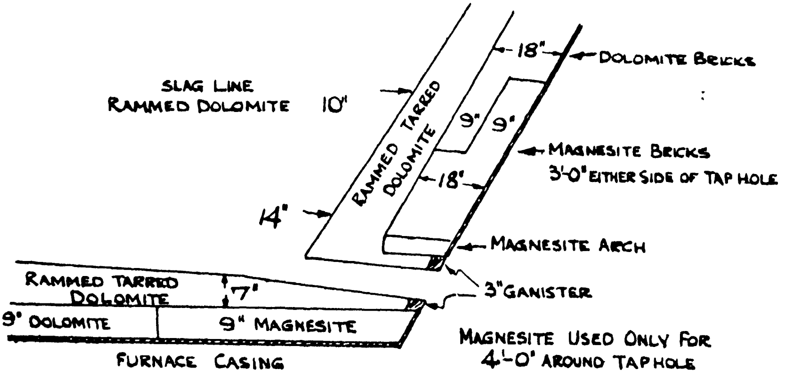

FIG. 110. Ramming a basic hearth—section through tap-hole.

When the ramming is completed, it should be dried, and the furnace gassed as described in Chapter 6, page 154.

After the furnace has been raised to a good working temperature, the corners are filled with further additions of dolomite, crushed to pea size, and the final shaping of the hearth completed. The furnace is then slagged. A quantity of basic slag, preferably fine material, is then thrown into the furnace and melted. When this slag is thoroughly melted it should be splashed around the banks. A considerable amount of the slag will be absorbed thereby filling the spaces between the dolomite grains tending to bind these grains together. The surplus molten slag is tapped off, whereby the tap hole is also consolidated. The furnace is now ready for charging.

If the hearth is *fritted into position* with either magnesite or dolomite, the tap hole is prepared as already described, prior to the furnace being gassed. When the furnace has been brought up to its maximum temperature, the fritting of the bottom may commence. This fritting of the hearth, however, requires a good temperature, if satisfactory results are to be obtained.

For dolomite hearths, graded dolomite mixed with a suitable flux is employed. The flux normally used is basic open-hearth slag, although some operators use mill-scale. The percentage of slag may vary up to 10 per cent., but will be regulated by the refractoriness of the dolomite and the temperature of the furnace.

The dolomite and slag should be fed into the furnace in layers of about half an inch thick. Each layer should be thoroughly fritted and tested before another feed is made.

The usual method of testing is to rub a rabble over the hearth after sufficient time has been allowed for the previous feed to frit. If the dolomite mixtures and the furnace temperature are correct, the hearth should be hard and slightly sticky, so that a small quantity of the fettling adheres to the edge of the rabble. If the bottom appears dry on the rabble the dolomite mixture is too refractory, and additional flux is necessary. If there are signs of excessive stickiness, it is indicative of too much flux. It will be found that the upper layers require less flux than the lower layers, owing to the greater penetration of the flame.

The same procedure is adopted when fritting the hearth with magnesite. Since magnesite is more refractory than dolomite, maximum heat must be carried during the burning-in process. Owing to the need of employing a high temperature, many operators cover the magnesite or dolomite bricks lining the pan, with one or two inches of tarred dolomite. The side walls are treated frequently in the same way. This precaution prevents excessive spalling of the brickwork.

Table XXV shows the history of the preparation of a magnesite hearth.

The amount of flux used when fritting in with magnesite depends on the magnesite employed. Some grades of magnesite are far more refractory than other grades, thereby requiring more flux. In recent years specially prepared magnesite, in which the necessary flux is incorporated, has been placed on the market. This aspect, however, is discussed in more detail in Chapter 16 dealing with Refractory Materials.

TABLE XXV

The Preparation of a Magnesite Hearth

The hearth-brick lining was covered with two inches of rammed tarred-dolomite to protect the brickwork from spalling.

The furnace was gassed at noon on September 26th, and commenced bottoming at 10 a.m. September 30th.

Progress of Bottoming

Date	Time	Additions (bags)		Remarks
		Magnesite	Slag	
Sept. 30	10.00 a.m.	3	1	
	3.30 p.m.	10	4	
	8.00 p.m.	10	4	
Oct. 1	1.00 a.m.	10	4	
	6.00 a.m.	10	4	
	9.00 a.m.	6	2	Mixed with tar for the banks
	10.10 a.m.	6	2	,, ,, ,,
	11.00 a.m.	6	2	,, ,, ,,
	2.00 p.m	6	2	
	6.00 p.m.	6	2	
Oct. 2	12.30 a.m.	6	2	
	7.30 a.m.	9	2	
	12.30 p.m.	10	2	
	4.00 p.m.	12	2	
	7.30 p.m	12	2	
	11.30 p.m.	12	2	
Oct. 3	3.00 a.m.	12	2	
	8 00 a.m.	6	1	
	10.00 a.m.	7	1	Shaping of the banks proceeding
	11.15 a.m.	7	1	
	2.30 p.m.	14	2	
	6.30 p.m.	14	2	
Oct. 4	1.00 a.m.	12	1½	
	7.30 a.m.	8	1	
	10.00 a.m.	—	4	Tap hole opened one hour after
	12.00 noon	8	1	
	4.30 p.m.	12	1½	
	9.30 p.m.	8	1	
Oct. 5	1.30 a.m.	8	1	
	5.30 a.m.	8	1	
	9.30 a.m.	8	0	Mixed with tar for shaping banks
	12.00 noon	4	1	Final dressing

Furnace was slagged three hours later and commenced to charge seven hours later.

NOTES.—One bag of magnesite weighed 157 lb. One bag of basic slag weighed 140 lb. The additions indicated were thoroughly mixed on the stage before being fed into the furnace. Top heat was carried throughout the fritting.

Analysis of the slag used as a flux

Silica	Lime	P_2O_5	Iron
9·6%	40·0%	8·8%	14·5%

Average analysis of cores taken between September 30th, and October 5th.

	SiO_2	CaO	P_2O_5	FeO	Fe_2O_3	MgO
Sept. 30th/Oct. 3rd	4·0%	—	1·97%	3·86%	8·92%	—
9.30 a.m. Oct. 5th	6·0%	25·20%	1·74%	2·15%	8·30%	54·00%

The relative merits of magnesite and dolomite have promoted some interesting discussions in recent years. Karnaokhov[1] has reviewed the subject very thoroughly and suggests the correct procedure to be adopted when preparing and maintaining hearths which have been sintered with magnesite. He advocates that magnesite hearths should be repaired with magnesite, and not with burnt dolomite, which appears the more usual practice.

The employment of magnesite is more favoured in the U.S.A. than in this country. Several brands of hearth-making materials with a magnesite base and containing a suitable bond have from time to time been introduced. In America considerable use is made of "Ramix", which is a high magnesia refractory, containing a chemical bond. This material is rammed into position cold, with pneumatic hammers, in a manner similar to that described in connection with tarred-dolomite. Excellent results have been obtained for such hearths, which are generally maintained with dead burnt dolomite.

In this country the usual practice is to employ dead burnt dolomite which may be either rammed or burnt to shape. These hearths are maintained with dolomite. The more scientific grading of dolomite, the more accurate control of the tar content, and the improved ramming technique, has resulted in excellent lives being obtained from this type of hearth. The grading recommended for an open-hearth furnace is shown in Table XXVI.

The most recent development in the preparation of the working hearth is the use of graphitized-dolomite. This method, however, will be discussed in more detail in connection with electric furnaces, where many successful trials have been made. The work on open-hearth furnaces is still in an experimental stage, although encouraging results are indicated.

As pointed out by Andrew and Jurbenko,[2] one of the most important factors in the performance of the working hearth is shape. It is important that a gentle flow towards the tap hole is maintained, as this prevents corrosive slags being allowed to collect.

Table XXVIA gives the chemical analysis of typical hearth materials.

TABLE XXVI

GRADING OF DRY DOLOMA FOR BASIC OPEN-HEARTH FURNACES

Grading B.S.I. Sieves	B.F.L.C. Grade		Range of grades employed at various steel plants
Over ½ in.	Nil	32%	Nil to 2·0%
½ to ¼ in.	14%		14 to 18%
¼ to 7 mesh	18%		18 to 22%
7 to 25 mesh	22%	37%	22 to 26%
25 to 72 mesh	15%		11 to 15%
72 to 150 mesh	11%	31%	9 to 12%
Less than 150 mesh	20%		14 to 20%
Packing Density, g/ml:			
(A.F.A. Rammer, 50 blows)	2·23		2·2 to 2·6
Bulk Density, g/ml:			
¼—½ fraction	3·06		—
¼ to 7 mesh fraction	2·85		—
Static angle of repose			
on 5 in. shelf	57°		-

The tar content for the doloma for the ramming of open-hearth furnace bottoms varied at a number of plants between 5 and 8 per cent. The material recommended for the preparation of open-hearth bottoms by the Committee was 6 per cent. tar.

TABLE XXVIA

TYPICAL HEARTH MATERIALS

Material	SiO_2	Fe_2O_3	Al_2O_3	MnO	CaO	MgO	Loss
Magnesite	2·56	3·11	2·09	1·48	3·72	86·20	0·36
Magnesite (Russian)	2·90	—	—	—	tr.	90·90	—
„ (Rhodesian)	2·00	1·04	0·46	—	2·09	93·50	0·70
„ (Canadian)	3·20	7·52	4·30	—	18·40	66·30	0·11
Dolomite	2·16	2·83	1·60	0·91	53·90	36·60	1·60
Dolomite	3·73	2·97	1·39	0·99	56·33	33.10	1·20
Dolomite	4·06	2·78	2·25	0·76	55·90	32·45	1·56
Dolomite (Tarred)	3·25	2·43	1·62	0·60	35·80	47·70	8·70
Doloset (Cement)	8·90	9·70	3·40	—	39·30	38·10	0·80

Selection of Raw Materials

The basic open-hearth process permits the employment of a wide range of raw materials. Materials high in silica, however, should be avoided, owing to their tendency to attack the basic lining and disturb the basicity of the slag, thereby retarding the refining process.

Normally basic pig iron is specified to the following analysis:

Carbon	3·0/3·6 per cent.
Silicon	1·0 per cent. max.
Sulphur	0·06 per cent. max.
Phosphorus	1·0/1·5 per cent.
Manganese	1·0/3·0 per cent.

The actual composition of the pig iron desired, however, depends to a large extent on the type of process adopted. When operating a cold pig iron and scrap process, employing a high percentage of cheap scrap, a high carbon is an advantage. When operating with a high percentage of molten pig iron and a minimum amount of scrap low carbon and phosphorus is undoubtedly ideal. These factors will, however, be more clearly appreciated when the various types of basic processes are reviewed and when the bath reactions are duly considered.

Reaction in the Bath

The *carbon* is readily removed by interaction between iron oxide and carbon, thus:

$$C + FeO \rightleftharpoons CO + Fe$$

the carbon monoxide produced escaping as a gas. This is an endothermic reaction, and it produces an agitation or "boil" which is of considerable, in fact fundamental, importance. The stirring action due to the elimination of carbon promotes or assists:

(1) The transfer of heat through the metallic bath.

(2) Assure better slag/metal contact, and thereby enhances the refining of the metal.

(3) Assists the segregation of the oxidation products which might otherwise be trapped as non-metallic inclusions.

(4) Assists the removal of hydrogen and other occluded gases. During the melting process considerable quantities of hydrogen may be occluded, but as shown by the Heterogeneity Committee,[3] much of this occluded gas will be evolved during the carbon boil. The gases held in solution in the molten metal will tend to diffuse into the carbon monoxide bubbles and will thus be eliminated.

The removal of *phosphorus* takes place in three stages. The phosphorus is first oxidized to phosphorus pentoxide:

$$4P + 5O_2 \rightarrow 2P_2O_5$$

This phosphoric acid temporarily forms iron phosphate. The iron phosphate, however, is very unstable. It is unable to withstand the action of the other reducing agents and is only stable in a highly oxygenated medium. The final stage is the conversion into a more stable phosphate of calcium.

The phosphorus reaction is highly exothermic. According to Dickman,[4] 1 kilogramme of phosphorus on oxidation to P_2O_5 evolves 5,965 calories. Pawloff[5] states that 1 kilogramme of P_2O_5 combines with lime to form $Ca_4P_2O_9$ with the evolution of 1,131 calories.

The exothermic nature of this reaction accounts for the difficulty frequently experienced in the removal of phosphorus in an extremely hot furnace. It is often necessary to cool the bath by cold scrap additions before the phosphorus can be satisfactorily eliminated. The factor is confirmed by Pullen,[6] who shows that at high temperatures carbon is removed before the phosphorus. At low temperatures the reverse is true. He quotes the drastic practice of taking all gas off the furnace to assist phosphorus removal in the case of difficulty due to high temperature conditions.

The form in which phosphorus passes into the slag has been the subject of some controversy and is apparently dependent on several factors. Bainbridge[7] and Scheiderholm[8] show that in a non-spar treated slag, phosphorus exists as silico-carnotite ($5CaO \cdot P_2O_5 \cdot SiO_2$). This phosphate has a high solubility in citric acid.

The addition of fluor-spar to the slag favours the formation of fluorapatite $3(3CaO\ P_2O_5)\ CaF_2$. This type of phosphate has a low citric acid solubility.

The reaction:

$$2P + 5FeO \rightleftharpoons 5\ Fe + P_2O_5$$

has been thoroughly investigated by several workers. In view of the exothermic nature of the oxidation of phosphorus, it will be influenced by temperature conditions. A rising temperature will reduce the velocity of the reaction on the direction of the upper arrow. The velocity of the reaction will also be affected by the concentration of the reacting constituents.

Balajiva and others[9] conducted a laboratory investigation into the phosphorus reaction and showed that the main factors in dephosphorization were the FeO, CaO and P_2O_5 contents of the slag. The equilibrium constant:

$$K = \frac{P_2O_5}{P^2\ (FeO)^5\ (CaO)^3}$$

was evaluated by statistical methods. McCance[10] shows the importance of the CaO content of the slag. He shows that the removal of phosphorus depends on the existence of oxidizing conditions which are influenced by the MnO/FeO ratio in the slag. High manganese pig iron tends to retard dephosphorization because:

(1) It reduces the oxidizing power of the slag.

(2) The sum of CaO + MgO + MnO is approximately constant within the limits 60/65 per cent. Therefore, high manganese will reduce the lime-carrying power of the slag and retard thereby dephosphorization.

The stability of phosphorus in the slag was discussed in detail by the author,[11] when he suggested that the amount of iron oxide in the slag required to prevent rephosphorization of the metallic bath was materially affected by the amount of fluor-spar used. Whereas 12 per cent. of Fe in the slag was necessary in the case of a non-spar treated slag, no rephosphorization occurred with 7·4 per cent. Fe in a spar-treated slag. The lime content of the slag was 46·0 per cent. and 51·2 per cent. respectively.

McCance[10] also shows the importance of the lime content of the slag. With a slag containing 6 per cent. P_2O_5 and 12 per cent. FeO he demonstrates that the phosphorus in the metal and the lime in the slag will bear the following relationship:

Phosphorus in the steel	CaO content of the slag
0·048 per cent.	40 per cent.
0·029 ,,	45 ,,
0·021 ,,	50 ,,

It is suggested that the stability of the phosphorus in the slag is dependent on the formation of fluorapatite rather than the less stable phosphates. In any case the phosphorus will return to the metal if the process of deoxidation is carried too far.

Pullen[6] states that in practice phosphorus can be removed and maintained in the slag by two different types of slag:

(1) A very rich basic slag, i.e. very high in lime and rather low in oxide of iron.

(2) A moderately basic slag, i.e. moderately high in lime but very high in oxide of iron.

The *sulphur reactions* are more complex. During the melting process considerable quantities of sulphur are absorbed from the furnace atmosphere. The amount absorbed appears to depend on the sulphur content of the fuel and the length of time there is flame/solid contact. The amount of sulphur absorbed after the formation of a slag covering shows a decrease.

Any sulphur in the furnace gases or fuel tends to become oxidized to sulphuric acid, which reacts with lime to form a calcium sulphate. This sulphate is decomposed at high temperature, the sulphur passing into the metal:

$$CaSO_4 + 4Fe \rightarrow FeS + CaO + 3FeO$$

The author[11] shows that under certain conditions this sulphur absorption may be considerable, as shown in Table XXVIIA.

TABLE XXVIIA

SULPHUR PICK-UP

Total Sulphur charged	Sulphur in the Steel	Sulphur in the Slag	Gain in Sulphur	
			tons	per cent.
	tons	tons		
0·03 tons	56×0·04 =0·0224	8·5×0·30 =0·0255	0·0179	56·6
0·024 ,,	57×0·04 =0·0171	10 ×0·16 =0·016	0·0091	38·0
0·0288 ,,	55×0·044=0·0242	9·6×0·228=0·0218	0·0172	60·6

McCance[10] discusses this sulphur pick-up in the light of more modern research and suggests that the reactions responsible are:

$$3Fe + SO_2 = FeS + 2FeO$$

$$10FeO + SO_2 = FeS + 3Fe_3O_4$$

Both reactions are exothermic and, therefore, the absorption of sulphur will be less at high than at lower temperatures. Quick-hot melting will diminish the extent of sulphur absorption by the metallic bath.

In the field of non-ferrous metallurgy considerable use is made of the so-called precipitation process for the recovery of metals from their sulphides. This process depends on the replacement of the metal from its sulphide by a less noble metal, that is, a metal having a lower sulphide tension. This principle underlies the desulphurization of iron by manganese. The addition of ferro-manganese to the bath results in the decomposition of iron sulphide and the formation of manganese sulphide, which segregates to the slag, thus:

$$FeS + Mn \rightleftharpoons MnS + Fe$$

This reaction, however, is not complete and a mixed sulphide of iron and manganese is formed, which is capable of existing either in the metal or the slag.

McCance[10] discusses this reaction and shows that the constant of the reaction is:

$$K_S = \frac{MnS}{FeS \, . \, Mn}$$

but at any fixed temperature, if the iron is saturated with MnS the concentration of this constituent can be neglected, so that:

$$K_S = S \, . \, Mn$$

This reaction is also influenced by temperature. A falling temperature favours the reaction proceeding from left to right. The solubility of MnS in iron decreases with a falling temperature. These two factors account for the desulphurization by manganese that takes place in the mixer.

It has been suggested by Kintyle[12] and Colclough[13] that MnS, in the presence of oxygen, possesses the property of evolving SO_2, which escapes from the furnace whereby the equilibrium of the slag is disturbed, rendering the slag capable of absorbing further quantities of sulphur. To what extent this reaction proceeds is doubtful, as already discussed by the author.[11]

It would appear that the removal of sulphur is dependent on its conversion into a form which is stable in the slag and not soluble in the metal. This form is calcium sulphide and the elimination of sulphur can be expressed:

$$FeS + CaO \rightleftharpoons FeO + CaS$$

A high concentration of lime favours the removal of sulphur, whilst a deficiency of lime or an over-oxidized condition of the metallic bath will tend to force the reaction in the direction of the lower arrow.

McCance[10] shows that the velocity of the reaction is influenced by the CaO/FeO ratio. In the Acid Process where CaO/FeO is very low, there is no elimination of sulphur. It is pointed out that the MnO will act in the same way as FeO in preventing sulphur transference, and in practice the effective ratio can be taken as $CaO/(FeO + MnO)$.

This reaction appears to be little affected by changes of temperature. In practice, however, temperature seems to play an important part, probably due to the increased fluidity and lime-carrying power imparted to the slag. Taking the heat of formation per lb. molecule of the reactants as:

FeO =	118,272	B.Th.U.
FeS =	43,200	„
CaO =	236,700	„
CaS =	169,740	„

the reaction is exothermic, thus:

$$FeS + CaO \rightleftharpoons FeO + CaS + 8{,}112 \text{ B.Th.U. per lb. molecule.}$$

The elimination of sulphur is enhanced by the presence of carbon. This is partly due to the better metal-slag contact promoted by the carbon boil and partly due to the reduction of FeO by carbon, thus:

$$FeS + CaO + C \rightarrow Fe + CaS + CO$$

The ideal conditions, therefore, for the elimination of sulphur are:

(1) A slag rich in lime.
(2) An active carbon boil.
(3) Control of the state of oxidation.
(4) High temperature conditions.

Several years ago the author's attention was drawn to the apparent effect of the FeO/Fe_2O_3 ratio on desulphurization. It appeared that the best conditions for the removal of sulphur existed when this ratio was between 1·2 and 2·0. This may be due to the presence of lime and iron oxide as tricalcic-ferrite $3CaO\ Fe_2O_3$ which would render lime readily available for desulphurization. Some typical results taken from a paper by the author are given in Table XXVIIB. It is probable that the reaction could be expressed by the equation:

$$3FeS + 3CaO\ Fe_2O_3 + 6C = 3CaS + 5Fe + 6CO$$

The presence of manganese which appears to accelerate the removal of sulphur may be due to indirect action, whereby the manganese reduces the FeO concentration and thereby favours the formation of CaS.

These reactions are also influenced by the other reactions taking place in the metallic bath. The elimination of silicon and manganese all affect the slag conditions and the state of oxidation and must therefore affect the sulphur reactions.

TABLE XXVIIB.

Heat No.	FeO/Fe_2O_3 Ratio	Desulphurization
1,981	5 7 : 1	Sulphur returned
735	3 0 : 1	Sulphur returned
1,973	2·2 : 1	Sulphur returned
1,978	1 9 : 1	Sulphur dropped from 0 046 to 0 038
1,979	1 8 : 1	Sulphur stationary
1,988	1 6 : 1	Sulphur dropped from 0 050 to 0 042
1,977	1 5 : 1	Sulphur dropped from 0 056 to 0 036
1,965	1 4 : 1	Sulphur dropped from 0 060 to 0 046
1,963	1 4 : 1	Sulphur dropped from 0 054 to 0 039

Silicon is readily removed in the basic furnace and passes into the slag, either as di-calcic or tri-calcic silicate. The silicon is oxidized:

$$Si + 2FeO \rightarrow 2Fe + SiO_2$$

and under normal conditions passes into the slag as tri-calcic silicate, $3CaO\ SiO_2$. In slags high in silica it may exist as di-calcic silicate $2CaO\ SiO_2$. In non-spar treated slags it is linked with phosphorus as silicacarnotite. $5CaO\ P_2O_5\ SiO_2$.

Manganese is removed in the basic open-hearth furnace according to the reaction:

$$Mn + FeO \rightleftharpoons MnO + Fe$$

the manganese oxide passing into the slag. The velocity of this reaction will be influenced by the concentration of iron oxide present in the slag. As early as 1907 Dr. Naske[15] investigated this equilibrium and demonstrated that when:

$FeO\ \% = MnO\ \%$ – equilibrium exists.

$FeO > MnO\ \%$ – Mn is eliminated.

$FeO < MnO\ \%$ – MnO is reduced.

The reaction has been more recently studied by McCance.[10] He shows that if the partition coefficients are L_{MnO} and L_{FeO} the equilibrium constant is:

$$K_{Mn} = \frac{L_{MnO}}{L_{FeO}} \cdot \frac{(MnO)\quad Fe}{(FeO)\quad Mn}$$

Herty[16] simplified this statement by asserting that the speed of the reaction is proportional to the concentration existing in the reacting media. The distribution ratio for manganese may be denoted by K_{Mn} and this can be calculated from the equation:

$$K_{Mn} = \frac{(MnO)\ Fe}{(FeO)\ Mn}$$

The manganese reaction, however, is also affected by the CaO concentration. An increase in the CaO concentration in the slag in the presence of an active carbon boil may result in the reduction of manganese oxide and the return of manganese to the bath :

$$CaO \cdot MnO \cdot SiO_2 + CaO + C \rightarrow 2CaO\ SiO_2 + Mn + CO.$$

Lawrie[17] gives several examples of the reduction of manganese following the addition of lime to the slag. These lime additions were accompanied by fluorspar, which rendered the slag more fluid and thereby increased the lime-carrying power. This feature, however, was only observed during the earlier stages of refining, and may be connected with the carbon boil. During the later stages of refining, further lime and fluorspar additions do not appear to cause any return of manganese to the bath, although the slag is increasing in basicity. It is suggested[17] that under these conditions manganese may form an acid oxide, such as MnO_3 or MnO_2 and thus produce a calcium manganite ($CaO\ MnO_2$).

It should also be remembered that the reaction:

$$FeO + Mn \rightleftharpoons Fe + MnO$$

is influenced by temperature. A rising temperature, such as exists during the latter stages, will accelerate the reaction in the direction of the upper arrow.

Slag Control

In the basic open-hearth furnace the slag is active and it has a two-fold function:

(1) It is the oxidizing medium responsible for the removal of the impurities from the metallic bath.

(2) It is the receptacle which receives and holds these impurities as stable constituents.

During recent years considerable research has been undertaken in connection with the constitution and control of this slag. The objects of slag control are:

(1) The efficient removal of impurities, whereby maximum rate of output is achieved.

(2) The reduction of metallic losses in the slag, which results in reduced ingot yield.

(3) The protection of the furnace lining, whereby the wear and tear of the working hearth is reduced as much as possible.

(4) The achievement of refining without excessive oxidation of the finished steel.

(5) The reduction of non-metallic inclusions in the finished steel to a minimum.

The degree to which slag control can be applied is largely the result of the work done recently on the constitution of basic open-hearth slags. Ferguson,[18] in a thorough microscopical examination of rapidly cooled slags, identified the following constituents:

(1) Tri-basic Ferrite (Tricalcic Ferrite) $3CaO\ Fe_2O_3$.

(2) A magnetic constituent, which appears to be a mixture of Fe_3O_4, Fe_2O_3, MnO and CaO. This constituent may contain an unstable combination of CaO and Fe_3O_4. He also suggests that under oxidizing conditions MnO may act as a base towards Fe_2O_3.

(3) Tricalcic Phosphate. In spar-treated slags fluorapatite $3\ (3CaO\ P_2O_5)\ CaF_2$ was separated in quantity.

(4) Tricalcic silicate. $3CaO\ SiO_2$.

(5) Dicalcic silicate, $2CaO\ SiO_2$, which appears to be characteristic of slags high in silica.

Ferguson[18] shows that cutting down and melting slags are dibasic in character. Cutting down slags fall within the range of CaO 24/30 per cent., Fe_2O_3 12/15 per cent. and FeO 30/50 per cent. Melting slags, which contain much free lime, normally range CaO 41/52 per cent., SiO_2 21/27 per cent. Fe_2O_3 1·0/3·6 per cent., FeO 3/12 per cent. and MnO 5·5/7·6 per cent. He suggests that finishing slags are tribasic, consisting of tricalcic silicate, free lime crystals, tribasic phosphate, tricalcic ferrite and a magnetic constituent.

In a more recent mineralogical investigation Agrell[19] has shown the principal constituents of tapping slags are lime (CaO), magnesio-wustite (Mg Fe Mn O), nagelschmidtite ($7CaO\ P_2O_5\ 2SiO_2$) di- or tri-calcium silicate and apatite. Smaller quantities of monticellite ($CaO\ MgO\ SiO_2$) merwinite ($3CaO\ MgO\ 2SiO_2$) and calcium ferrites ($CaO\ Fe_2O_3$, $2CaO\ Fe_2O_3$ and $3CaO\ Fe_2O_3$) were also identified.

In general this work agrees with the findings of Bainbridge[20] and Scheiderholm[21] which shows that:

1. In non-spar treated slags phosphorus exists chiefly as silico-carnotite ($5CaO\ P_2O_5\ SiO_2$). This form of phosphate has a high solubility in citric acid.
2. In spar-treated slags the phosphorus exists chiefly as fluorapatite $3(3CaO\ P_2O_5)\ CaF_2$. This type of phosphate has a low citric acid solubility.

The chemical composition will vary with the impurities present in the furnace charge. Normally CaO will range between 45 and 55 per cent., whilst the P_2O_5 and SiO_2 together will represent 16 to 25 per cent. From these considerations the lime or limestone requirement of any charge can be calculated thus:

If the charge contains:

60 tons of scrap with 0·1 per cent. Si and 0·05 per cent. P.
40 tons of pig iron with 0·5 per cent. Si and 1·50 per cent. P.

the lime requirement will be:

Silica $60 \times 0{\cdot}1/100 = 0{\cdot}06$ ton silicon
$40 \times 0{\cdot}5/100 = 0{\cdot}20$ ton silicon
$0{\cdot}26 \times 2{\cdot}2 = 0{\cdot}571$ tons SiO_2

Silica adhering to raw materials $\dfrac{100 \times 0{\cdot}75}{100} = 0{\cdot}750$ tons SiO_2
Silica from furnace structure .. .. $= 0{\cdot}250$ tons SiO_2
Total $1{\cdot}572$ tons SiO_2

Phosphorus $40 \times 1{\cdot}5/100 = \dfrac{0{\cdot}60 \text{ tons Phos.}}{0{\cdot}60 \times 2{\cdot}3} = 1{\cdot}38$ tons P_2O_5

The total of acids ($SiO_2 + P_2O_5$) is 2·95 tons and assuming that there is in the finished slag about 20 per cent., the slag volume will be:

$$\frac{2{\cdot}95 \times 100}{20} = 14{\cdot}75 \text{ tons of slag.}$$

Since the slag contains approximately 50 per cent. CaO, the burnt lime requirement is approximately 7·5 tons per 100 tons charged.

Several systems of slag control have been suggested including rapid methods of determining the basicity and iron oxide content. These methods can be summarized as follows:

(1) *Chemical methods*, in which a sample of metal and slag is taken and necessary lime and iron oxide are made according to the analysis obtained. In spite of rapid methods of analysis introduced, there is a time-lag of about half an hour between the ascertainment of bath conditions and the application of the remedy.

(2) Herty[22] applies the *viscosimeter* discussed in Chapter 7, page 169, in which the inches of flow, indicative of the viscosity of the slag, are related to chemical composition, as shown in Fig. 110A. This method of measuring the viscosity is, however, affected by:

  (a) Melting point, superheat, specific heat and surface tension of the slag.
  (b) Variations in sampling and pouring technique.
  (c) Fluorspar additions and varying oxide contents of the slag.

  When the sampling and pouring technique is carefully standardized and controlled, this method has been successfully applied, in conjunction with iron oxide analysis, to the premelt stage. The relationship between viscosity and composition is shown in Fig. 110A and the elaborate system suggested has been thoroughly outlined by Herty.[23]

(3) *The colour of the slag* when cooled in water is also indicative of the composition, thus:

Colour	FeO content	CaO/SiO_2 ratio
Black	Medium	Low
Grey	Low	Low
Light brown	Medium	Medium
Dark brown	Medium	High
Chocolate brown	High	High

  "Very similar colours are evident in finely powdered slag samples".[24]

(4) *The appearance of the slag* in the furnace is another useful guide to its condition. The following types of slag can be readily identified:

(a) Heavy lime slag, which contains ample CaO and frequently a medium to high iron oxide content, appears as a "lumpy corn meal mush".[24] This type of slag reacts to fluorspar additions, and when opened out in this manner favours rapid removal of phosphorus and sulphur.

(b) Thin, watery slag, usually high in SiO_2 and low in iron oxides, requires additional lime.

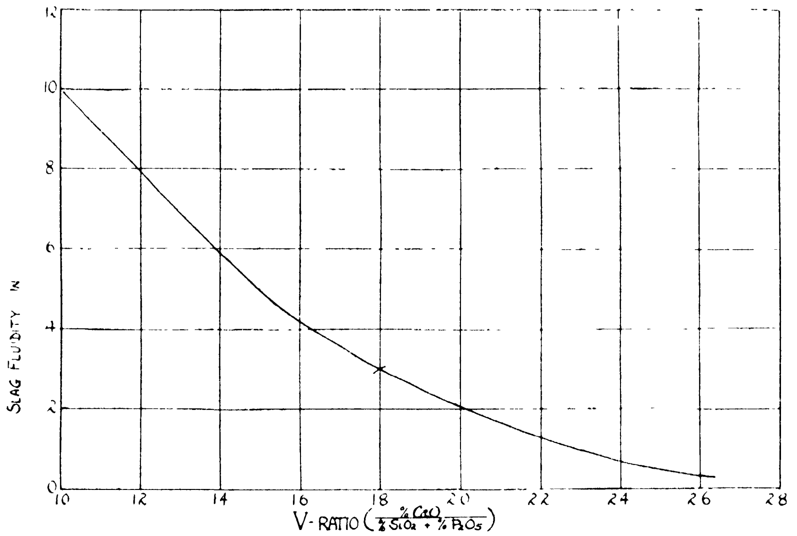

FIG. 110A. The general relation between slag viscosity and basicity during melting (Herty[23]).

(5) *A slag pancake method* has been suggested and discussed by Back,[25] Soler[26] and Mauer.[27] These pancakes are formed by rapidly pouring the molten slag into a cast steel mould or receptacle, about 4½ in. dia. by ¾ in. deep, with a slightly tapering wall, from which they should be removed as soon as they are set. If they remain in contact with the mould too long, the under surface becomes dull. From the examination of these pancakes, considerable information regarding basicity can be judged from the furrows, creases and markings of the upper surface, from the reflectivity of the lower surface and the colour of the slag. Careful standardization, however, is necessary owing to the fact that the specimens obtained are influenced by the raw materials employed, the melting practice and the pouring technique.

Acid slags, having a $CaO/SiO_2 + P_2O_5$ ratio below 1·0, show a flat or slightly concave surface, and are characterized by a columnar or acicular fracture with fibres running perpendicular to top and bottom surfaces. There is frequently a shrinkage void in the centre. This type of slag requires additional lime.

When the basicity ratio ($CaO/SiO_2 + P_2O_5$) ranges from 1·0 to 1·2, deep lines or creases appear and the sample shows a stony fracture. The slag is usually grey, but relatively high FeO and MnO contents make it dark brown.

As the basicity increases the creases become broader and finally change to furrows. This change is accompanied by a colour change from grey to black. Still further increases in basicity cause the number and depth of the furrows to decrease until a slight circular furrow remains. The bottom surface is shiny, with acid slags, becoming dull as the basicity increases.

This method of slag control offers many practical possibilities when once the technique of sampling and pouring have been standardized to meet local conditions. The change points are, however, greatly influenced by the iron oxide content.

(6) *Microscopic methods* have also been suggested, but the time factor is a disadvantage. A rapid method described by Tenenbaum and Joseph[28] consists of polishing slag lumps and examining them with reflected light under a metallographic microscope. This method required about ten minutes.

The Application of the Process

The actual application of the process and the type of charges employed depends on local conditions and markets. At composite works a high percentage of hot metal will be employed. In other localities, due to the existence of a plentiful supply of cheap scrap, large quantities of steel scrap will be incorporated in the charge.

The various types of metallic charge, which depend on local and market conditions, may be classified as follows:

(1) *All liquid metal* either direct from the blast furnace, or, as is the more usual practice, hot metal from activated mixers.

(2) *Liquid iron and liquid steel.* The molten iron is supplied from blast furnaces or mixers, whilst the liquid steel is produced in Bessemer convertors.

(3) *Liquid metal and solid steel.* Charges of this type usually contain

hot metal plus about 20 to 25 per cent. steel scrap, which arises in the rolling mills from current ingot make. In some cases this percentage is increased by steel scrap purchased from outside sources.

(4) *Solid iron and steel scrap.* This modification of the process is frequently referred to as the *cold pig and scrap process.* It is usually adopted at plants divorced from blast furnaces and situated in scrap producing localities.

The proportion of iron and steel used in the mixture charged depends to a large degree on supply and market conditions of the two commodities.

(5) *All steel scrap*, in which the carbon necessary to provide the refining boil is supplied by anthracite coal or coke. This recarburizing material is charged into the furnace with the steel scrap. Generally the practice is modified so that only part of the pig iron content of the mixture is replaced by the anthracite or coke.

The Process

The process can be divided into four stages:

(1) Charging.
(2) Melting.
(3) Refining.
(4) Finishing.

These stages will vary somewhat according to the modification of the process adopted and will gradually merge into each other.

The method of *charging* will depend on the percentage of hot metal, if any, that is employed. In the *cold pig and scrap* process it is the normal practice to charge the steel scrap first and to finish the charge with the necessary lime and pig iron. Pig iron, especially if sand-cast, should not be placed on the furnace hearth. Some plants prefer to sandwich the pig iron and lime, suggesting that such a method gives a better melting time. The important feature would appear to be the placing of the lime, and iron-oxide if the percentage of pig iron demands it, underneath the pig iron, so that more rapid slag formation and elimination of impurities takes place. This precaution also protects the banks and bottom of the furnace from the corrosive action of the impurities in the pig iron. Typical heats are shown in Tables XXVIII to XXXIII.

TABLE XXVIII

Working of a Typical Charge in a 250-ton Tilting Furnace

P.M.

3.00 Tapped two ladles.
3.15 Fettled breasts and ends.
3.50 Remainder of steel tapped.
4.10 General fettling started.
4.50 Fettling finished except top of back wall.
4.55 Commenced charging burnt lime.
5.10 5 tons 5 cwt. burnt lime charged.
5.15 Finished back wall fettling.
5.45 Commenced charging ore.
6.10 22 tons Swedish ore charged.
7.30 30 tons scrap charged.
7.46 13 tons Swedish ore charged.
7.50 Started charging 30 tons of scrap and 4 tons 4 cwt. of burnt lime.
8.30 All cold material charged.
8.30 55 tons mixer metal.
8.50 52 tons mixer metal.
9.10 50 tons mixer metal.
9.50 44 tons mixer metal; Analysis. Si 0·47 per cent., S 0·05 per cent.

A.M.

2.05 6 tons slag off.
4.00 26 cwt. Swedish ore.
4.45 6 tons slag off.
5.00 6 tons slag off.
5.00 Sample: C, 0·40 per cent., P, 0·45 per cent.
5.15 4 tons burnt lime.
6.00 35 cwt. scale.
7.30 26 cwt. ore.
8.30 21 cwt. burnt lime and 18 cwt. scale.
9.00 14 cwt. burnt lime and 18 cwt. scale.
9.20 Sample: C, 0·25 per cent., S, 0·05 per cent., P, 0·20 per cent.
9.45 21 cwt burnt lime, 18 cwt. scale, 12 cwt. ore.
10.35 Sample: C, 0·22 per cent., P, 0·08 per cent.
10.45 21 cwt. burnt lime, 12 cwt. ore.
11.10 14 cwt. burnt lime.
11.15 Sample: C, 0·085 per cent., Mn 0·17 per cent., P, 0·05 per cent.
11.40 14 cwt. burnt lime.

P.M.

12.30 Sample: C, 0·05 per cent, P, 0·024 per cent.
12.40 Tapped.

Weight of ingots, 286 tons.

Pit Sample Analysis

Cast No.	C%	Si%	S%	P%	Mn%	
A. 791	0·09	0·06	0·040	0·026	0·29	Semi-killed, soft
A. 792	0·09	0·014	0·039	0·065	0·40	High-phosphorus rimming.
A. 793	0·085	0·014	0·042	0·073	0·38	
A. 794	0·085	0·009	0·040	0·067	0·41	

(*Geary and Crook*[30])

TABLE XXIX

LOG OF A 200-TON CHARGE—APPROXIMATELY 75 PER CENT. HOT METAL

P.M.	
9.55	Tapped third ladle of last charge.
10.00	Bottom tested.
10.05	5 boxes limestone.
10.10	Fettling commenced.
10.50	Fettling finished and commenced charging.
11.10	15 tons scrap charged.
11.30	Half the remainder of the limestone.
11.40	Half the ore.
11.50	Remainder of limestone.
A.M.	
12.05	Remainder of ore.
12.25	Mixer iron, 50 tons 8 cwt.
12.45	Mixer iron, 52 tons 0 cwt.
12.50 to 4.45	Remainder of scrap.
4.55	Remainder of iron, 42 tons.
5.00	Finished charging.
8.30	Slagged off.
8.40	Sample: C, 0 50 per cent.; S, 0 047 per cent.; P, 0·25 per cent.; Fe, 4 5 per cent.,
9.30	½ cwt. spar, 3 cwt. scale. Sample: C, 0·36 per cent.; P, 0·18 per cent.; Fe, 4·0 per cent.
10.35	3 cwt. scale, 20 cwt. lime. Sample: C, 0·30 per cent.; P, 0·12 per cent.
11.25	3 cwt. scale, 10 cwt. lime. Sample: C, 0·23 per cent.; P, 0·06 per cent. Fe, 6·0 per cent ; Mn, 0·24 per cent.
11.50	20 cwt. lime. Sample: C, 0·16 per cent.; P, 0·05 per cent.
P.M.	
12.10	10 cwt. lime.
	Tapped: 12.10 P.M. first ladle. 12.20 P.M. second ladle. 12.45 P.M. third ladle. Delay on this ladle, crane breakdown.
	Tap-to-Tap, 14 hr. 50 min.

STEEL ANALYSIS

	C Per cent.	Si Per cent.	S Per cent.	P Per cent.	Mn Per cent.
Order	0·21-0·24	0·05	0·05	0·05	0·6–0·70
First ladle, pit sample ..	0·22	0·05	0·033	0·038	0·70
Second ladle, pit sample.	0·21	0·05	0·034	0·035	0·68
Third ladle, pit sample ..	0 23	0·05	0·036	0·039	0·72

Charge: 144 tons 8 cwt. mixer metal.
79 tons 10 cwt. mixed scrap.

Weight of ingots, 219 tons.

Analysis of mixer metal: Si, 0·42 per cent.; S, 0·051 per cent.; P, 1.69 per cent.; Mn, 2·04 per cent.

Additions to moulds: Nil.

(Davies[31])

TABLE XXX

Log of 185-Ton Charge

P.M.			
11·55	Charge tapped. Approx. 10 tons of steel left in furnace.		
A.M.			
12·30	Fettling finished.		
12·35	Commenced charging.		
1·25	25 tons	17 cwt.	Swedish ore.
	7 „	0 „	burnt lime.
3.40	52 „	16 „	steel scrap.
5.25	1 „	8 „	burnt lime.
5.30 to 6·30	106 „	0 „	mixer metal; 0·40 silicon; 0·06 per cent. sulphur.
9·40		14 „	burnt lime.
9·45	14 „	0 „	blast furnace metal; 0·80 per cent. silicon; 0·07 per cent. sulphur.
11·05	16 „	0 „	slag taken off: Total P_2O_5, 19·1 per cent.
11·15	3 „	3 „	burnt lime.
11·45	1 „	10 „	scale.
P.M.			
12·50		10 cwt.	burnt lime.
1·10			Tapped 63 tons 0 cwt. = 185 tons 2 cwt.
1·20			Tapped 62 „ 12 „
1·30			Tapped 59 „ 10 „

Rate of working = 13·6 tons per hr.
Time tap-to-tap = 13 hr. 35 min.

Bath Sample Analysis

	A.M. 11·15	P.M. 12·05	P.M. 12·20	P.M. 12·35	P.M. 12·45	P.M. 1·05
Carbon per cent. . .	0·35	0·33	0·30	0·26	0·22	0·17
Sulphur per cent. . .	0·057	—	—	0·052	—	—
Phosphorus per cent	0·030	—	0·04	—	—	
Manganese per cent. . .	—	—	—	—	0·23	—

Tapping slag = 10·40 per cent. total iron.

Pit Sample Analysis

	C %	Si %	S %	P %	Mn %
Specification . .	0·17–0·20	0·04–0·06	0·050	0·040	0·65–0·175
Cast analysis . .	0·185	0·051	0·049	0·038	0·70

(Winter[32]).

TABLE XXXI

Log of Cold-Metal Charge

Charge : 24 tons basic pig iron (40 per cent.).
36 tons steel scrap (60 per cent.).
60 Tons Total. (Total lime charged 5 tons.)

Time from commencing to charge to tap = 10 hrs. 10 mins.

Order of Charging

A.M.
1.25 20 tons steel scrap.
30 cwt. lime.
2.10 6 tons steel scrap.
3.30 3 tons 10 cwt. Lime.
24 „ 0 „ Basic pig iron.
4.15 10 „ 0 „ Steel scrap.

Between 6.30 a.m. and 10 a.m. there were frequent light feeds of fluorspar and mill scale to flux the lime. At 9.50 a.m. the charge was clear melted.

Sampling

Time A.M.	C%	S%	P%	Mn%	Fe in slag	Slag	Heat	Feed
10.00	0·90	0·036	0·40	—	4·3	Thick	Just melt	10 cwt. fluorspar
10.30	0·82	—	0·08	—	6·5	Slightly thick	Moderate	40 cwt. mill scale
11.5	0 70	—	0·03	—	7·9	Fluid	Fair	15 cwt. iron ore
11·20	0·65	—	—	—	7·7	„	Good	
11.30	0·63	—	—	0·30	8·5	„	Good	6 cwt. lime

Furnace tapped 11.35 a.m. Specification: 0·65/0·70 per cent. C and 0·50/0·60 per cent. Mn. Result: 0·68 per cent. C; 0·029 per cent. S; 0·030 per cent. P; 0·56 per cent. Mn.

Ladle Additions

4 cwt.	75 per cent. Ferro silicon.
5¼ cwt.	80 per cent. Ferro Manganese.
28 lb.	Anthracite coal.

Typical Finishing Slags

CaO	52·06 per cent.
SiO_2	14·75 per cent.
P_2O_5	3·80 per cent.
FeO	12·30 per cent.
Fe_2O_3	2·30 per cent.
Fe	11·30 per cent.
MgO	3·00 per cent.
MnO	4·80 per cent.

TABLE XXXII

TYPICAL COLD PIG AND SCRAP CHARGE—60 TONS

Charge: Steel scrap .. .. 48 tons.
Basic pig iron .. 16 tons.
Total 64 tons.

Lime charged 3 tons.

ORDER OF CHARGING

12 noon	30 tons of heavy steel scrap. 30 cwt. of lime.
P.M.	
1.40	18 tons scrap rails.
2.00	10 tons miscellaneous scrap.
3.15	30 cwt. lime. 16 tons basic pig iron.

Between 7 and 8 p.m. occasional light feeds of mill scale were made to open out the charge. At 8 p.m. the charge was clear melted.

SAMPLING

Time	Metal				Fe in slag	Heat	Feed
	C	S	P	Mn			
8·20	1·90	0·080	0·50	0·20	6·8	Just melted	3 tons iron ore
9·20	0·98	0·080	0·40	0·22	7·1	Fair	2 tons lime
10·25	0·50	0·095	0·09	0·25	11·5	Good	10 cwt. scale
11·40	0·38	0·060	0·06	0·30	12·0	Good	20 cwt. lime
12·10	0·31	0·046	0·04	0·25	—	Good	10 cwt. lime
12·20	0·30	—	—	0·24	—	Tapping	2 cwt. spiegel

Furnace tapped	C	Si	S	P	Mn
Specification	0·5/0·55	0·20	0·050 max	0·050 max	0·60/0·65
Results .. .	0·53	0·21	0·044	0·045	0·63

LADLE ADDITIONS

4½ cwt. 75 per cent. Ferro silicon.
7¼ cwt. 80 per cent. Ferro manganese.
483 lb. Anthracite.

	Tons	Cwt.	
Total lime consumed (in charge)	3	0	5 tons 10 cwt.
„ „ (in refining)	2	10	

TABLE XXXIII

LOG OF A LOW PIG IRON PLUS COKE CHARGE

Furnace charge	51 tons steel scrap. 80 per cent.
	10 tons pig iron. 16 per cent. (Si 0·6 P 1·4)
	3 tons cast iron. 4 per cent. (Si 2·0 P 1·5)
	64 tons total.

2¾ tons lime.
10 cwt. coke.

ORDER OF CHARGING

A.M.	
10.00	8 tons scrap on the bottom.
	30 cwt. of lime.
	10 cwt. of coke.
	10 tons rough steel scrap.
11.00	17 tons heavy steel scrap.
P.M.	
12.45	25 cwt. of lime
	10 tons of pig iron.
	3 tons of cast iron.
	5 tons of steel scrap.
2.15	6 tons of steel scrap.
2.35	5 tons of steel scrap.

Furnace charged 4 hrs. 35 mins.

SAMPLING

Time	C	S	P	Mn	Fe in slag	Condition of slag	Feed
7·40	0·60	0·054	0·08	—	7·5	Mushy	10 cwt. mill scale
8·16	0·35	0·048	0·05	—	9·5	Fluid	5 cwt. mill scale
9·00	—	—	—	—	—	Fluid	9 cwt. lime
9·30	0·12	—	—	0·10	—	Good	—

Furnace tapped 9.45 p.m. Time commencing to charge to tap 11 hrs. 45 mins.

	C	Si	S	P	Mn
Specification ..	18/22	0·10/0·15	0·050 max.	0·050 max.	0·55/0·70
Finished steel ..	0·21	0·115	0·041	0·026	0·63

Ladle additions	4¾ cwt. 45 per cent. Ferro Silicon.
	11 cwt. 80 per cent. Ferro Manganese.
	105 lb. anthracite coal (5 bags at 21 lb.).

FINISHED SLAG

SiO_2 %	Fe %	FeO %	Fe_2O_3 %	Al_2O_3 %	CaO %	MnO %	MgO %	P_2O_5 %	S %
13·40	12·80	11·19	5·86	4·95	50·8	4·05	4·18	3·53	0·41

At plants where cast-iron scrap replaces some or most of the pig iron, the same remarks apply to cast iron, which is usually high in silicon and has therefore a more corrosive effect.

When high percentages of steel scrap are employed with a recarburizer, the anthracite or coke is usually charged early. Sufficient lime should be placed under the recarburizer to deal with the attendant ash. The author prefers the use of lumpy coke, which appears to have a greater and more regular efficiency than smaller material. Two cwt. of coke is equivalent to about 30 cwt. of pig iron. If anthracite is used the material should be pea size, free from fine material and with a low volatile content (max. 6·0 per cent. volatile matter). A high volatile and too much fine material may cause explosive mixtures, especially if charged into any liquid material. A typical heat is shown in Table XXXIII.

When using *hot metal* the charging procedure will depend on the percentage of metal employed. The usual practice is to charge the steel scrap, lime and iron oxide prior to the addition of any molten metal. It appears advisable to bring this solid material to a state of incipient fusion before the hot metal addition. Typical heats are shown in Tables XXVIII to XXX. The hot metal addition frequently causes a violent reaction, which may cause the reaction slag to overflow the furnace doors.

There are two schools of thought concerning the charging of *lime and/or limestone.* The use of limestone, of course, is cheaper, and it contains less sulphur than burnt lime, but it requires an increased heat input for its calcination, and its use tends to retard the rate of output. Whether all the lime required to refine the metal should be charged, or whether only part of that total should be charged, and the rest added during refining, is another controversial point. The charging of all the lime retards the rate of melting, but favours early slag formation and assists refining operations. Tables XXXI and XXXII demonstrate the merits and demerits of the two methods.

When the bath is clear melted the *refining process* is the same in either the hot or cold metal practice. Providing the silicon is not abnormally high and sufficient oxide is available, or else is fed into the furnace, the carbon boil will commence. If the silicon content of the charge is very high, this silicon must be oxidized and converted into a silicate of lime before the elimination of carbon proceeds.

The essential conditions for the *removal of the impurities* have already been discussed, but they are best summarized:

(1) *Sulphur* requires:

  (a) An active carbon boil.
  (b) Control of the state of oxidation.
  (c) Highly basic slag.
  (d) High temperature condition which renders a highly basic slag fluid.

(2) *Phosphorus* requires:

  (a) A basic slag.
  (b) A sufficiently high iron oxide content in the slag to render the calcium phosphate formed stable. It is suggested a lower iron oxide content is necessary in a slag treated with fluorspar than in a non-spar-treated slag.
  (c) An excessively high temperature will retard the removal of phosphorus. Sometimes in the hot metal practice it is necessary to cool the bath by steel scrap additions to obtain satisfactory dephosphorization.

(3) *Manganese* elimination presents no difficulty. It is a function of the basicity and the iron oxide content of the slag. The conservation of manganese is, however, important from an economic and quality aspect, and will be considered later.

The application of these fundamental principles is more clearly announced in the Tables, which give the history of various heats.

When it is possible to remove slag, as in a tilting furnace, the refining and deoxidizing stages can be enhanced. The removal of the oxidizing slag essential to the elimination of phosphorus allows a less oxidizing slag to be employed towards the end of the process. This practice will reduce the amount of deoxidizers required and the tendency to form non-metallic inclusions. The advantages of this practice are shown in Tables XXVIII and XXIX.

The object of the various methods of operation is *to finish* the process with a slag as low in iron oxide as is consistent with the non-return of phosphorus into the metal. As shown by McCance:[10]

$$\frac{\text{FeO in metal}}{\text{FeO in slag}} = L_{\text{FeO}}$$

where L_{FeO} is the partition coefficient, which is a function of temperature. As the temperature increases, so the solubility of FeO in the metal increases. A high FeO in the slag favours a high FeO in the metal.

A high FeO in the metal results in over-oxidized steel, which requires more alloy additions to adjust the state of oxidation and produce the necessary chemical analysis specified for the particular steel. The extra quantity of deoxidizers employed gives rise to additional deoxidation products, which in turn aggravate the non-metallic inclusion problem. The object, therefore, is to keep the FeO content as low as possible, consistent with the prevention of rephosphorization.

The slag at the end of the refining has also an important bearing on the quality of the product. Thin slags frequently indicate over-oxidized steels, which may possess a high occluded gas content. Very thick slags, however, prevent the free escape of gases from the metallic bath and are often associated with mild steel. These aspects are discussed more fully, however, in Chapters 10 and 15.

Normally the *finishing additions* are made in the ladle. A limited degree of deoxidization can be carried out in the bath by the use of lumpy ferro-manganese or silico-manganese. The use of large quantities of these alloys, or of ferro-silicon, may disturb the basicity and state of oxidation of the slag to such an extent as to cause rephosphorization. The re-boil or pigging back referred to in connection with the acid process in Chapter 7 is sometimes employed. When this practice is employed frequently spiegel or high silicon pig iron is used. In hot-metal shops a dash of hot metal greatly assists the removal of excessive oxidation and correction of the final condition of the metal and slag. Care must, however, be taken to avoid carrying the deoxidation too far.

The cleanliness of the final steel can also be assisted by the correct selection of the deoxidizers. As shown by Herty,[29] silico-manganese alloys favour more the production of cleaner steels, than do ferro-manganese and ferro-silicon. The merits of silico-manganese alloys are discussed in Chapter 7. Even when added in the ladle, silico-manganese promotes cleaner steel. If ferro-manganese and ferro-silicon are employed in the ladle, the ferro-manganese should be added first if the tendency to form non-metallic inclusions is to be reduced. There is also an old adage that hot steel is clean steel. Providing the steel is not tapped too hot so as to cause other troubles, the increased fluidity of hot steel does assist the elimination of non-metallics.

The temperature of the bath may be judged either by:

(1) The immersion pyrometer discussed in Chapter 14.

(2) The rod test, which consists of inserting a mild steel rod into the metallic bath, and noting its appearance on withdrawal. The rod is moved about through the metal for a few moments and then

withdrawn. Its appearance indicates the temperature and slag conditions as shown in Fig. 110B.

(3) The spoon test, which consists of inserting a sample spoon, which is thoroughly slagged, and withdrawing a spoonful of metal. This metal is poured slowly on to the stage. If a perfectly clean spoon, free from any skulling, is left, the heat is satisfactory.

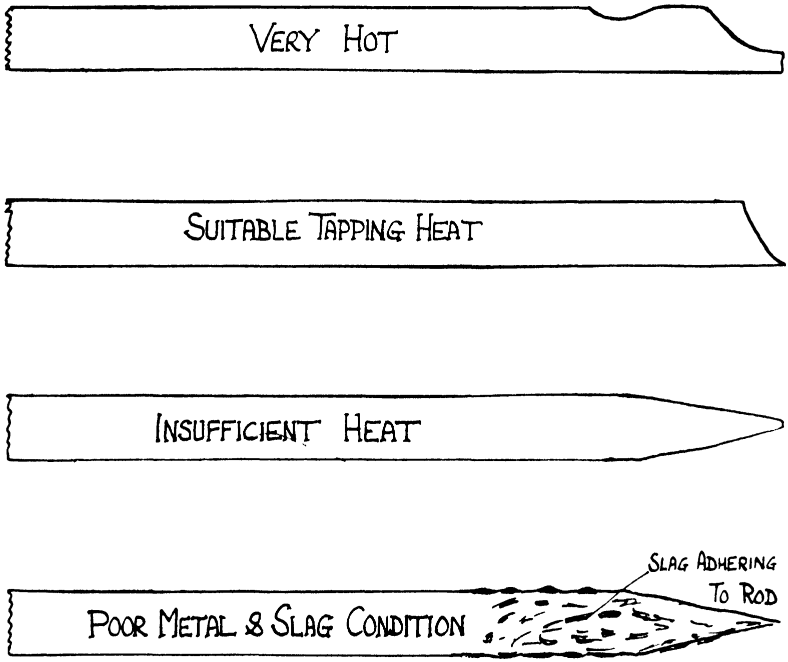

FIG. 110B. Examples of the rod test.

The amount of skull left in the spoon is an indication of bath temperature. The method of slagging the spoon is important. It should be rotated in the slag two or three times before being inserted into the bath.

Typical Examples

Tables XXVIII to XXXIII show some typical applications of the basic open-hearth process; tables XXVIII to XXX are examples of the *hot metal process*. The practice described by Geary and Crooks[30] is designed to produce a soluble slag suitable for fertilizing purposes. The use of fluorspar is avoided and the greater part of the slag is run off when the carbon has been boiled down to about 0·40 per cent. This slag has usually the following analysis:

CaO	42·5 per cent.
SiO_2	15·2 „
FeO	5·0 „
Fe_2O_3	3·9 „
Total P_2O_5	16·82 „
Soluble P_2O_5	15·95 „
Percentage soluble	94·8 „

A new slag is made up with lime and scale and the metal worked to specification. The details of operation are given in Table XXVIII. Table XXIX, taken from the work of Davies,[31] shows the operation of a 200-ton tilting furnace, employing approximately 75 per cent. of hot metal. In discussing this process the author expresses the opinion that tilting furnaces of this capacity require a minimum of 65 per cent. hot metal for the attainment of optimum operating conditions. The process employs two slags:

(1) The primary, taken off from the charging side of the furnace, is sold for grinding.

(2) The secondary is returned to the Company's blast furnaces as a flux.

The approximate composition of these slags is:

	Primary Slag.	Secondary slag.
SiO_2	12·28	9·90
FeO	9·80	14·66
Fe_2O_3	3·86	4·28
Al_2O_3	2·10	2·30
MnO	13.11	7·90
CaO	37·00	46·00
MgO	6·40	4·70
SO_3	0·172	0·137
P_2O_5	15·11	8·97

If owing to a troublesome slag fluorspar has to be used, the slag is classed as secondary, due to the lower solubility of the phosphate. The heat is finished with a slow-boiling creamy slag, and the steel, when tapped, requires very little, if any, mould additions.

Table XXX, taken from the work of Winter,[32] is another example of hot metal practice. A slightly different method of charging is adopted.

Two examples of the cold pig iron and scrap process are given in Tables XXXI and XXXII. In the first case practically all the lime required to refine the metal was added with the charge, whilst in the other example only part of the necessary lime was charged and the remainder fed during refining. The first method gives a lower slag volume, 16·6 per cent. against 17·2 per cent. Normally the magnesia content of the slag is higher in the latter method, indicating more erosion of the furnace lining.

An example of a coke charge is given in Table XXXIII. The very low metallurgical load, producing a small volume of slag, will be noted.

Types of Steel Produced

The types of steel produced by the basic open-hearth process cover a wide range of plain carbon and alloy steels. The general improvement in the control of the process, the more rigid selection of raw materials, the more suitable alloys available for deoxidation and degasification, and the developments in casting-pit practice and technique, have all contributed to the ability of the process to meet the demands for more rigid specifications and higher qualities of the final product.

The methods of adjusting the carbon, silicon and manganese is similar to those discussed for the Bessemer and acid open-hearth process. The required amount of alloy in cwt. can be determined from the expression:

$$\frac{(\%\text{ of element specified less residual }\%) \times (\text{weight of charge in cwt.})}{\%\text{ of element in alloy employed less loss on addition.}}$$

The loss on addition varies according to local factors, such as residual percentages of the elements concerned, the tapping temperature, the slag composition and condition, the state of oxidation and so forth. The estimation of the anticipated efficiency of any bath or ladle addition demands considerable skill from the melter. The methods of calculation are simple, but the anticipated loss must be judged from local practice and experience.

With ferro-silicon a normal loss is 25 per cent. of the silicon present in the alloy. This addition is made to the ladle and should follow the addition of ferro-manganese. The addition of ferro-silicon before ferro-manganese promotes the tendency to form non-metallic inclusions, owing to the type of oxidation product formed. As already discussed in connection with the acid open-hearth process, the cleanliness of the steel can be improved by the use of silico-manganese alloys.

Manganese can be adjusted by ferro-manganese, silico-manganese or spiegel. The loss encountered will largely depend on the state of oxidation, varying from 10 to 20 per cent. When employing a ferro-manganese containing 80 per cent. of manganese on a 60-ton charge, the author found the loss was approximately 20 per cent. when added in the ladle. If the metal was partially deoxidized by an addition of 4 cwt. of silico-manganese (60 per cent. manganese and 20 per cent. silicon) to the bath twenty minutes before tapping the heat, this loss in the ladle addition was reduced to 10 per cent.

When making a high manganese steel, it is advisable to add most of, if not all, the manganese in the bath before tapping. The loss of such additions will range from 25 to 35 per cent., depending largely on the factors already mentioned and the "time factor". If the ferro-manganese is added 15 to 20 min. before tapping the loss will average 30 to 35 per cent., but there is a tendency to produce cleaner steel. When the furnace is tapped almost immediately the ferro-manganese addition is made, a higher manganese recovery results, but there is a serious possibility of increased non-metallic inclusions.

Comparing furnace and ladle additions it should be remembered that ladle additions usually result in:

(1) Higher recoveries.
(2) Less thorough mixing.
(3) Greater likelihood of non-metallic inclusions.
(4) Less reaction with the furnace slag, which may result, reduces the risk of rephosphorization.

In the production of high carbon steel the older method of boiling the carbon right down and recarburizing with anthracite has been largely superseded by "catching the carbon" on the drop. When the carbon is within the specification required, and the sulphur and phosphorus are down to the required limits, the rate of the carbon boil is retarded by the addition of lime, which thickens up the slag, or by the addition of small quantities of silico-manganese or spiegel. Prior to this procedure a sample is taken and analysed for carbon and manganese The furnace is tapped and the carbon adjusted by small anthracite additions, due allowance being made for the carbon introduced by the ferro-manganese and other alloy additions. Allowance must also be made for any carbon eliminated whilst the sample was being analysed, and this act calls for considerable skill on the part of the melter.

In the manufacture of alloy steels the same general principles apply as in connection with the acid open hearth. The nickel and molybdenum contents are precisely the same, no loss being experienced. Ferro-chrome is frequently added to the ladle when low percentages of chromium are required. The loss is about the same as in the acid process. The alloy should be preheated before addition. When higher percentages of chromium are required, the ferro-chrome should be added to the bath. This addition to the furnace will result in a lower recovery. If the ferro-chrome employed contains 60 per cent. of chromium, the effective percentage will probably be about 50 per cent., depending on the length of time between its addition and the tapping of the furnace. Ferro-vanadium additions should be made to the ladle after the metal has been deoxidized by the less expensive alloys.

Metallic Yield

The metallic yield will be influenced by:

(1) The type of process adopted.
(2) The slag volume employed, owing to the iron content of the slag.
(3) The shape of the furnace hearth, which will affect the amount of metal that drains from the furnace after the ladle is filled. For this reason a tilting furnace usually gives a higher yield when other factors are the same.
(4) The amount of flame oxidation.

In the hot metal process, employing a high percentage of hot metal and iron ore, the yield will be considerably over 100 per cent. Frequently this yield averages 105 per cent., depending on the percentage of hot metal employed. The removal of the metalloids reduces the iron ore and the reduced iron passes into the metallic bath.

In the cold metal process, the yield will depend on the percentage of iron employed and the composition of that iron. The normal range is 89 to 91 per cent. When "off grade" pig iron is employed which necessitates high slag volume, this yield may drop to 87 to 88 per cent. Charges consisting of high percentages of scrap and a recarburizer produce small slag volumes, and the yield is usually about 92 to 94 per cent.

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CHAPTER 9

ELECTRIC FURNACES

Electrical energy is converted into heat owing to the resistance of a conductor. The resistance is a function of the voltage and amperage of the current passing through the conductor. According to Ohm's Law:

$$V = 1R$$

where V = volts (pressure)
l = amps (current)
R = ohms (resistance)

The power expanded in overcoming resistance equals the current squared multiplied by the resistance or:

$$P = 1^2R$$

Since one kilowatt hour (kWh) is equal to 860 kilogramme calories or 3,412 B.Th.U. these electrical values can be converted into heat units. The heat produced in any circuit equals:

$$H = \frac{1^2RT}{1{,}000}$$

where H = heat produced.
1 = current in amps.
R = resistance.
T = time during which the current is flowing.

On this basis the thermal equivalent would be:

$$H = 0{\cdot}860\ 1^2RT \text{ kilogramme calories}$$
$$\text{or } H = 3{\cdot}412\ 1^2RT \text{ B.Th.U.}$$

The electrical requirement to produce a definite thermal effect can be calculated.

Steel melts at approximately 1,500 deg. C. or 2,730 deg. F. It must therefore be raised through (2,730 − 60) or 2,670 deg. F. Since the specific heat of steel is 0·165 and the latent heat of fusion 195,000 B.Th.U. per ton,[1] the total heat required to melt one ton of steel is:

$$\begin{aligned} 2{,}240 \times 2{,}670 \times 0{\cdot}165 &= 970{,}000 \text{ B.Th.U.} \\ \text{Latent heat of fusion} &= \underline{195{,}000 \text{ B.Th.U.}} \\ & \quad 1{,}165{,}000 \text{ B.Th.U.} \end{aligned}$$

that is:

$$\frac{1{,}165{,}000}{3{,}412} = 342 \text{ kWh.}$$

Due allowance must be made, however, for radiation and electrical losses. Nevertheless, this theoretical figure assists the assessment of the thermal efficiency of the installation.

The production of heat by electrical power may be accomplished by two methods:

(1) By means of an arc, which produces a concentrated source of heat. The temperature of the electric arc is over 3,000 deg. C. The current flows through an indeterminate conductor consisting of the vapour between the electrodes.

(2) By means of resistance, in which the current flows through a definite conductor and heat is produced by the resistance of that conductor to the passage of the current. The heat is thereby uniformly radiated.

Both these methods are employed in the production of steel.

Arc Furnaces

Arc furnaces can be conveniently divided into two types:

(1) Indirect or Arc Radiation Furnaces.

(2) Direct or Arc Resistance Furnaces.

In the *arc radiation* or indirect arc furnace, the heat is produced by an arc formed above the charge. This heat is then transferred to the charge by direct radiation from the arc and by reflection from the furnace lining. Although extensively employed for the production of iron and non-ferrous castings, this type of furnace has a very limited application in the production of steel. The general characteristics of this type of furnace are discussed in Volume 1, Chapter 12.

In the production of steel it is the *direct or arc resistance* furnace which is employed. In this type of furnace the heat is generated by the arcs formed between the electrodes and the charge and by the resistance which the charge offers to the passage of the current.

Normally a *non-conductive hearth furnace*, the general principles of which are shown in Fig. 111, is employed. This type of furnace is a development of the Heroult furnace. In such furnaces the electrodes enter the furnace chamber through the roof. The number of electrodes usually depends on the current employed. When operating on a three-phase current, three electrodes are employed, one electrode being connected with each phase. Some of the smaller furnaces operate on a single-phase current and have two electrodes.

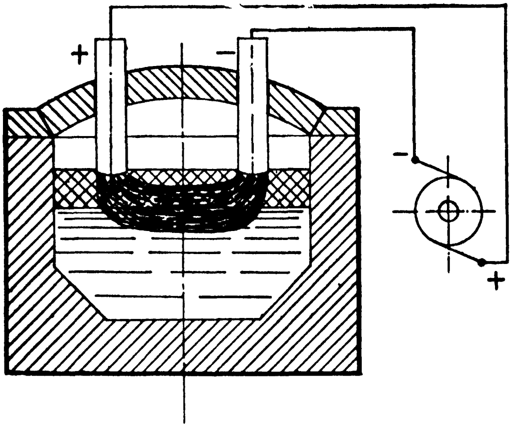

FIG. 111. Non-conductive electric arc furnace.
(By courtesy of the Staffordshire Iron and Steel Institute)

A few furnaces of the conductive-hearth type are still in use. These furnaces are a development of the Girod furnace shown in Fig. 112.

In this particular case the positive pole of the current circuit was connected to one or more carbon electrodes which entered the furnace through the roof, whilst the negative pole was connected to steel electrodes, the upper ends of which passed through the hearth and were in actual contact with the metallic bath.[2] The number of carbon electrodes above the bath varied according to the capacity of the furnace, but whatever their number they were mounted in parallel and connected to the same terminal of the circuit. The other terminal of the circuit was connected to the steel electrodes.

Since the heat is produced in these furnaces by the formation of arcs

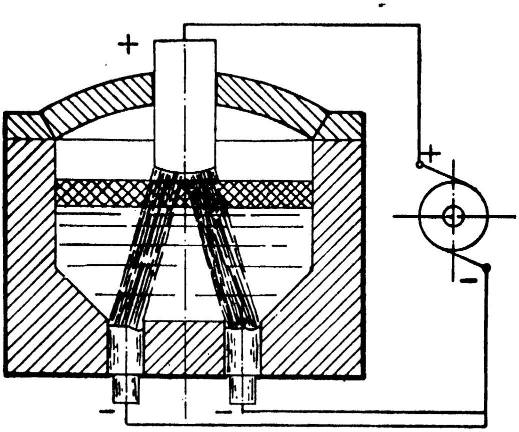

FIG. 112. Conductive electric arc furnace.
(*By courtesy of the Staffordshire Iron and Steel Institute.*)

between the electrodes and the metallic charge, and by the resistance of the charge to the passage of the current between the upper and lower electrodes, it is claimed that they offer greater efficiency for the melting of cold charges. As pointed out by Boylston,[3] the production of heat in the hearth and the introduction of conducting material into the hearth

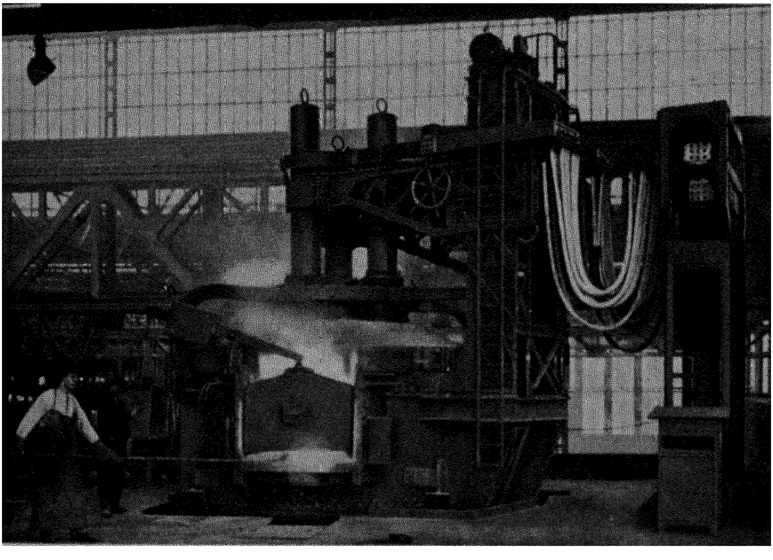

FIG. 113. A 15-ton Siemens-Halske arc furnace.
(*By courtesy of Siemens-Schuckert, Ltd.*)

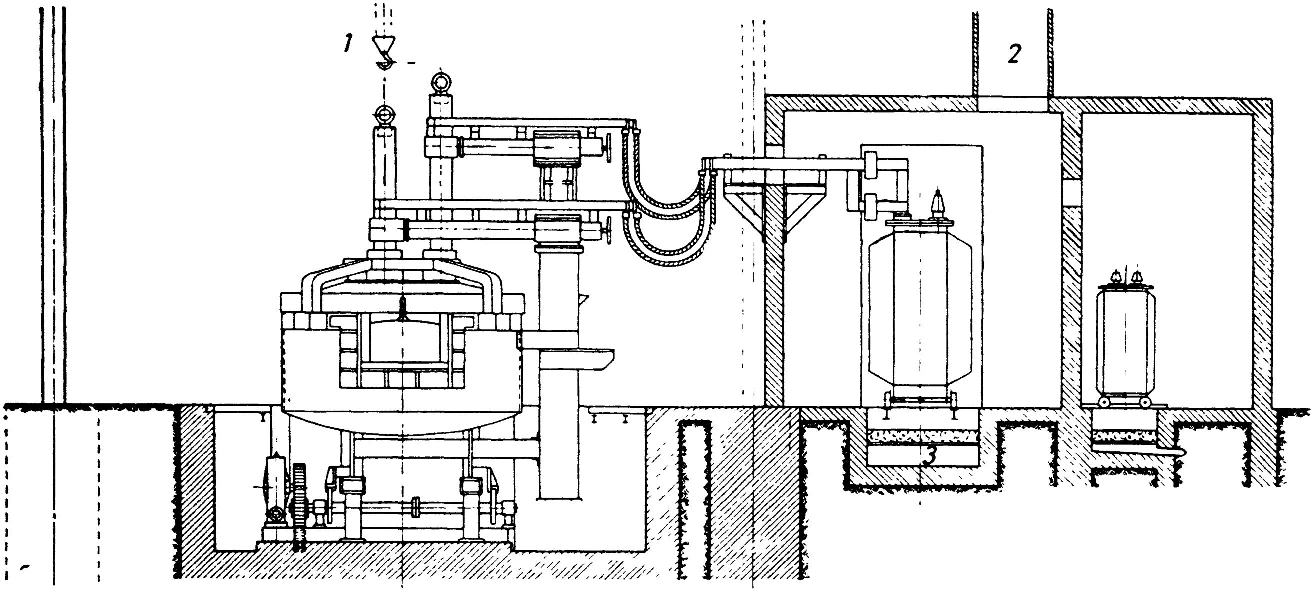

FIG. 114. Siemens-Halske furnace showing power transmission.

(By courtesy of Siemens-Schuckert, Ltd.)

(1) Crane hook
(2) Ventilation duct
(3) Fresh air inlet duct
(4) Crane travel dimension "o"
(5) Pouring door
(6) Slagging door
(7) Tilting equipment controller
(8) Transformer room
(9) Controller for electrode control gear
(10) Switchboard
(11) Instrument pillar
(12) Automatic electrode control panel
(13) Choke coil
(14) Choke coil oil circuit breaker
(15) Main oil circuit breaker
(16) Incoming feeder cubicle
(17) Terminal board

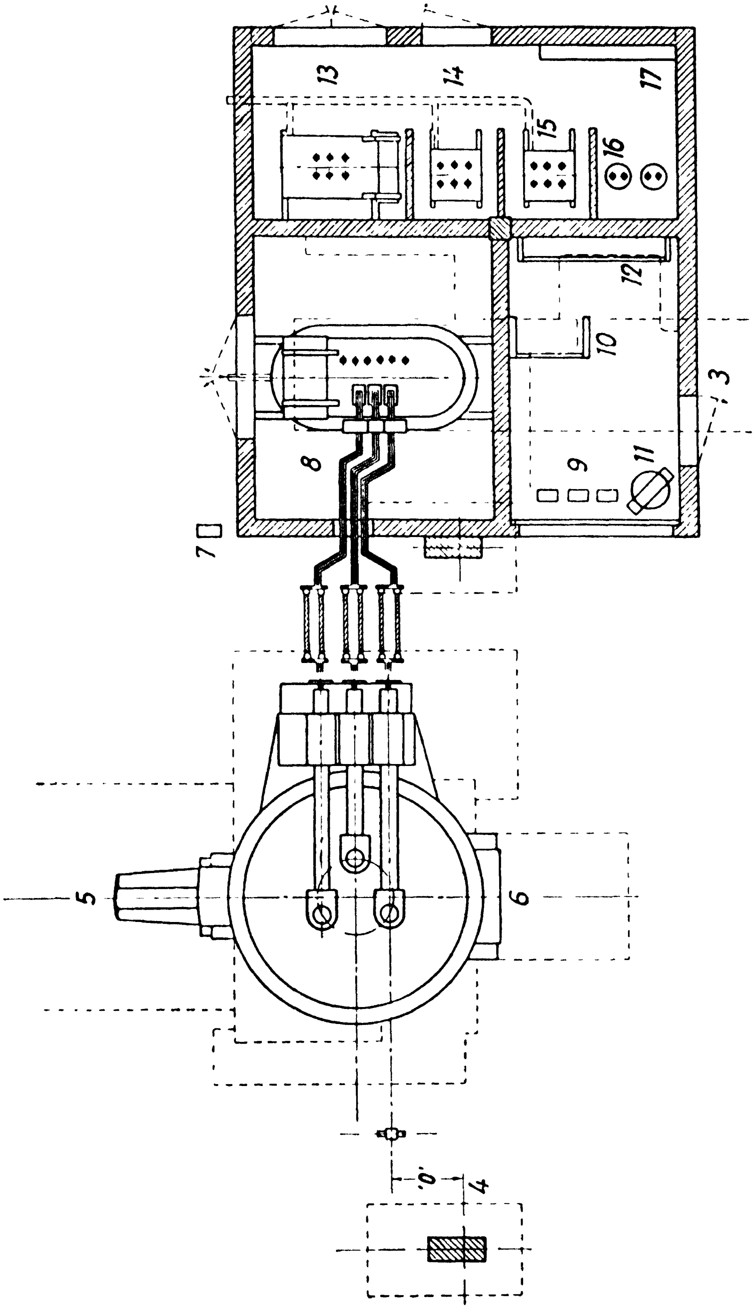

FIG. 115. Arc furnace plant layout, showing power supply.

is an unusually dangerous practice and is likely to aggravate hearth troubles.

In the Greaves Etchell furnace, which is ably described by Lester,[3] the steel electrodes are replaced by a copper bridge embedded in the furnace hearth. Lester outlines the preparation of a conductive hearth. This type of furnace, however, makes the construction and maintenance of the furnace difficult, and it has been superseded by the non-conductive-hearth type.

During recent years the development of the *non-conductive hearth* type of furnace has been rapid. They have been extensively employed in the production of steel castings and high quality alloy steels. Such furnaces vary in capacity up to 30 to 35 tons, although in America a few larger furnaces up to 80 tons capacity have been installed. For foundry work furnaces of 5 to 15 tons capacity seem to be favoured. For the production of alloy and special steel ingots, furnaces of 25 to 35 tons prove most economical.

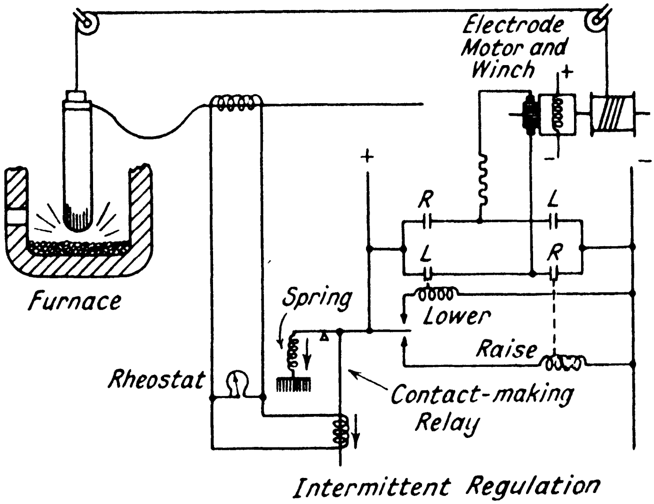

FIG. 116. Electrode-control diagram.

The furnace consists of a steel shell, lined with suitable refractory materials, mounted on tilting mechanism. This mechanism allows the furnace to be tilted through a small angle towards the front for slagging and through a larger angle in the opposite direction for tapping. The electrodes, which enter through the roof, are supported by rigidly constructed electrode holders, which permit them to be raised or lowered by hand gearing or electrical means.

The power is supplied to the furnace by a single or polyphase step-down transformer. These transformers are designed to give a voltage range, so that the voltage can be varied according to the stage of the process. During the melting of the cold stock the current is usually supplied at a voltage of 160 to 180, and this is reduced as the process progresses. In the refining stage, when less heat input is required, this voltage is frequently dropped to 95 volts. The general arrangement of a Siemens-Halske furnace is shown in Figs. 113 and 114.

The power is conveyed from the transformer to the electrodes by heavy busbars or flexible cables which permit the current to be maintained when the furnace is tilted. The general arrangement of this is shown in Figs. 114 and 115.

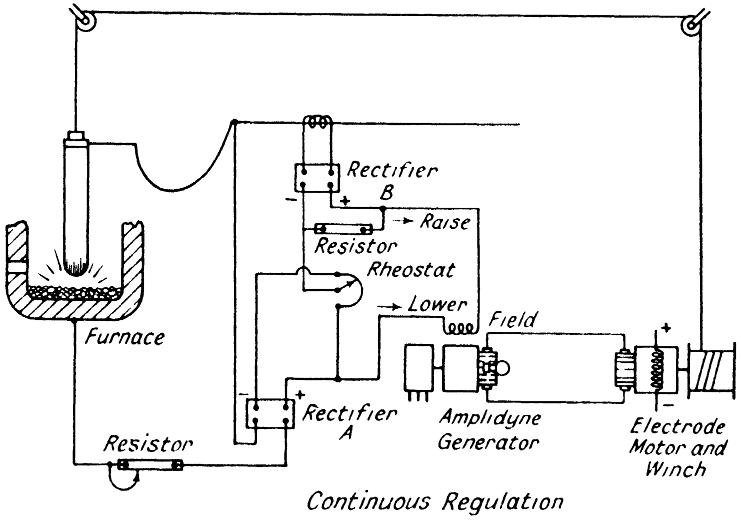

FIG. 117. Electrode-control diagram.

A very important feature of electric furnace operation is the *control of the electrodes*. In the early days the electrodes were raised and lowered as required, by hand. Sometimes these electrodes were forced down against obstructions, thereby causing excessive breakage of electrodes. Most modern furnaces are fitted with devices for the automatic control of electrodes. When the power is switched on at a given voltage, the length of the arc, and thereby the temperature, is automatically controlled by impulses relayed to the control panel. This arrangement is shown in Figs. 116 and 117

The practical application of electrode control differs slightly with the different makes of electric furnaces. In the case of the Electromelt, each electrode is suspended by a steel cable passing over sheaths to a separate electrical winch and thence to a counterweight. The arm, therefore, is practically free or floating and requires very little power to activate its movement, thereby permitting very slight adjustment to the length of the arc. The winches which raise or lower the electrodes are energized by sensitive current relays, so that each electrode is moved as required, to keep the power input to the arc at a desired value. Manual adjustment of the electrodes is also possible. One interesting feature of this arrangement is the friction grip of the cable on the winch-drum. In the event of over-running, or of any obstruction to the electrode, this friction grip is released and no further strain is put on either the motor or the electrode. The general arrangement is shown in Fig. 118.

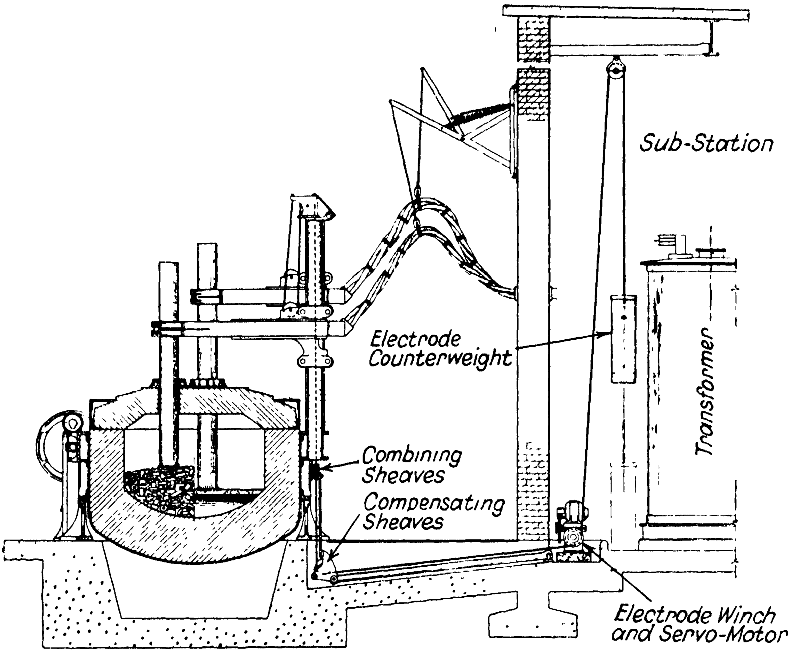

FIG. 118. " Lectromelt " electrode-control.
(*By courtesy of Birmingham Electric Fces. Ltd.*)

Naturally this control of the electrodes is of great importance from the economic aspect of electrode consumption, as breakages can be considerably reduced.

The main consumption of electrodes does not always take place at the arcing end of the electrode, but over the whole surface of the electrode within the melting chamber and up to a point some 12 in. or 15 in. above the roof. This erosion is due to the oxidizing flames or fumes ascending and passing through the space between the electrode and the opening in the roof which permits the entry of the electrode. It is not unusual to find electrodes badly tapered at this point, and many devices have been suggested to reduce this erosion. The main successful devices are those which prevent the escape of hot gases at this point, and these usually consist of a water-cooled gland which reduces the space between the electrode and the periphery of the opening.

These electrodes may be made of graphite or amorphous carbon rammed into shape. The actual consumption of electrodes varies considerably from about 14 to 30 lb. per ton. It is slightly higher in the case of amorphous carbon electrodes, which of course are cheaper.

The power consumption varies considerably with the size of the furnace and the particular practice employed. When melting cold scrap charges the power consumption may approximate 600 kWh. per ton falling to 450 kWh. per ton in larger furnaces. Additional power of course is consumed during refining, and this figure depends on the length of the refining period. In the production of ordinary grades of steel, with a one slag process, the additional power may be 150 to 180 kWh. per ton, whilst in the production of special transformer sheet steel with very low watt losses, an additional 300 to 400 kWh. per ton. There are other factors which affect this power consumption and these have been discussed in detail by C. H. von Baur[6] in an excellent paper on Heat Transfer. He[6] showed that the method of charging played a very important part. He also tested the effect of varying the power input on a 2- to 3-ton capacity Heroult furnace producing steel castings. The furnace was worked on a 3-phase current at 110 volts per phase—the inherent reactance was 10 to 12 per cent. No choke coil was used and the transformer was rated 1,200 kVA.

During the first period, which consisted of 7½ weeks, the maximum power available was put into the circuit, and the average power consumption for 272 heats was 701 kWh. per ton. These results were, however, adversely affected by a high consumption in the last three weeks, when the side walls were very badly worn, allowing excessive radiation. During the first 4½ weeks the power consumption was about 685 kWh. per ton.

In the second period, which extended over about 3 weeks, the power input was reduced considerably with the corresponding reduction in consumption. This consumption fell to about 647 kWh. per ton as shown in Fig. 119. These experiments had demonstrated that on this particular furnace it was more economic to operate at 5,000 to 6,200 amperes, than at 8,000 and 10,000 amperes. These results are shown graphically in Fig. 120. It is advisable to prove the most efficient operating conditions for each particular furnace, with due respect to the raw materials and process employed.

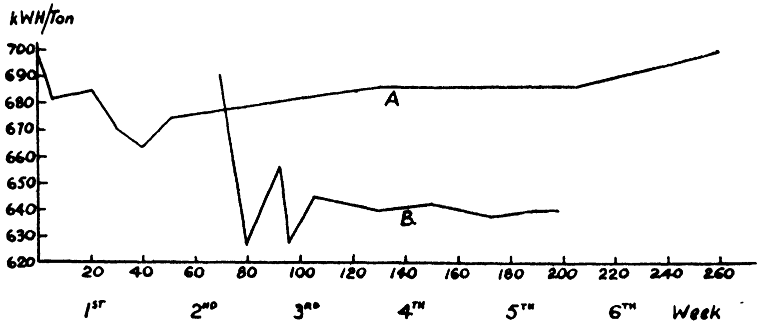

FIG. 119. Power-consumption graph.
(*By courtesy of the Staffordshire Iron and Steel Institute*)

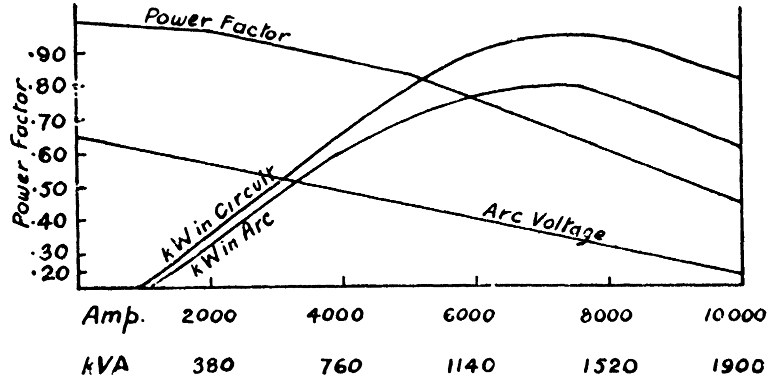

FIG. 120. Power-consumption graph.
(*By courtesy of the Staffordshire Iron and Steel Institute.*)

One peculiarity of arc furnaces is the load characteristics. During the melting down of the charge violent fluctuations occur, but when liquid metal has been formed the arcs become more regular and the load characteristics more normal. At this stage about 85 per cent. of the peak load is required. During melting a voltage of 120 to 180 volts is usually carried, but this is decreased during refining, when less heat is required, to 80 to 100 volts. The total power used during this refining period is about 25 to 30 per cent. of the peak load, and towards the end of the process this load factor may decrease to as low as 10 per cent. of the peak. Since the price paid for current is dependent on the load factor, it is obviously important to maintain the highest possible load factor. If two furnaces can be operated together so that their melting and refining is staggered, a better load factor can be obtained, as shown in Fig. 121. This Fig., taken from a paper by the author,[7] shows that by operating two furnaces together a load factor of 76 per cent. can be obtained, as compared with 63·3 per cent. when one furnace is worked independently.

In an arc furnace continuously at work, the distribution of power is approximately:

	Per cent.
To the charge	62
Electrical losses	9
To heat in cooling water ..	4
Radiation losses, etc. ..	25
	100

The effect of charging on the power consumption is discussed by C. H. von Baur[6] and Baker.[8] In order to obtain minimum consumption of power and optimum melting conditions, the larger pieces of scrap should be placed on the bottom of the hearth within the electrode circle. Upon these should be placed the medium-size scrap and a charge finished with smaller pieces. This allows more satisfactory arcing.

The rapid charging of electric furnaces has been accomplished by a removable roof. According to an article in *Steel*[9] top charging of electric furnaces was first patented in 1925, but was not really developed until 1932, since which date it has become general practice. In the older furnaces, charging was done through the door and was naturally a prolonged operation. With furnaces provided with a removable roof, the charge is loaded into a drop bottom bucket, which is lowered into the furnace after the roof has been removed, so that the furnace is charged in minimum time. With a removable roof, or a sliding hearth, it is possible to charge a 10- to 15-ton furnace in from 5 to 15 minutes. This development has resulted in great economy.

In America the charging of two 25-ton furnaces, one provided with a removable roof and the other designed for door charging, demonstrates the advantage of top charging. The adoption of top charging has

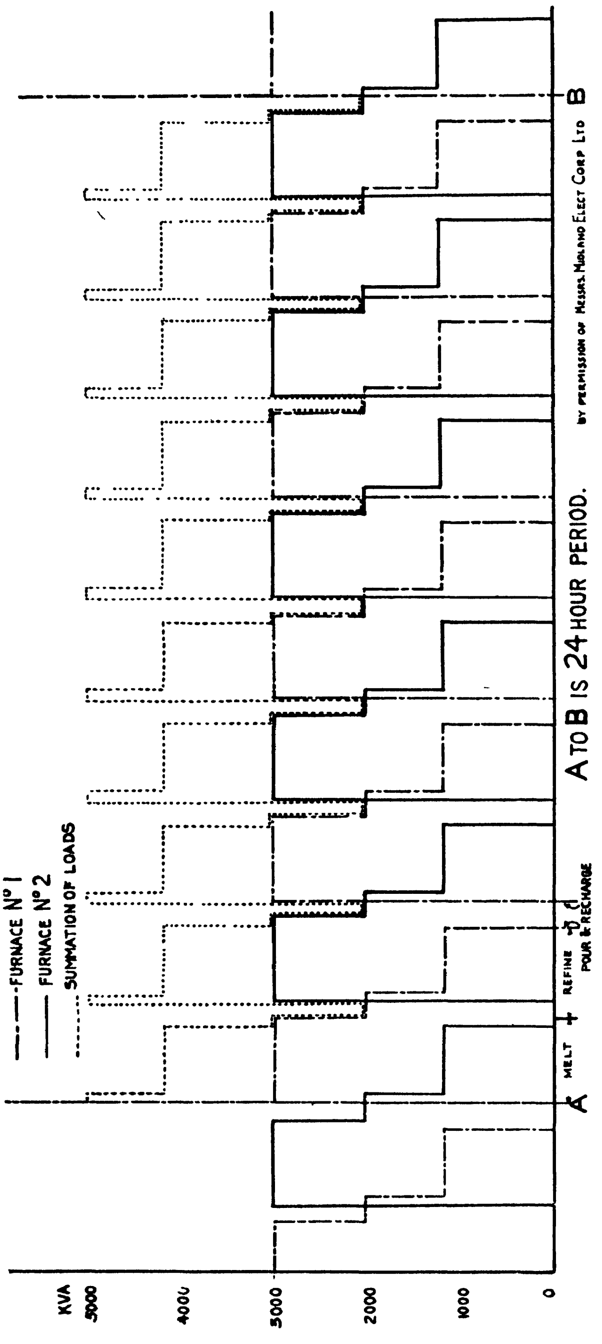

FIG. 121. Load factor diagram.

(By courtesy of the Staffordshire Iron and Steel Institute.)

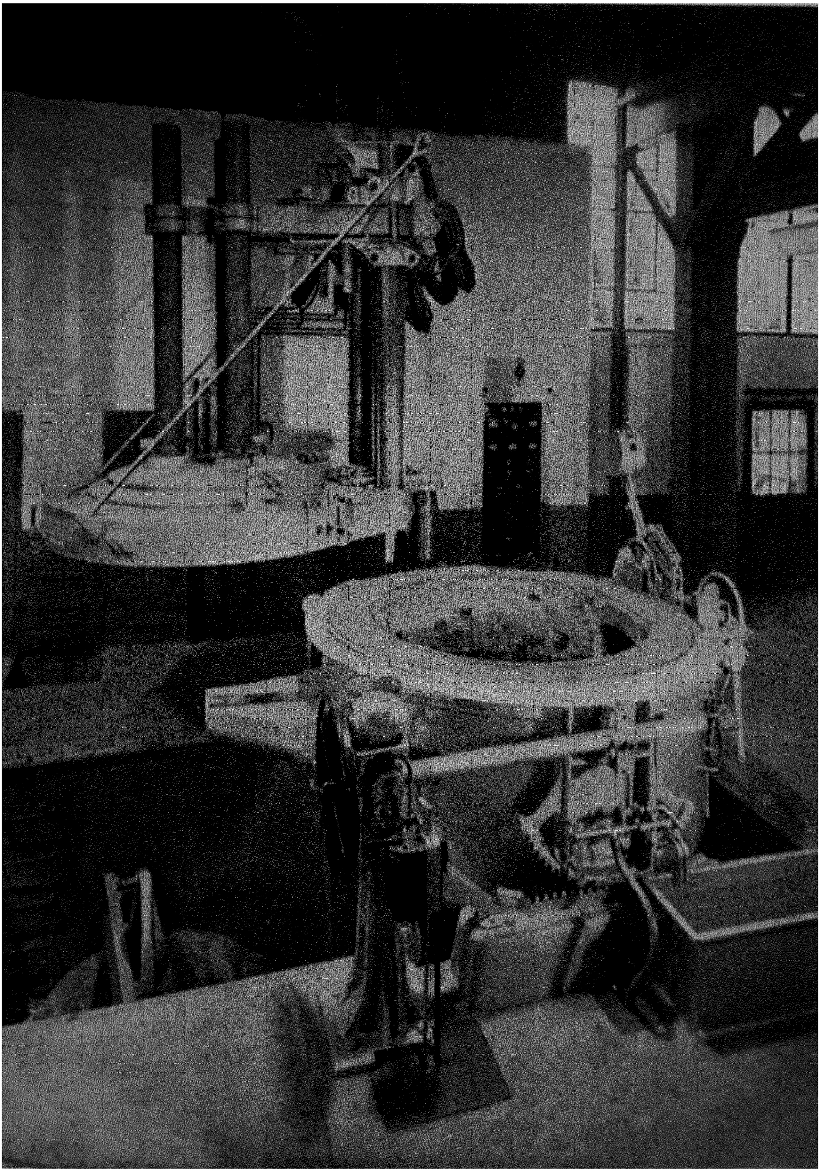

FIG. 122. Swing-roof type electric arc furnace.

(By courtesy of Birmingham Electric Fces. Ltd.)

resulted in an increased production of 30 per cent., whilst the refractory costs decreased 22 per cent., the electrode consumption 15 per cent. and the power consumption was reduced from 670 to 625 kWh. per ton.

Three methods of top charging are available:

(1) The swing roof, as shown in Fig. 122, where the roof swings round so as to enable the drop bottom bucket to be inserted.

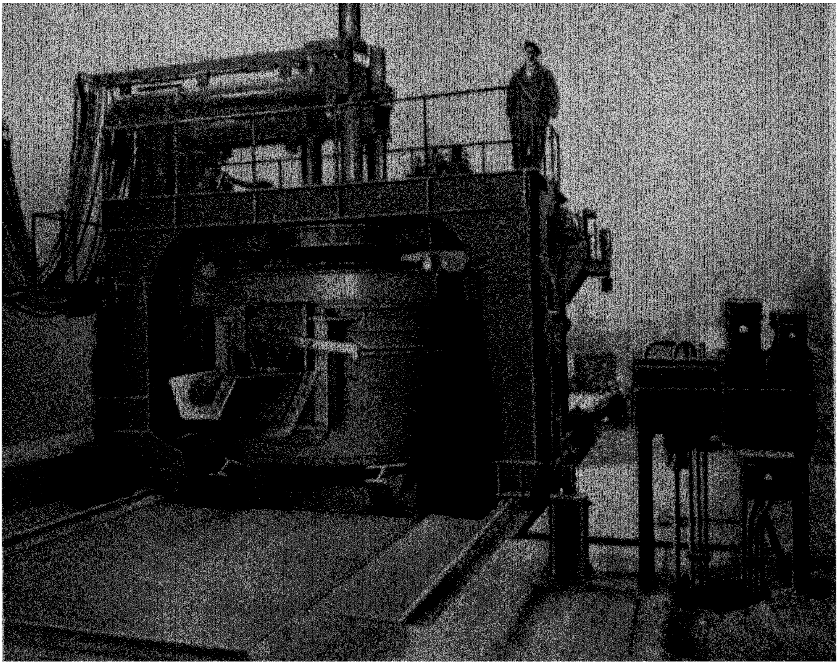

FIG. 123. A 12-ton Siemens-Halske arc furnace with removable cover.
(By courtesy of Siemens-Schuckert, Ltd.)

(2) The sliding roof, in which the roof slides to one side allowing the same method of charging (Fig. 123).

(3) The sliding hearth, in which the furnace is provided with a fixed roof but the hearth is designed to slide away fron the roof. It has been suggested that towards the end of its useful life the roof tends to become weak and some danger may result in its removal for top charging. This disadvantage is overcome in the sliding or removable hearth furnaces.

Arc Furnace Melting Practice

There are two methods of operating the arc furnace:

(1) The basic process.
(2) The acid process.

The basic process, which is carried out in a basic lined furnace, is used for the production of high grade tool steel, alloy and special steels in ingot form. The basic process is also used for the production of steel castings, whilst the acid process operating with a silica-lined hearth is used chiefly for the production of steel castings.

Basic Process

In the basic process a wide range of scrap can be employed. The normal procedure is to melt scrap, which is recarburized with coke, anthracite or broken electrodes, so as to give sufficient carbon at the "melt out" to assure a good carbon boil. As a rule no pig iron is employed.

For the basic process the furnace is lined as follows:

Roof

The roof is usually constructed with silica bricks, although some works prefer a sillimanite roof. At one particular works a silica roof gave a life of 126 heats compared with 190 heats from a sillimanite roof. In some cases a 4½ in. course of chrome-magnesite bricks is included on the outer circle of the roof, which is adjacent to a chrome-magnesite or magnesite skew-back. This reduces the erosion of the basic side walls by silica drippings from the roof.

Recently an extensive investigation has been carried out by electric arc furnace roofs and this has been the subject of a special report.[10] The data collected demonstrate that the fixed domed type of roof has a longer life than the flat centred, movable roofs, which fail frequently, due to spalling. It is shown that the other factors which affect the roof life are:

(1) Nature of the refractories employed, and this aspect will be discussed in more detail in Chapter 16, dealing with refractory materials.
(2) The expansion allowance and rise of the roof. Undoubtedly the stresses set up in the roof structure have an important bearing on its life. In the report it is recommended that whenever permissible the domed roof should be adopted. The best thickness

appears to be twelve inches, although nine-inch roofs may be preferred on smaller furnaces. It also points out that the use of smaller, special shaped bricks enables the more uniform distribution of expansion joints, which renders the roof more elastic and better able to accommodate local stresses.

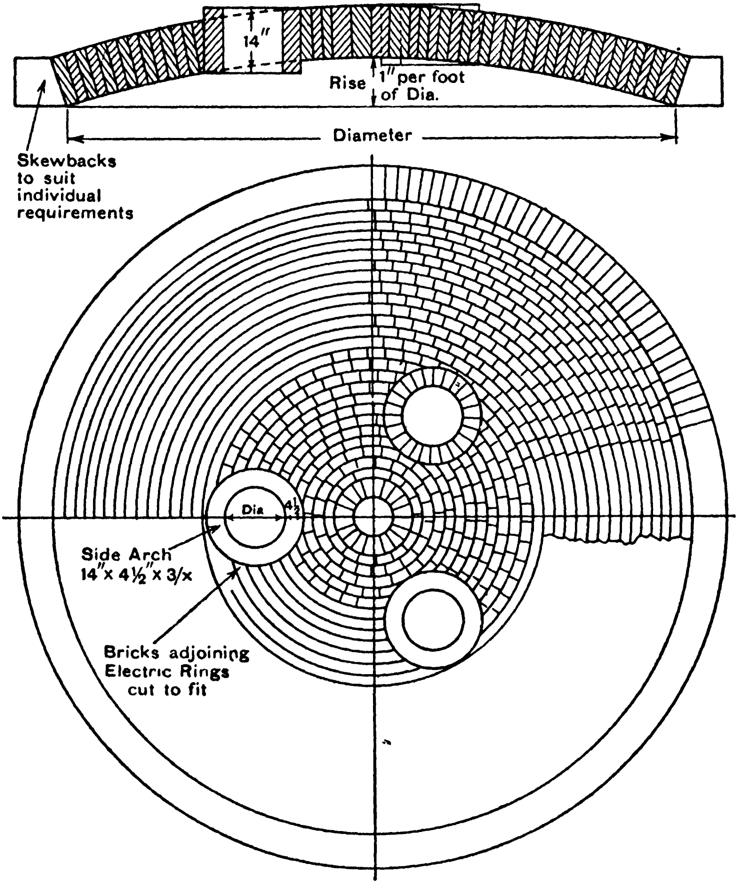

FIG. 124. The bricking arrangement of recommended roof design.
(By courtesy of the Iron and Steel Institute.)

It appears that the allowance made for expansion varied between $\frac{1}{16}$ to almost $\frac{1}{4}$ in. per ft., but the committee recommends an allowance of $\frac{1}{8}$ in. per ft. A roof rise of 1 in. per ft. diameter is recommended. The general arrangement of the recommended roof design is shown in Fig. 124.

(3) The rate of initial warming up.

(4) The power input which affects the temperature attained during melting and refining.

Side Walls

The side walls of basic electric furnaces are still bricked with silica bricks at many works, although the modern tendency is to use magnesite bricks. Chrome magnesite bricks have been introduced for side walls, but the possibility of the chromite being reduced has deterred their employment. According to Brown[11] in America the general tendency is to line the lower side walls with magnesite, whilst the upper side walls and the roof are constructed with silica.

Hearth

The general practice is to brick the hearth shell with magnesite bricks, although stabilized dolomite bricks are frequently employed. This brickwork should be thoroughly dried before making the working hearth. The working hearth may be prepared with either magnesite or dolomite in a manner similar to that described in connection with open-hearth furnaces in Chapter 8.

When the bottom has been rammed to shape, it is dried for about twenty-four hours with a coal fire. After drying thoroughly the furnace is cleaned out and charged with coke to slag line. The arcs are then struck on the coke and the furnace brought up to a sufficiently high temperature to assure the complete fritting of the hearth. This fritting of the hearth will require about 36 hours, and the maximum temperature permissible should be maintained.

After the hearth has been well burnt, it is sound practice to slag the banks and bottom before proceeding with normal production. In a 12-ton capacity furnace a charge consisting of about 2 tons of turnings, 1 to 2 cwt. of lime and a small quantity of iron ore is melted, and this liquid is splashed over the bottom and banks. Some of this charge will be absorbed by the hearth, due to the spaces between the magnesite or dolomite grains. If the hearth has been thoroughly rammed and correctly fritted, between 25 and 30 cwt. will remain. This surplus metal will be tapped through the tap hole by tilting the furnace.

The contour of the hearth is of supreme importance. When the furnace is tilted in the tapping position, the hearth should slope towards the tap hole. This feature prevents slag or metal, which may have a corrosive action on the hearth, being left in the furnace after tapping.

According to Farnsworth and Johnson[12] many American plants employ a mixture of 80 per cent. graded periclase grain and 20 per cent. of finely ground Austrian magnesite, to which is added sodium silicate as a binder. The mixture is graded as follows:

20 per cent. $\frac{1}{2}$ to $\frac{1}{4}$ in. mesh periclase grain	90 per cent. MgO
20 per cent. $\frac{1}{4}$ to $\frac{1}{8}$ in. mesh periclase grain	90 per cent. MgO
40 per cent. $\frac{1}{8}$ in. to fines periclase grain	90 per cent. MgO
20 per cent. $\frac{1}{16}$ in. to fines dead burnt Austrian magnesite	83 per cent. MgO

This mixture requires five quarts of liquid silicate of soda per cwt. to give good plastic ramming. It is rammed in 2 to 3 in. layers until the desired contour and thickness is attained, after which it is dried with a wood fire for 24 hours. After drying, the arcs are struck and the furnace is raised as quickly as possible to a maximum temperature, at which it is maintained for at least twenty-four hours. At the end of this period the hearth should be sintered to a depth of 4 to 5 in. and is ready for immediate operation.

One of the latest developments in the preparation of the working hearth is the use of "graphitized dolomite". This material consists of dolomite ground to flour and mixed with about 1 per cent. of graphite. The mixture is tramped into position with rammers and when consolidated the furnace is charged. During the melting of the first charge the dolomite is sintered into a monolithic mass. Several highly successful hearths have been installed in this way. The practice is covered by patent rights.[13]

The charge consists of steel scrap, lime and/or limestone and a carburizer. The material may be charged by mechanical chargers through the furnace door as in the case of the open-hearth process; but more frequently basket charging in conjunction with a movable roof is adopted.

During the *melting down* pools of liquid are formed immediately beneath the electrodes. The rate of melting can be accelerated by taking power off for a few minutes and pulling the unmelted material into these pools of liquid. Some furnaces are designed so that the hearth may be rotated slightly. In furnaces provided with a rotating hearth, as soon as the pools of liquid metal have been formed, the electrodes are raised and the hearth is rotated a few degrees. The arcs are then struck on the unmelted material.

The rate of melting is also affected by the lime and/or limestone charged. This material should either be placed on the bottom or around he banks. It should not be in the vicinity of the electrodes, otherwise t will interfere with the arcing. Some prefer not to charge lime, but to add it as required during refining or towards the end of the melting down. This practice, however, may cause excessive erosion of the hearth, due to the acid oxides formed during melting down. It would appear to be beneficial to add sufficient lime with the charge to satisfy these oxides and to make further additions as required during refining.

The *refining* can be conveniently divided into two stages:

(1) Oxidation.
(2) Deoxidation.

During the oxidation stage phosphorus and silicon are removed by the same reactions which take place in the basic open-hearth process. Their removal demands oxidizing conditions and a basic slag. Since fluorspar is freely used most of the phosphorus exists in the slag as fluorapatite $3(3CaO\ P_2O_5)\ CaF_2$ (see Chapter 8).

According to Brown[11] petrographic analysis of melt down or oxidizing slags, at the end of refining and immediately before removal, shows them to contain di-calcic silicate ($2CaO\ SiO_2$), a solid solution phase consisting of the magnesium oxide, ferrous oxide and manganese oxide ($MgO . FeO . MnO$), undissolved magnesium oxide (MgO) magnetite (Fe_3O_4) and in some cases di-calcium ferrite ($2CaO . Fe_2O_3$). Typical analyses of these oxidizing slags are given in Table XXXIV.

In order to assure satisfactory elimination of the impurities and control of oxidation, it is essential that the charge should melt out with 0·3 to 0·4 per cent. more carbon than is required in the finished steel. The removal of 0·3 to 0·4 per cent. of carbon promotes the necessary boil which assists the removal of impurities, control of oxidation, good slag-metal contact, control of occluded gases and heat transfer.

When the phosphorus has been reduced to the requisite limits, the oxidizing slag is removed. As much as possible is run off through the slag-notch provided in the sill-plate of the charging door. It is essential that the whole of this oxidizing or black slag is removed. Its complete removal cannot be achieved by pouring, so when as much as possible has been removed in this way, the power is taken off, which causes the remaining slag to congeal. This congealed slag is then raked off.

After complete removal of this black slag, the power is put on, and the reducing or white slag is formed. Under this slag, typical analyses

TABLE XXXIV

Basic Electric Refining and Finishing Slags

Source		CaO	SiO_2	FeO	Fe_2O_3	Al_2O_3	MgO	MnO	P_2O_5	S	CaC_2	CaF_2
Walther . .	O	48·78	23·40	8·93	1·93	1·50	7·10	5·26	0·24	0·17	—	1·20
	F	60·62	24·54	0·29	Nil	3·08	4·24	0·09	0·07	C = 0·77 0·36	2·54	2·80
Brown . .	O	48·7	19·3	12·8	3·25	1·50	7·10	6·50	0·34	0·19	—	—
	F	61·5	24·5	0·43	Nil	2·76	4·34	0·09	—	C = 0·67 0·41	3·54	1·80
	O	38·59	10·52	18·16	6·44	2·20	8·00	9·28	0·59	0·14	—	—
	F	63·74	20·19	0·58	Nil	0·30	8·25	0·10	Nil	0·32	2·00	—

O = oxidizing or first slag.
F = reducing or second slag.

of which are given in Table XXXIV, the metal is deoxidized and the sulphur reduced to very low limits. Although some desulphurization takes place under the oxidizing slag, the final desulphurization is only possible with reducing conditions.

The removal of sulphur depends on the reaction:

$$CaO + FeS \rightleftharpoons CaS + FeO$$

which has been discussed in Chapter 8, Page 192.

The reducing conditions prevailing during the final stages of the basic arc process, however, cause the velocity of the reaction to be accelerated in the direction of the upper arrow. With a carbide slag the following reactions may take place:

$$CaO + MnS + C \rightarrow CaS + Mn + CO$$

$$CaC_2 + 3FeS + 2CaO \rightarrow 3Fe + 3CaS + 2CO$$

These, or similar reactions, result in the attainment of very low sulphurs.

The precise nature of the *reducing slag* depends to some extent on the type of steel being produced. These slags and their formation have been discussed by Brown,[11] Farnsworth and Johnson[12] and Walthir,[14] and may be classified as follows:

(1) The lime-silicon slag which is used extensively in the production of low carbon steels where carbon pick-up must be avoided. This slag can be formed by appropriate ferro-silicon and lime additions.

(2) The lime-aluminium slag, which is a very powerful deoxidizer.

(3) The lime-silica slag, which is composed chiefly of lime and sand. The value of this slag depends on the lime-silica ratio.

(4) The calcium-carbide slag, which is most frequently employed and which when correctly made is a powerful deoxidizer.

There are several methods of making up the calcium-carbide slag. Boylston[3] suggests that the mixture should be six parts of lime, two to three parts of coke and one to two parts of fluorspar. The mixture is varied slightly at different plants to meet local practice and conditions. Providing that the metal has not been over-oxidized during the refining or oxidizing stage, the author has found that the following proportions usually prove adequate:

Lime	50 lb. per ton.
Fluorspar	10 lb. per ton.
Sand	7 lb. per ton.
Coke	10 lb. per ton

At the temperature of the electric arc this mixture forms calcium carbide, thus:

$$CaO + 3C \rightleftharpoons CaC_2 + CO$$

The maintenance of a truly carbide slag may require further coke additions, but the necessity for such additions can be readily judged. When cooled and damped with water the slag should give a distinctive carbide smell. A true carbide slag should be white and falling. Typical analyses are given in Table XXXIV.

According to Walthir[14], efficient slag control during the finishing stage depends on:

(1) Moisture-free materials.
(2) Consistent chemical and physical properties of the slag forming materials.
(3) Correct bath temperature.
(4) Quick fluxing action.
(5) Correct slag volume and mixture.
(6) Power input.
(7) Time factor. It usually requires 1½ to 2½ hours for the deoxidizing slag to complete its work.
(8) Bottom upkeep. Hearth trouble may seriously interfere with the deoxidizing stage.

Correct bath temperature is of supreme importance. McQuaid[15] suggests that at the end of the oxidation period the steel should be increasing rapidly in temperature, approaching at least 1,600° C. He adds that the carbon should be about 10 points below the desired minimum, the silicon should be nearly zero and the residual manganese should be approximately 0·20 per cent.

The time factor is important and is sometimes standardized. The work of the reducing slag can be greatly accelerated by stirring the bath with iron rabbles, thereby reducing the time necessary to complete deoxidation and degasification.

Table XXXV shows a typical log of a 70-cwt. "Lectomelt" furnace producing ordinary mild steel, whilst Table XXXVI is the typical operation of a 15-ton basic lined furnace.

The furnace discussed in Table XXXVI has a nominal capacity of 15 tons and a rating of 4,500 kVA. It is provided with a motor-operated tap-change gear on the high-tension side of the transformer so as to

TABLE XXXV

Log of 3½-ton Basic Arc Furnace

Top Charged—Transformer Rating, 1,500 kVA.—Voltage Tapping in use.

No.1 = 95 V. No. 2 = 115 V. No.3 = 148 V. No.4 = 185 V.

Heat No. 900—19/6/40. *Metal—soft* 0·20/0·25 C, 0·70 Mn.

Time

3.58 Charged—Foundry discards 1,680 lb. Heavy turnings 2,240 lb. Short heavy steel 3,472 lb. Anthracite 8 lb.

4.00 Power on No. 4 tap—approx. 1,400—1,650 kW.

4.35 Power off, changed to No. 3 tap—charged 50 lb. limestone.

4.36 Power on.

4.45 Power off. Pulled in unmelted scrap, added 30 lb. haematite ore.

4.47 Power on—115 V. Added 50 lb. limestone. 20 lb. lime. 20 lb. fluorspar.

5.00 Power off—changed to No. 1 tap. 95 V.—bath clear melted and boiling.

5.10 Steel sample taken, forged, cooled and fractured. From fracture carbon estimated to be low enough for slag removal. Sent sample to laboratory for analysis.

5.15 Power off—slagging.

5.18 Power on—No. 1 tap—95 V.—charged 40 lb. silico-manganese (65 per cent. Mn, 22 per cent. Si). 68 lb. limestone. 44 lb. lime, 30 lb. fluorspar. 10 lb. sand.

5.23 Slag fused. Added 15 lb. ground anthracite. 8 lb. powdered ferro-silicon (47 per cent.). Analysis results reported 0·19 C, 0·27 Mn.

5.30 Slag and steel samples taken. Slag—white and falling. Steel poured into sand mould and allowed to cool with anti-piping compound on top. Examined for piping—O.K.

5.35 Added 10 lb. silico-manganese. 30 lb. ferro-silicon (45 per cent.). 8 lb. slag mix (approx. 4 lb. lime, 1 lb. anthracite, 1 lb. FeSi and 1 lb. fluorspar to keep atmosphere good.)

5.37 Steel and slag sample—temperature O.K. Slag carbide.

5.40 Power off—tapping—2½ lb. aluminium to ladle. Analysis of ladle sample; 0·22 C, 0·69 Mn, 0·028 S, 0·019 P, 0·39 Si.
Castings poured—machine moulded green sand work.

Details of charge, etc.

Total metal charged, including ferro alloys = 7,472 lb.
Yield of steel (poured in foundry) = 7,050 lb.

Current consumption	1,900 kWh.
kWh. per ton charged	571
kWh. per ton of steel	603

Time table

Total time—power on to tap	1 hr. 40 min.
Time taken to charge (not included in above item)	0 hr. 02 min.
Power to melt	1 hr. 00 min.
Boiling	0 hr. 15 min.
Refining	0 hr. 22 min.

TABLE XXXVI

Typical Operation of a 15-ton Basic Lined Arc Furnace

Furnace fettled after previous heat and 5 cwt. of lime charged on the bottom.
Furnace basket-charged through removable roof.

Charge:	Steel scrap	15 tons.
	Broken electrodes	36 lb.

Melting cycle

A.M.

8.00 Furnace charged. 156 volts—15,000 amps.

8.45 Power off ten minutes—pulling down unmelted scrap.

9.53 Voltage reduced to 140—15,000 amps.

10.00 Lime and fluorspar additions made.

10.20 Power reduced—10,000 amps. at 140 volts. Lime addition made.

10.45 Charge melted. Voltage reduced to 102—10,000 amps. Sample taken and when carbon low enough the furnace is slagged.

11.20 Power on after slagging at 90 volts—10,000 amps. Ferro-silicon added. Lime for second slag added.

P.M.

12.50 Voltage reduced to 80—8,000 amps. Slag mixture (approximately 4 parts of fine coal and 3 parts of fluorspar) added. Carbide slag maintained by coal addition if and as required.

1.45 Furnace tapped.

Power consumption	763 kWh. per ton.
Electrode consumption	20 lb. per ton.

supply current to the electrodes at eight different voltages—80, 90, 102, 115, 140, 156, 176 and 200. The power input to the furnace is proportional to the voltage selected. The load input to the furnace can be varied by altering the low-tension amps. flowing through the electrodes, and this depends on the distance between the end of the electrode and the metallic bath. This distance can be controlled either by hand control of the electrode motors or by the automatic control provided by the contactor panel.

On a hot furnace melting is frequently commenced with a voltage of 176, although 200 volts can be employed. This higher voltage necessitates greater care to avoid damage to the furnace structure. When operating on a cold furnace it is advisable to start the melt with a lower voltage, which is maintained until the electrodes are somewhat buried in the charge.

The efficient operation of the voltage relays on the contactor panel depends on the voltage drop between the electrodes and the charge.

This drop is obtained by means of bars inserted in the rammed lining, to which electrical connections are made. When the hearth is cold, however, the dolomite will not be electrically conductive, but the difficulty can be overcome by inserting a thin steel rod into the charge and connecting it to the door plate. This will provide an effective earth. This automatic control has been recently discussed in detail by Lewis[16] and the general principles are shown in Figs. 116 and 117.

The control of *hydrogen and occluded gas* is an important feature of high-grade steel production. The sources of hydrogen according to German[17] are:

(1) Wet and rusty scrap.
(2) Wet slag-forming materials.
(3) Moisture in the furnace atmosphere.
(4) Moisture arising from leaking electrode holders, etc. This moisture is split up according to the reaction:

$$H_2O + Fe \rightleftharpoons H_2 + FeO.$$

The hydrogen thus formed is dissolved or occluded by the bath. The absorption of hydrogen by the bath is apparently accelerated in the vicinity of the arcs, where atomic hydrogen is probably formed.

The amount of hydrogen absorbed increases with the hydrogen available in the furnace chamber, the atmospheric pressure within the furnace and the temperature of the metal. Molybdenum, nickel and chromium also affect the amount of hydrogen absorbed in this way.

The danger of hydrogen absorption, however, can be greatly reduced by the selection of raw materials, and the provision of a vigorous carbon boil. Steel produced under vigorous boil contains less hydrogen than steels produced with a weak boil. Hydrogen and nitrogen tend to diffuse into the carbon monoxide bubbles " until the partial pressure of each gas in the bubble is equal to the partial pressure of gas in the steel, and the bubble rising to the surface of the slag will carry with it, not only carbon monoxide, but also any other gases which are in solution in the metal, in the ratio of their concentrations in the bath".[17] This opinion stresses the importance of the carbon boil and the careful selection of any materials used after the carbon boil has ceased.

The basic-lined arc furnace is sometimes operated on a *single slag practice*. In some cases this practice merely consists of melting steel scrap, with a little coke, and refining it under an oxidizing slag in a manner similar to the basic open-hearth process. When the steel has been refined to the necessary limits it is tapped into the casting ladle, to which the necessary deoxidizers are added. Robiette[19] describes the operation of a 10-ton Birlec furnace in which the charge consisted of steel scrap plus a small quantity of recarburizer. This charge was melted in about one hour and subsequently refined with ore and limestone additions under a black oxidizing slag. Samples of the bath were taken to ascertain the progress of refining and when the bath was within the required limits the furnace was tapped and the necessary carbon and alloy additions were made to the ladle. The time taken varied between one hour thirty-five minutes and one hour forty-five minutes per heat. This furnace was producing 140 tons per twenty-four hours with a power consumption of 570 kWh. per ton and an electrode consumption of 6½ lb. per ton. The steel was teemed into small ingot moulds, varying from 3 in. × 3 in. to 6 in. × 6 in. in section, and the ingots were subsequently rolled into billets.

When reclaiming alloys from steel scrap, it is the normal practice to employ a *reducing slag only*. Little, if any, iron ore is added to the charge. The oxide adhering to the scrap is usually sufficient to oxidize any silicon present, and the silica thus formed is neutralized by lime additions. A reducing slag is then formed by ferro-silicon, coke and lime additions, and these deoxidizing conditions result in any oxidizable alloys being reduced to the metallic bath. These conditions, however, cause any phosphorus in the slag to be reduced and returned to the metal. When adopting this practice, therefore, all raw materials must be low in this element, otherwise the phosphorus in the finished steel will exceed the required limits. The practice is extensively employed for the production of alloy steels containing chromium, vanadium and other oxidizable alloys from alloy steel scrap.

Phelps[21] suggests that melting with as little oxidation as possible and changing the melt-down slag into a deoxidizing slag without removal is probably slower than the straight two-slag process, but 100 per cent. recovery of the alloy content of the scrap charged can be obtained. This consideration is of supreme importance when producing high-speed tool steel from such materials as 14 per cent. tungsten scrap, where a high alloy recovery is essential.

Large quantities of alloy steels are, however, produced by the two-slag process. In this case oxidizable alloys such as chromium, vanadium, etc., are added to the bath in the form of ferro-alloys, towards the end of the deoxidizing stage. The recovery of the alloys from these additions should be high. According to Nelson,[22] on a double-slag process, in which the FeO in the slag is under 2·0 per cent., a 95 per cent. recovery of manganese, chromium, silicon and vanadium can be expected, when these additions are made to the furnace.

Nelson[22] also states that under a single oxidizing slag the alloy recoveries are similar to those obtained in good open-hearth practice. Open-hearth alloy recoveries, depending on the carbon content of the bath and other factors, he[22] quotes as:

55 per cent. to 75 per cent. manganese recovery when most of the manganese is added to the bath.
65 per cent. to 85 per cent. manganese recovery from ladle additions.
70 per cent. to 80 per cent. chromium recovery when added to the bath.
75 per cent. to 90 per cent. silicon recovery from ladle additions.
75 per cent. to 85 per cent. vanadium recovery from ladle additions.

Nickel and molybdenum can be added at any stage, and are frequently recovered from scrap containing these elements. There is no loss of alloy content. Nickel can also be added at any stage as metallic nickel with 100 per cent. recovery. Sometimes molybdenum is added to the charge as calcium molybdate (MnO_3 – 69 per cent. and CaO – 19 per cent.) or as ferro-molybdenum (74 per cent. Mo) towards the end of refining.

Tables XXXVII and XXXVIII show the details of the manufacture of two typical alloy-steel heats under a two-slag and a one-slag process respectively.

TABLE XXXVII

ARC-FURNACE RECORD OF NORMAL ALLOY HEAT. (15-TON FURNACE)

Analysis ordered: C, 0·15 to 0·20 per cent.; Mn, 0·40 to 0·60 per cent ; S, 0·025 per cent.; P, 0·025 per cent.; Si, 0·15 to 0·30 per cent.; Cr, 0·45 to 0·75 per cent.; Ni, 1·00 to 1·50 per cent.

A.M.
8.30 Previous heat tapped.
8·40 Began charging: 36,000 lb. common scrap.
265 lb. nickel.
120 lb. anthracite coal.
9.10 Finished charging; power on high voltage.
10.10 Add lime, 300 lb. Intermediate voltage.
11.00 Add lime, 300 lb. Low voltage.
11.15 Add iron ore, 200 lb.; nearly melted.
11.25 Add iron ore, 250 lb; completely melted.
11.40 Bath stirred. Carbometer test 0·06 per cent. C.
11.42 First steel and slag test.
11.45 Start slag off.

NOON
12.00 Finish slag off.

TABLE XXXVII—*continued.*

P.M.

12.08 Refining slag mix added: 800 lb. lime.
150 lb. fluorspar.
100 lb. sand.
90 lb. coke.

12.20 First steel test reported: 0·16 per cent. Mn, 0·012 per cent. P, 0·77 per cent. Ni, 0·04 per cent. Cr.

12.30 Add coke, 90 lb.

12.45 Bath stirred; slag carbide.

12.50 Second steel and slag test.

12.55 Add coke, 50 lb.

1.10 Stir bath; temperature test.

1.15 Add chromium, 270 lb.: 70·00 per cent. Cr, 4·60 per cent. C.

1.20 Add coke, 40 lb.

1.30 Second steel test reported: 0·08 per cent. C, 0·15 per cent. Mn, 0·026 per cent. S, 1·24 per cent. Ni, 0·05 per cent. Cr.

1.35 Stir bath; temperature test.

1.40 Add manganese, 182 lb.: 80·00 per cent. Mn, 6·70 per cent. C.

1.40 Add silicon, 106 lb.: 76·00 per cent. Si.

1.55 Power off; stir bath; temperature test.

2.00 Tapped.

LADLE ANALYSIS

0·17 per cent. C, 0·57 per cent. Mn, 0·013 per cent. P, 0·005 per cent. S, 0·22 per cent. Si, 1·22 per cent. Ni, 0·57 per cent. Cr.

SLAG ANALYSIS

	CaO	SiO_2	FeO	Fe_2O_3	Al_2O_3	MgO	MnO	P_2O_5	S	CaC_2
First	38·59	10·52	18·16	6·44	2·20	8·00	9·29	0·59	0·14	—
Second	60·86	17·30	0·91	Nil	0·72	4·10	0·40	Nil	0·31	2·40
Tapping	63·74	20·19	0·58	Nil	0·30	8·25	0·10	—	0·32	2·00

TABLE XXXVIII

ARC-FURNACE RECORD OF A 15-TON STAINLESS-STEEL HEAT WITH STAINLESS-SCRAP ADDITIONS

(Farnsworthy & Johnson[12])

Analysis ordered: C, 0·10 per cent. (max.); Mn, 0·30 to 0·50 per cent.; S, 0·025 per cent.; P, 0·025 per cent.; Si, 0·30 to 0·50 per cent.; Cr, 16·50 per cent,

A.M.

8.10 Previous heat tapped.

8.20 Began charging: 13,000 lb. ingot iron scrap: 0·03 per cent. C.
19,000 lb. stainless scrap: 0·09 per cent. C, 17·20 per cent. Cr.
400 lb. iron ore.

TABLE XXXVIII—*continued.*

A.M.
8.50 Finish charging. Power on high voltage.
9.05 Add slag mix: 300 lb. lime.
100 lb. fluorspar.
100 lb. sand.
9.50 Add slag mix: 300 lb. lime. Intermediate voltage.
100 lb. fluorspar.
100 lb. sand.
10.25 Nearly melted. Low voltage.
10.40 Completely melted; stir bath.
10.45 Add crushed silicon, 150 lb.: 75·00 per cent. Si.
10.55 Stir bath.
11.00 Test for laboratory.
11.10 Start slag off } one-half to two-thirds of slag removed.
11.20 Finish slag off }
11.25 Add lime, 300 lb.
11.30 Add chromium, 2,500 lb.: 70·20 per cent. Cr, 0·20 per cent. C.
11.40 Test reported: 0·07 per cent. C, 0·25 per cent. Mn, 0·08 per cent. Si, 9·03 per cent. Cr.
NOON
12.00 Add chrome, 2,500 lb.: 70·20 per cent. Cr, 0·20 per cent. C.
P.M.
12.30 Stir bath; chrome in solution; temperature tested.
12.40 Add lime, 150 lb.; crushed silicon, 50 lb.: 75·00 per cent. Si.
12.55 Add manganese, 95 lb.: 82·00 per cent. Mn, 0·60 per cent. C.
1.10 Stir bath; temperature test.
1.20 Tapped; added to ladle 190 lb. Ca-Si: 62·00 per cent. Si.

LADLE ANALYSIS

C %	Mn %	P %	S %	Si %	Cr %
0·086	0·43	0·016	0·012	0·43	17·30

Acid Process

Acid-lined arc furnaces have a very limited application and are chiefly employed in steel foundries. Acid furnaces require selected scrap and raw materials low in sulphur and phosphorus. In many foundries acid furnaces are used for the remelting of the scrap arising in the foundry, whilst basic-lined furnaces are used for converting the miscellaneous brought scrap into low sulphur and phosphorus steel.

The general construction of the furnace is similar to the basic furnace, except that the basic refractories are replaced by silica. The roof and side walls, which are built of silica bricks, have a longer life than that obtained in basic furnaces. Roof lives of 300 heats are quite normal.

The *hearth* usually consists of a course of firebricks next to the shell, upon which two courses of silica bricks are laid. The working hearth is then rammed into position on this brickwork. Frequently the hearth mixture is composed of silica sand mulled with about 4 per cent. of ball clay. This mixture is moistened with a 5 per cent. solution of sulphite of lye.

An excellent description of the *acid practice* adopted by the K. &. L. Steelfounders & Engineers Ltd., has been given recently.[19] The furnace employed is a 5-ton Birlec nose-tilting arc furnace of the swinging-roof type. The power is supplied to this furnace by a 1,200-kVA. transformer, whilst a similar furnace working on a basic lining is provided with an 1,800-kVA. transformer.

The charge consists of specially selected low sulphur and phosphorus scrap, and melting is commenced at 180 volts. No sand or slag is added. If the slag becomes thick and low in iron oxide towards the end of the melt, iron ore is added. If the slag is fluid and black, no ore additions are made until the carbon boil begins. The bath attains a temperature of approx. 1,620 deg. C. before the commencement of the carbon boil, and a vigorous boil is maintained with ore additions as required, until low residual silicon and manganese figures are obtained and until the carbon is reduced to the desired limits. The bath is subsequently deoxidized by means of silico-manganese and ferro-silicon additions. The steel is adjusted to the desired specification by alloy additions to the bath.

The reactions in the acid-lined furnace are similar to the reactions in the acid open-hearth furnace, except of course that little refining takes place in the electric furnace. The acid-lined arc furnace is merely a melting medium for selected steel scrap and is eminently suitable for the reclamation of alloy steel scrap.

Induction Furnaces

As early as 1887, Ferranti demonstrated that iron and steel could be melted by the resistance of the metal to the passage of electric currents induced within it. The Ferranti induction furnace can be regarded as an electric transformer in which the primary coil induces a current in an annular chamber or bath of metal which replaces the secondary winding. The same principle was adopted in the U.S.A. by Colby, although the most successful early induction furnace was that designed by Kjellin about 1900.

In the Kjellin furnace, shown in Fig. 125, the metal lies in a circular trough, this ring of metal forming a closed circuit through which a secondary current is induced by means of a primary high voltage current passing through specially wound coils. The primary coil is level with the bath of metal and is wound around one leg of the transformer, which is built up of laminated plates.

Another well-known early induction furnace was designed by Frick, and was similar to the Kjellin furnace. A ten-ton Frick furnace was successfully operated with a power consumption of 620 kWh. per ton of steel produced. The annular bath of this furnace had an outside diameter of 9 ft. and an inside diameter of 6 ft. 4 in. The charge was made up to approx. 10 tons, of which between 6 and 7 tons were teemed off at the end of the process. The remainder of the charge, which after allowing for depreciation amounted to about 2 to 2½ tons, was left in the furnace to form the secondary circuit for the subsequent heat. The primary current was at 5,000 volts, with a frequency of 5 cycles per second, the power being equal to about 265 amps. The power factor of this furnace was only 0·528.

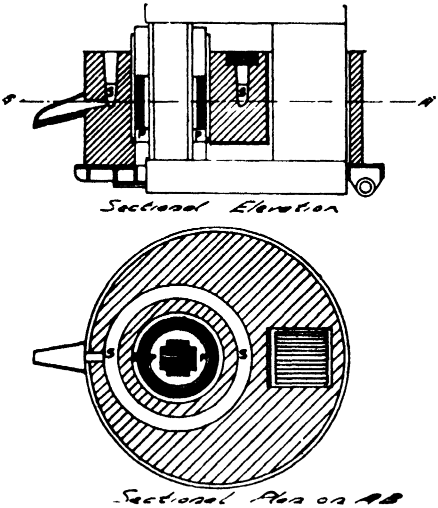

FIG. 125. Kjellin induction furnace.
(By courtesy of the Staffordshire Iron and Steel Institute.

Since these early experiments, considerable progress has been made in the design and operation of induction furnaces, which can be conveniently divided into two types:

(1) Low-frequency furnaces.
(2) High-frequency furnaces.

In both cases the heat is produced by the resistance of the charge to the passage of induced currents. The early induction furnaces were of the *low-frequency type*, operating on a frequency of 5 to 10 cycles per sec., and were similar in design to the Kjellin and Frick furnaces.

The space or distance between the primary coil and the annular bath forming the secondary circuit was always comparatively large. There was, therefore, a large area in which stray magnetic fields were induced and in which much leakage took place, resulting in a low power factor. The load factor was fairly regular and the distribution of energy approximated:

To the charge	.. ..	65 per cent.
To electrical losses	.. ..	5 per cent.
To radiation losses	.. ..	25/35 per cent.

The radiation losses are high owing to the difficulties experienced in designing suitable furnace covers and the general design of the melting chamber.

This type of furnace is merely a melting medium, there being little or no space for slag refining, even if a sufficiently fluid slag could be maintained. It is also necessary to leave a quantity of molten metal in the furnace after each tap to form the secondary circuit for the subsequent heat. When starting up an empty furnace, this secondary circuit must be provided in the form of a metal ring or similar device, which is laid in the annular bath.

In spite of these disadvantages, low-frequency furnaces are still employed for the production of special steels. Farnsworth and Johnson[12] discuss the operation of a 6-ton furnace in the U.S.A. employed for the reclamation of stainless steel scrap. This furnace is of the horizontal ring type having a crucible 7 in. wide at the bottom and 15 in. wide at the top. The crucible was 15 in. deep. The primary coil was supplied with current from a single-phase generator at 2,200 volts and 900 amps. with a frequency of 8·57 cycles per sec. The furnace was operated in conjunction with an arc furnace.

One of the chief uses of the low-frequency furnaces is in conjunction with open-hearth or electric-arc furnaces. The molten metal is transferred from these furnaces to the low-frequency furnaces where it is deoxidized and quality refined.

The power factor of induction furnaces is of considerable importance. As shown by Wright,[23] the power factor of any induction furnace can be determined from the formula:

$$\text{Tan Y} = \frac{c\,n\,a}{l\,s}\left(\frac{1}{W_s} + \frac{1}{W_p}\right)$$

where: Y = the angle of phase displacement
n = the frequency
a = the area of the cross-section of the steel in the annular bath
l = the length of the bath or channel
s = the specific resistance of the steel
c = a constant
W_s = the magnetic resistance around the secondary circuit
W_p = the magnetic resistance around the primary circuit

The power factor (Cos Y) is highest when Tan Y is the lowest, or when:

(1) The magnetic resistances are high
(2) The frequency is low
(3) The electrical resistance of the secondary circuit is high.

The low power factor which was often characteristic of the early induction furnaces was due to the excessively low resistance of the secondary circuit and the large area in which leakage of the magnetic lines of force could occur.

When, for metallurgical and other reasons, it was decided to dispense with the iron core of the early induction furnaces, it was necessary to *increase the frequency* of the current in the primary coil, in order to induce a secondary current of sufficient magnitude to melt a body of metal. An A.C. current does not pass through the whole surface of the conductor, but tends to cling to the surface. The more the frequency of the current is increased, the more pronounced this skin effect becomes. Therefore, as the frequency is increased, so the heat produced is increased according to the formula:

$$P = \frac{5{,}030\sqrt{r}}{\sqrt{F\mu}}$$

where P = the depth of penetration of the current (cm.)
r = the resistance of the charge (ohms/sq. cm.)
F = the frequency in cycles/sec.
μ = the magnetic permeability of the charge which at high temperatures is unity.

If the frequency is increased from 50 to 2,500 cycles per sec., the depth of penetration is decreased to approximately one-seventh.

Since resistance is effected by the area of the conductor, an increased frequency resulting in less penetration, thereby causing a reduced effective area of the conductor, produces a greater thermal effect. The increased frequency results in a greater heat production for the same total area of the conductor. In other words, the higher the frequency, the greater the thermal effect per unit of true cross-sectional area of the conductor, which in the case of high-frequency furnaces is the metallic charge.

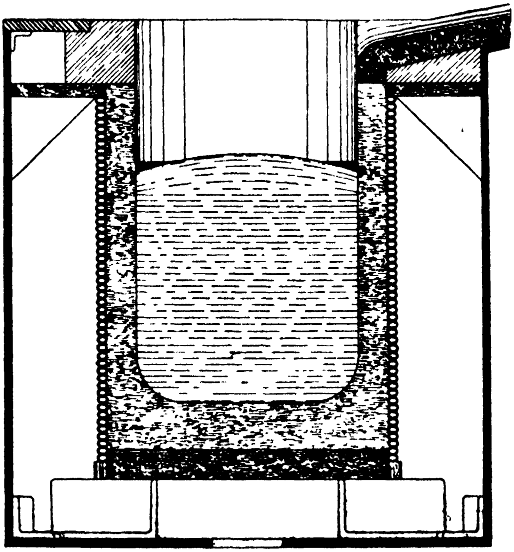

FIG. 126. High frequency furnace.
(By courtesy of the Staffordshire Iron and Steel Institute.)

The *high-frequency furnace* is usually supplied with current at 1,000 to 2,000 volts, with a frequency of 1,000 to 2,000 cycles per sec. In the earlier furnaces this high-frequency current was supplied by static convertors of the Test a spark-gap type. In recent years, however, motor generator sets have been designed to supply frequencies from 500 to 2,500 cycles per sec.

In the high-frequency furnace the metallic charge or bath, which is contained in a crucible made of either acid, basic or neutral refractory, is surrounded by a primary coil. The heating or primary coil is made up of water-cooled copper tubes, and although it is situated very near to the crucible, it is maintained at a very low temperature. The space between the crucible and the coil is packed with sand or suitable insulating material. The general design is shown in Fig. 126.

The crucible is sometimes a formed crucible, although the modern practice is to make the crucible by ramming suitable refractory material around the outside of a metal former or template. This rammed material is heated by a current induced in the metal former until the refractory lining has fritted. The former or template may then be withdrawn or it may be melted with the first charge.

Although the early high-frequency furnaces offered an ideal medium for melting metals without fear of contamination of the charge by carbon, sulphur, or other deleterious elements, they also presented some disadvantages.

The removal of the iron core which formed an important feature of the low-frequency induction furnace, resulted in the reduction of the conductivity of the magnetic flux to the magnetic conductivity of air with a corresponding reduction of flux. It also resulted in the magnetic field being uncontrolled in its distribution, which not only limited the efficiency of the furnace, but also made it imperative to avoid magnetic material in the construction of the furnace. This is the reason for some of the earlier high-frequency furnaces being encased in a wooden box.

At a later date when a structural steel casing was used for the coreless-type induction furnace, a magnetic shield of sheet copper was introduced to prevent over-heating of the furnace casing by stray magnetic

In this furnace an alloy steel path surrounds the inductor coil and this steel path passes underneath the crucible, terminating in the nucleus of a core in a central position under the crucible and in line with the vertical axis of the inductor coil. This design results in a strong and centrally situated magnetic field within the crucible containing the charge, with a corresponding reduction in the tendency for stray magnetic fields and obvious valuable economies. This aspect has been discussed in technical literature[25] and is explained in Figs. 127 and 128.

Some difficulty was experienced in refining steel in the early high-frequency furnaces. This was mainly due to the erosion of the furnace crucible, which was comparatively thin. The erosion was accelerated

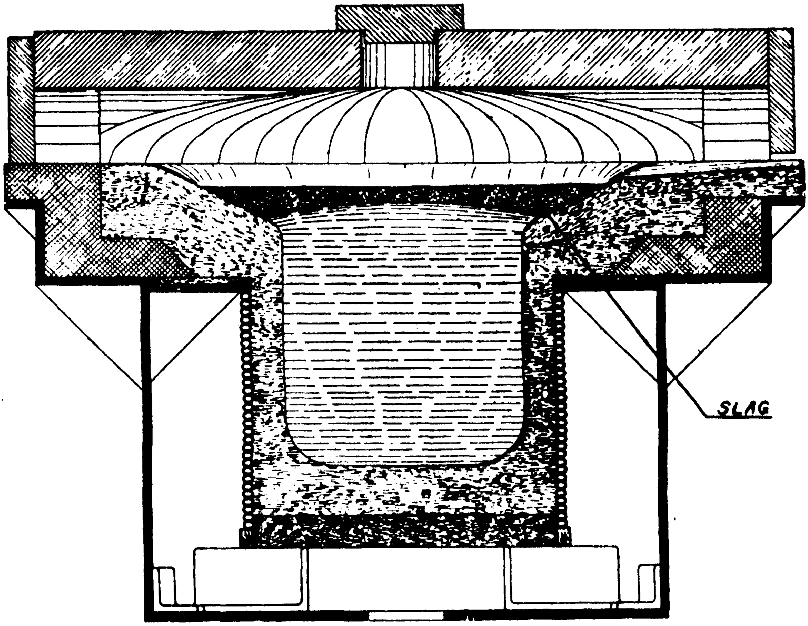

FIG. 129. Stobie high frequency furnace.
(*By courtesy of the Staffordshire Iron and Steel Institute.*)

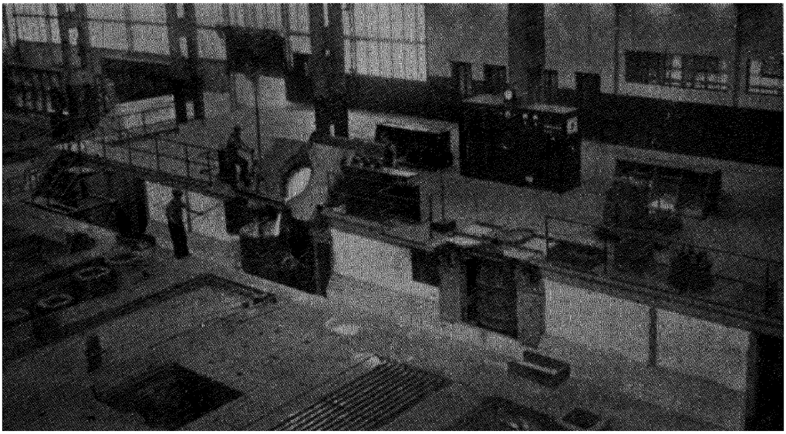

FIG. 130. Layout of high frequency furnace plant.

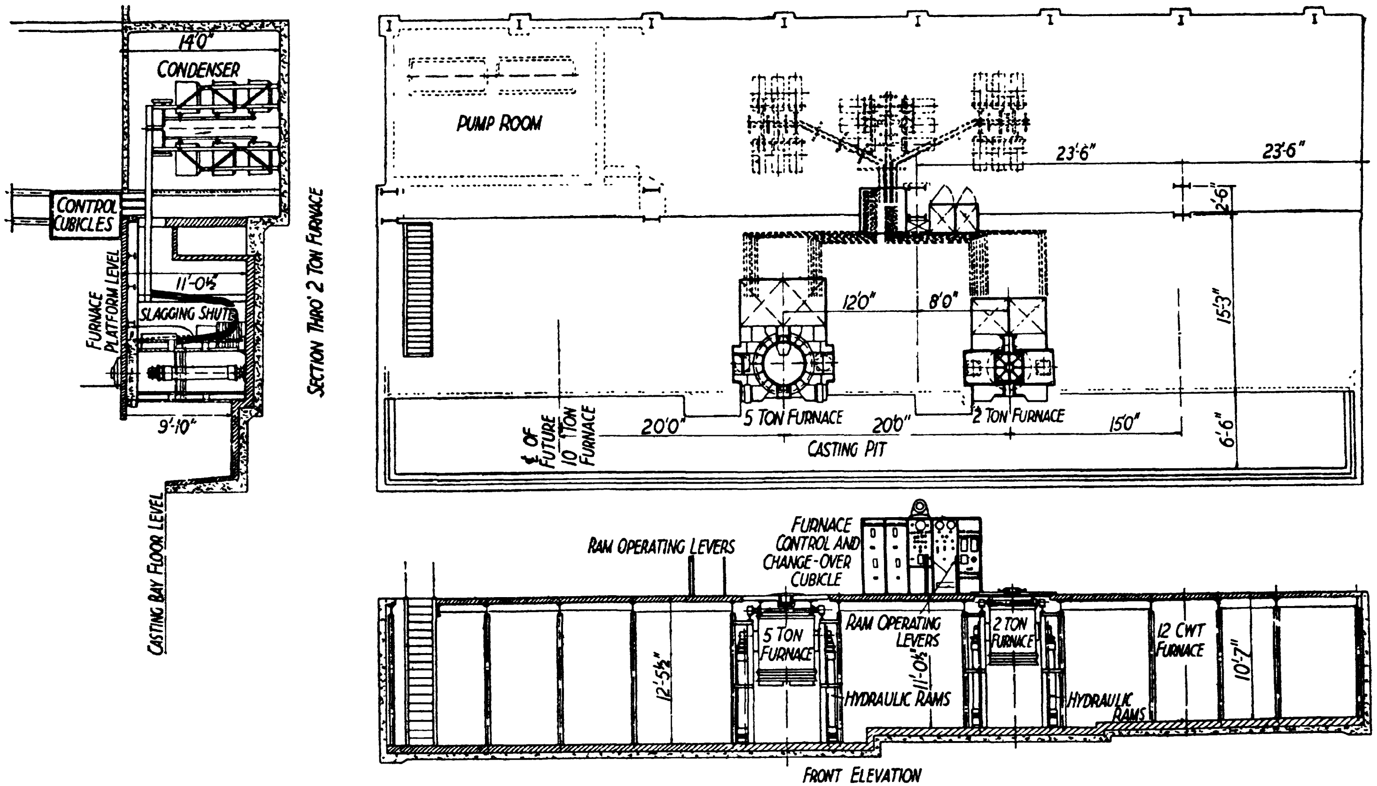

FIG. 131. Layout of high frequency furnace plant.

by the turbulence of the metal in the crucible, as the induced current causes the level of the metal to rise in the centre in the form of a fountain. This turbulence was at one time considered necessary to successful refining, but Stobie[26] has demonstrated to the contrary.

The difficulty connected with this excessive erosion of the crucible around the slag line, is to a great extent overcome in the Stobie High Frequency Furnace, shown in Fig. 129. In this furnace the walls of the crucible are carried above the height of the inductor coil, and are made considerably thicker at the slag line. On the slag line the walls of the crucible are also sloped similar to the banks of an arc or open-hearth furnace.

The typical layout of a high-frequency furnace consists of a motor generator for producing the high-frequency current, the control gear, two banks of condensers and the furnace proper, as shown in Figs. 130 and 131.

The frequency and voltages employed vary considerably, but as a rule the frequency lies between 1,000 and 2,250 cycles, the higher value being used for small furnaces, that is, those under one-ton capacity. The voltage in the furnace coil is generally between 1,000 and 2,000 volts.

The control gear usually permits manual adjustment or automatic working. One bank of condensers is set in series to maintain constant voltage and another bank in parallel to maintain the power factor. The bank of condensers set in parallel is usually divided into one bank permanently connected across the furnace terminals and a bank subdivided into sections which can be switched into circuit individually as required. The fixed bank is generally about 40 per cent. of the total, whilst the number of units which can be switched into circuit vary from 6 to 12. This is all controlled from the control panel.

As pointed out by Swinden,[27] the high-frequency furnace should not be regarded as a refining medium. It is, however, an excellent melting medium which offers the following advantages:

(1) There is no carbon pick-up, and it is therefore especially useful when producing low-carbon special steels.

(2) Sulphur contamination is unknown.

(3) Temperature control is definite and accurate.

(4) A high uniformity of the charge is obtained owing to the motion of the bath due to the currents within the bath.

(5) A high conservation of alloys is possible. Nickel, copper and

cobalt are stable, whilst no appreciable loss of chromium, molybdenum or tungsten takes place. Very small losses occur with the deoxidizers such as manganese, silicon, aluminium, vanadium, titanium and zirconium. Swinden quotes a case of the manufacture of a 13 per cent. aluminium steel in which the aluminium loss was only 0·4 per cent.

(6) As far as deoxidation products are concerned, the non-metallic inclusions can be reduced to a minimum.

The chief disadvantage would appear to be the fact that the slag is usually too cold to effect any refining, although a white-falling slag, as employed in the arc furnace, can be, and at some works is, being regularly produced.

Swinden[27] states that steel formerly being made exclusively in coke-fired crucibles can now be made satisfactorily in the high-frequency furnace, whereas highly alloyed tool steels can be produced with even greater success. The high-frequency furnace is the only satisfactory method of manufacturing steels containing high percentages of aluminium.

There appears no doubt that the high-frequency furnace will largely replace the crucible process for the production of high-grade special steels. It is also employed extensively in conjunction with small open-hearth and electric-arc furnaces for the production of special steels on a duplex process. The molten metal is transferred from these furnaces to the high-frequency furnace, where it is deoxidized and quality refined. When it is desired to carry a white-falling slag, a basic lining is employed.

The practice discussed by Swinden[27] concerned the production of special steels at the Stockbridge plant of the United Steel Cos., Ltd. This layout has also been described in considerable detail by Inman and Wainwright.[28] It consists of one 2-ton and one 5-ton high-frequency furnace. The 5-ton furnace was operating on a frequency of 1,125 cycles per sec. at a voltage of 3,000 and a rating of 1,250 kW. It was working with an acid lining on cold charges and producing 180 tons of steel per week. The power consumption was 650 kWh. per ton.

According to the Fuel Efficiency Committee[1] the heat balance sheet of a high-frequency furnace should approximate:

To the charge	59 per cent.
To cooling water	3 ,, ,,
To electrical losses, etc. ..	22 ,, ,,
To radiation losses, etc. ..	16 ,, ,,

The load factor is fairly good. The power factor, which is usually low, can be improved by the use of condensers as already indicated.

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CHAPTER 10

INGOTS AND INGOT PRODUCTION

Casting-Pit Practice

Whatever process of steelmaking is adopted, the liquid steel must be cast into a suitable form so that when solidified it can be either rolled, forged or pressed into a finished or semi-finished article. The ingots produced in this manner vary in size and shape according to the use for which the steel is intended. The production of ingots from liquid steel can be conveniently termed *casting-pit practice.* Correct casting-pit practice is of considerable importance, as sound steel may be ruined by faulty methods of ingot production. The speed of teeming, the temperature of teeming, the rate of rise in the ingot mould, the method of pouring, the design of the ingot mould, and the preparation of the ingot mould, are some of the many factors of vital importance to the production of sound ingots.

The actual methods adopted for the disposal of the liquid metal and slag depend on the layout of the particular plant. In a few of the older open-hearth melting shops the pitside runs parallel with the furnaces, as shown in Fig 42. In this case the pitside is sunk below the ground level and is divided into two parts, one for receiving the slag pans and the other part in which the ingot moulds are set.

Where a sunken pitside is employed, the casting or metal ladle, which receives the molten metal from the furnace, travels in a bogie. This bogie or carriage is moved over the moulds by a jib-crane, which is also used for the stripping of the ingots when solid.

This type of pitside has been largely superseded, except for the production of very large ingots for forging or pressing. The method has several disadvantages:

(1) The upkeep of the brickwork and bogie tracks is high, due to the heavy loads employed.

(2) Any breakout from the furnace is liable to drain into the sunken pit, aggravating any trouble.

(3) The space available is usually confined, due to the span of the bogie tracks.

(4) It is impossible to change casting arrangements to meet any change of mill programme or irregular furnace melt.

In the older Bessemer shops the pitside was also a sunken pit adjacent to the converters. Many typical examples of these pitsides are given in the older textbooks. They possessed disadvantages similar to those mentioned in connection with the open-hearth plant.

Modern practice favours overhead crane casting, in which case the ladle is suspended in the arms of an overhead crane. At some plants the ladle is held under the furnace chute for tapping, whilst at other plants the ladle is placed in a stand underneath the chute and removed by the crane when it is filled. The latter method has the advantage of releasing the crane for other work and reducing the amount of crane power necessary for the pitside operations in general.

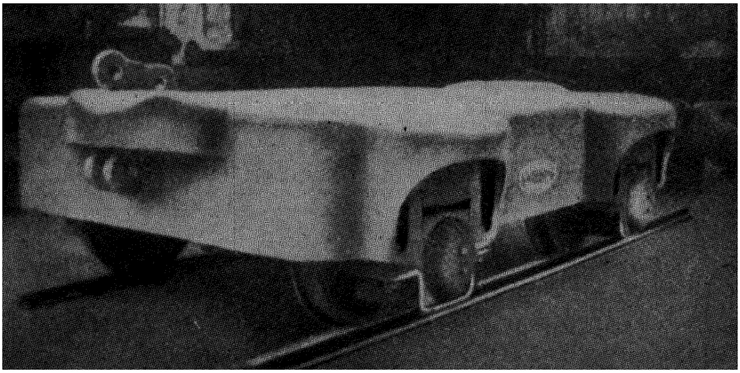


FIG. 132. Ingot casting bogie.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

When the ladle is filled it is moved by the crane to the ingot moulds, which are usually set on specially designed ingot casting bogies. These bogies, as shown in Fig. 132, are designed to accommodate from two to six moulds and are streamlined to prevent any metal adhering to them in the event of any breakout from the moulds or splashes from running stoppers, and such occurrences. The ladle may remain suspended in the arms of the overhead crane during the teeming of the ingots, the crane travelling from mould to mould as required. A better practice, however, is to place the ladle in a ladle stand and to pass the moulds on bogies under the ladle by means of hydraulic or electrical gearing. This method again releases the crane for other work and reduces the crane power required. Typical layouts are shown in Figs. 31, 32, 33 and 42.

When the moulds are filled they are transferred by loco-power to the stripping bay, where the moulds are removed as soon as the ingots are solid. When the moulds have been removed the bogies are shunted to the rolling mill soaker bay, where the ingots are removed. At most plants these ingots are weighed on route to the mills. In the case of wide-end-up moulds the ingots are removed from the moulds and sent to the mill in specially constructed ingot cars. Care should be taken to see that these ingots are placed in an upright position so as to avoid any displacement of the central pipe within the ingot.

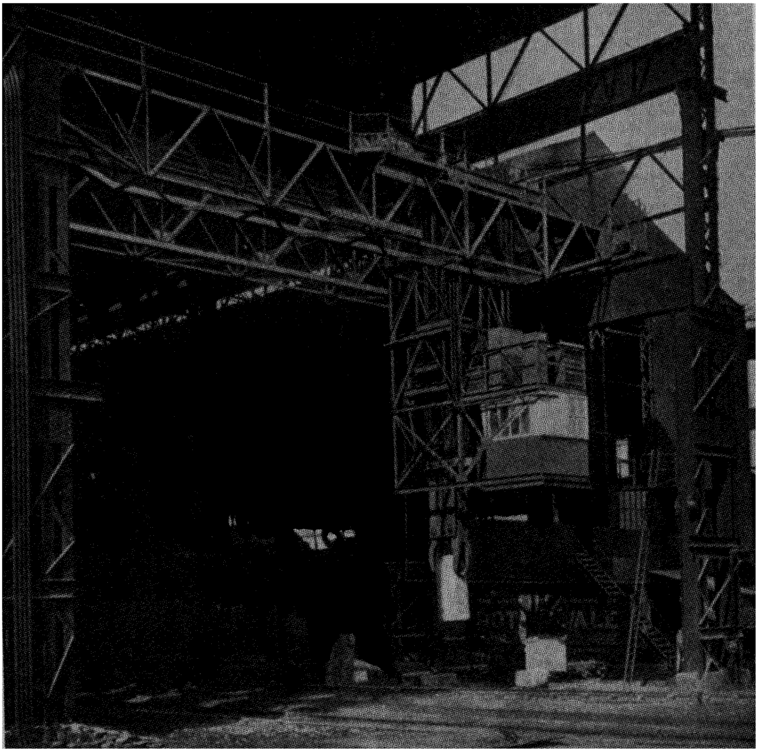


FIG. 133. Stripping crane.
(By courtesy of the Wellman, Smith, Owen Engineering Corporation, Ltd.)

When employing narrow-end-up moulds the common practice is to strip the moulds from the ingots and to reset them on another set of bogies ready for further use. Frequently the stripping is performed in a bay adjacent to the actual casting bay. This practice allows more space for the preparation and assembly of moulds and mould bottom plates. It has special advantages when a large percentage of bottom run steel is being produced, as the necessary equipment for cleaning the runner bricks after they have been set into the bottom plate. Fig. 133 shows a stripping crane at work. The actual design of the arms depends on the shape of the lifting lugs on the ingot moulds, whilst the central ram on the crane can be brought into operation in the event of any ingot sticking in the mould.

Slag Disposal

At plants where overhead crane casting is adopted the slag from the furnace is allowed to run over the metal ladle into the slag ladles as shown in Fig. 134. These slag ladles are cast haematite bowls, supported in a specially shaped bogie. They may be either side- or end-tipping, according to the method of subsequent disposal of the slag. In either cases the mode of tipping the ladle is simple. The loco coupling is disconnected from the draw-bar of the bogie and attached to a chain which operates over a system of pulleys whereby the bowl is rotated in its trunnions.

This method of slag disposal is much more economic than the older method employing slag pans. The life of these slag pans was usually short, in spite of the fact that they were made of haematite iron, and the labour involved in tipping these pans and subsequently loading the slag into wagons for the tip was high. On the other hand the cost of maintaining slag ladles is comparatively moderate, providing reasonable care is taken in their use. It is important that metal should not be allowed to overflow into these ladles, as this practice will considerably shorten their effective life.

Metal Ladle

The metal or casting ladle consists of a cylindrical steel shell, made of ¾ to 1¼ in. steel plates, according to capacity of the ladle. This shell is lined with a high-grade firebrick, the properties of which are more fully discussed in Chapter 16 dealing with refractory materials. The refractory lining varies from six to eight inches according to local practice. Care should be taken to see that the joints between the bricks are as small and as close as possible. The normal practice is to have a double thickness of brickwork, and the joints in the two rows of bricks

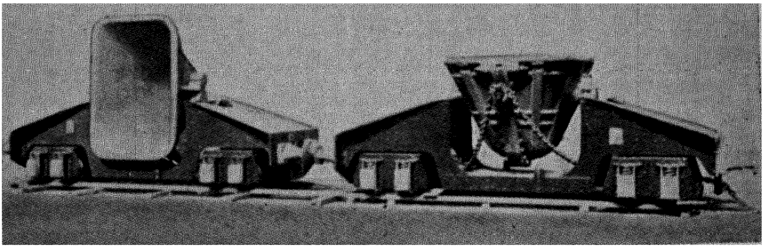


FIG. 134. Slag ladle.
(From "Practical Steelmaking", by W. Lister, by courtesy of the author.)

should be staggered. The author has found that even on 80- and 100-ton capacity ladles a 6-in. lining has proved satisfactory. In this case the back layer consisted of 1½-in. bricks and seldom had to be replaced. The facing layer consisted of 4½-in. bricks—the ordinary 9 × 4½ × 3 in. standard

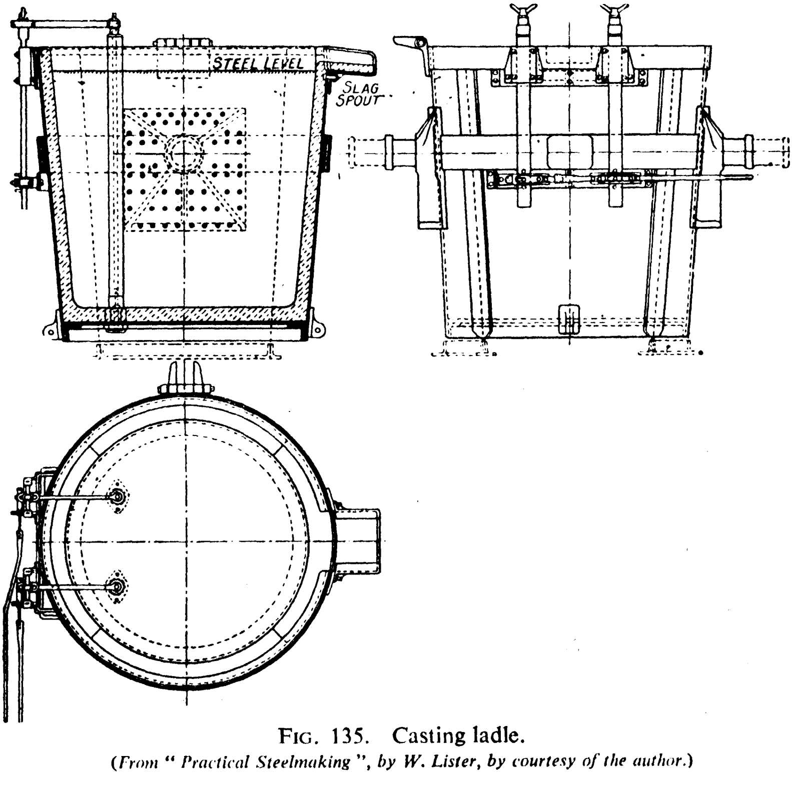


FIG. 135. Casting ladle.
(From "Practical Steelmaking", by W. Lister, by courtesy of the author.)

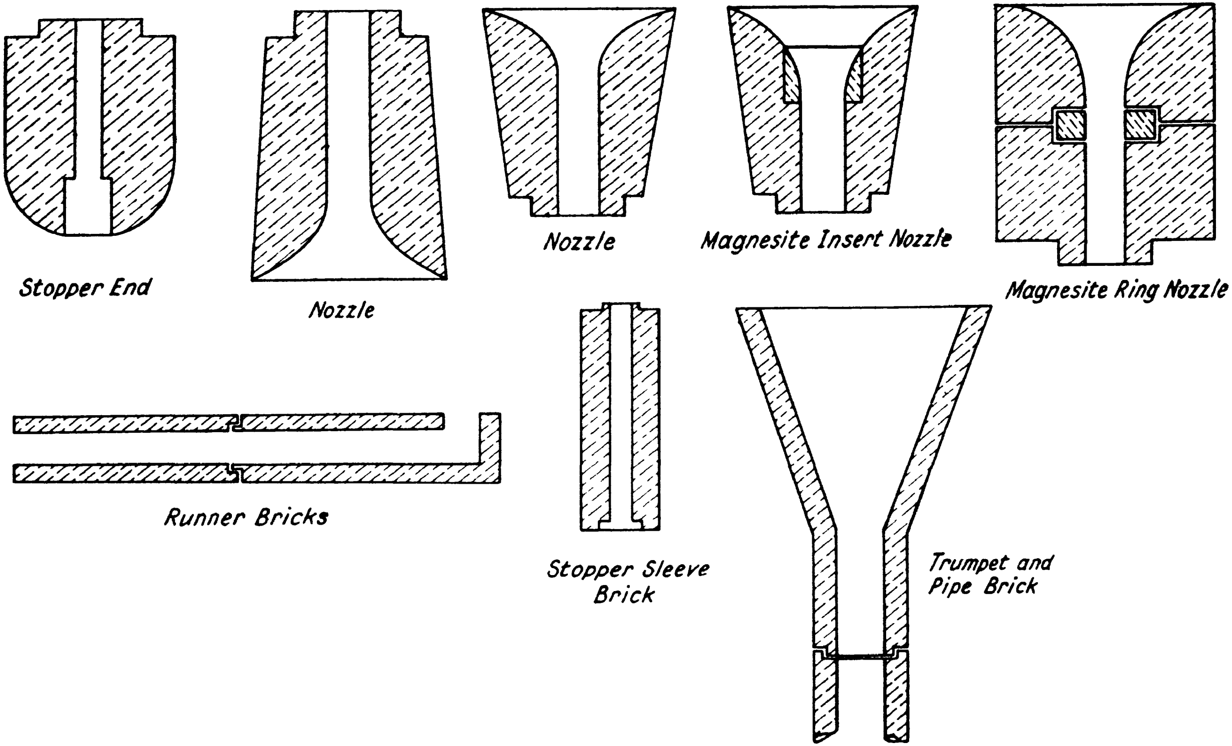


FIG. 136. Casting pit refractory materials.

brick—with side-arch bricks as required to give the requisite contour. Some operators, however, prefer to employ specially shaped bricks, which actually follow the exact contour of the ladle. These ladles are usually circular, although in some cases they are oval in shape, especially where the older methods of sunken pitsides are still employed.

The method of bricking the ladle bottom varies considerably in different shops, some people employing specially shaped bricks and others being content to use the standard square and side-arch bricks. Both methods appear to give equal success, the most important feature being to ensure that the bottom of the ladle has a good slope towards the nozzle, through which the metal flows from the bottom of the ladle. In bricking the bottom of the ladle a space is left in the brickwork for the insertion of the nozzle and the space between the *nozzle* brick and the bottom brickwork is filled in with rammed ganister. The normal method of setting the nozzle is shown in Fig. 135. Each ladle is provided with one or two nozzles, the more modern practice being two nozzles.

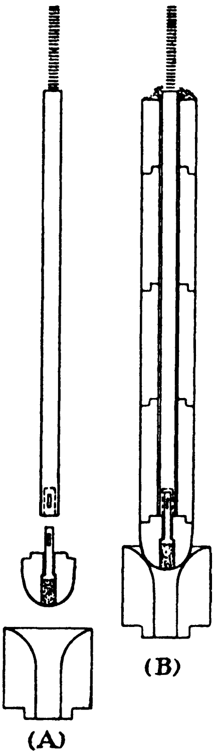


FIG. 137.
Stopper arrangement.
(From "Practical Steelmaking", by W. Lister, by courtesy of the author.)

Different plants vary their nozzle design slightly, some having a convex and others a concave top, as shown in Fig. 136. It is probable that the convex top gives a better fit with the stopper end or head. In most cases the nozzle is slightly tapered. Where the nozzle passes through the bottom of the ladle shell it is secured by a ladle gland. This gland, shown in Fig. 137, consists of a steel casting, which is secured to the ladle by means of cotters. The gland is provided with two holes, through which pass pins attached to the bottom of the ladle shell. These pins have a short slot which enables the gland to be cottered to the ladle. In designing the ladle gland, care should be taken to ensure that it is the right thickness and that the lower section of the nozzle is sufficiently long to project slightly through the gland. If it projects too far there is a likelihood of the nozzle being chipped, whereas if it fails to project far enough the metal of the gland may chill the stream of metal as it leaves the ladle. Any chilling of the metal at this point results in an irregular splashing stream, which will cause the ingots to have a bad surface.

The flow of metal from the ladle through the nozzle is regulated by means of a *stopper*. The stopper consists of a steel rod, lined with firebrick tubes or sleeves. The lower brick of the stopper which fits into and closes the nozzle is called the *stopper-end*. The stopper-end is secured to the rod by means of a bolt and cotter pin. This bolt passes through the centre of the stopper-end and projects into a slot in the stopper-rod, to which it is fastened by means of a cotter pin, which in turn passes through both the bolt and the stopper-rod. The general assembly of the stopper is explained diagrammatically in Fig. 137.

This stopper is activated by means of gearing on the outside of the ladle. This gearing consists of a steel flat which moves easily up and down in a slide bracket. The upper end of the slide or flat is designed to receive the swan-neck, which connects it to the stopper inside the ladle. The lower end of the slide is connected with a lever, by means of which the slide and stopper is operated. The whole mechanism is shown in Fig. 135.

Preparation of Ladles

When the ladle has been used, it must be freed from any skull, and any slag must be removed. Any bad joints are made good by ramming ganister into the spaces. In this way a ladle will give considerable life before requiring rebricking. The actual life will very largely depend on the amount of slag that is allowed to flow into the ladle, as it is the slag that has the most erosive effect on the firebrick lining. Bad joints which allow any skull to lodge in them also reduce the life of ladles, due to the fact that the removal of this skull frequently disturbs the lining. Normally the bottom bricks last much longer than the side lining.

Whether a new bricked ladle is being used or one which has been repaired, it must be carefully dried before the molten metal is run into it.This drying is frequently done by placing an ordinary fire-bucket into the ladle. At most modern plants, however, special ladle-drying apparatus, employing producer-gas, mixed gases or oil, is used.

Ingot Moulds

Except in the case of foundries where the steel is poured into special moulds consisting of sand or similar material having the shape of the finished article, the molten steel is cast into ingot moulds. These moulds

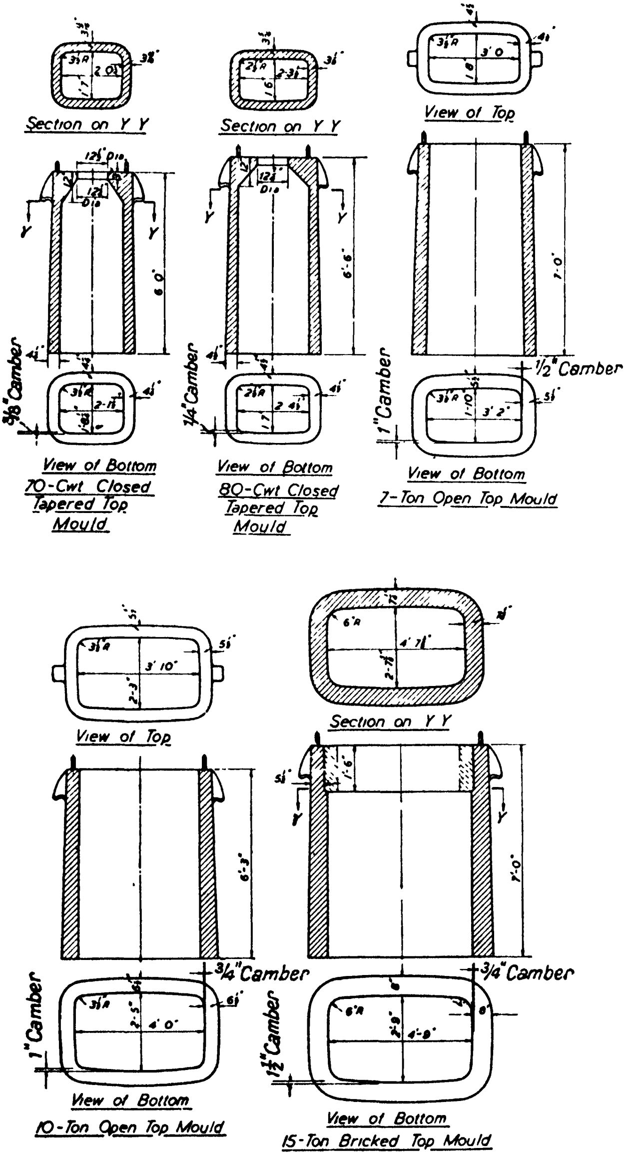


FIG. 138. Typical ingot moulds (various sources).

are haematite castings and may be square, rectangular, round or polygonal in section, according to the use for which the steel is required.

Square cross-section moulds are the chief type and are employed for

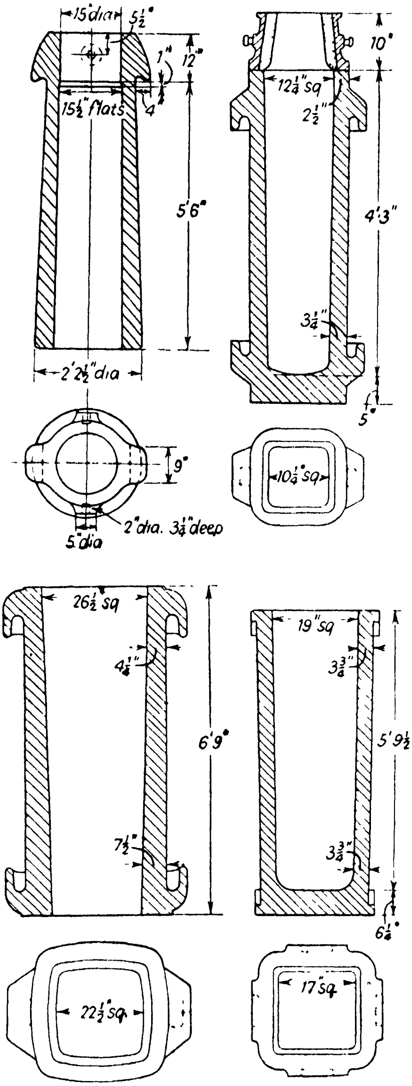


FIG. 139. Typical ingot moulds (various sources).

the production of ingots for subsequently rolling into rails, billets, sections, girders, etc. Rectangular moulds, which are usually referred to as *slab moulds*, are used for steel for the production of plates. Round moulds, which often cause rolling troubles, as discussed later, are used for the tube trade. Polygonal moulds are employed for tyre steels and steel for special purposes, such as forgings.

TABLE XXXIX

DIMENSIONS OF INGOT MOULDS

Capacity	Type	Cross-section	Length	Wall Thickness	Corner Radius
5 Ton	Slab	Top 3 ft. 2 in. × 1 ft. 5 in. Base 3 ft.5 in. × 1 ft. 11 in.	5 ft. 0 in.	5 in.	—
25 Cwt.	Square	Top 14 in. square Base 15½ in. square	5 ft. 0 in.	4½ in.	1¾ in.
42 Cwt.	Square	Top 18½ in. square Base 20 in. square	4 ft. 6 in.	5½ in.	1½ in.
With Feeder Head .. 3 Tons 0 Cwt. ..	Solid Bottom	Top 1 ft. 8 in. square Base 1 ft. 6 in. square	6 ft. 3 in. Internal 5 ft. 9 in.	4 in.	1½ in.
With Feeder Heads 3 Tons 10 Cwt. ..	Solid Bottom	Top 1 ft. 9¼ in. square Base 1 ft. 8 in. square	6 ft. 4 in. Internal 6 ft. 0 in.	4 in.	2 in.
3 Tons 10 Cwt. ..	Square	Top 1 ft. 6 in. square Base 1 ft. 8 in. square	7 ft. 0 in.	5 in.	1½ in.
3 Tons 12 Cwt. ..	Square	Top 1 ft. 8 in. square Base 1 ft. 9½ in. square	6 ft. 6 in.	5 in.	—
5 Ton	Octagonal	Top 2 ft. 2 in. diameter Base 2 ft. 4 in. diameter	Head 2 ft. 0 in. Chill 3 ft. 8 in. Total 5 ft. 8 in.	5 in.	—

The size of these ingot moulds will vary according to the purposes for which the steel is to be employed. A popular size varies from 3 to 3½ tons in the case of section rolling, whilst in the manufacture of plates a slightly larger ingot, up to 6 tons, is frequently favoured. In the ordinary cogging mill a 3- to 3½-ton ingot generally represents the most economic and easily handled ingot. Various sizes and types of moulds are shown in Figs. 138-139 whilst Table XXXIX gives some typical dimensions and uses of ingot moulds.

A few specially shaped moulds are employed for the production of wheel centres and similar articles. These specially shaped moulds are sometimes referred to as cheese moulds, owing to their shape. The former practice of making specially shaped ingots for tyres has been replaced by the use of octagonal moulds. The ingots produced in these moulds are subsequently cut up into discs, each of which is rolled into a tyre. This practice gives a better control of ingot quality.

As pointed out by Bacon[1] the cost of ingot moulds forms a very important item in the conversion cost of steel production. After allowing for the salvage value of the discarded ingot mould, the cost of ingot moulds probably averages about 2s. per ton on the steel produced. It is, therefore, obvious that the question of mould life is of considerable economic importance.

There are two principal causes for mould failure, major cracking and grazing or minor cracking. This subject has been ably reviewed by Pearce.[2] Major cracking may result from defective founding, defective mould design, which causes cracks to develop at a particular spot, and from thermal shock during use. Major cracking may also result from ill-usage in the works. Pearce[2] points out that cast iron has a maximum resistance to thermal shock when it is mechanically soft and ferritic, rather than when the grain is open, with a larger amount of graphite present.

Grazing or minor cracking is primarily due to the phenomenon known as growth, which is identical in character with scaling. Grazing or fine cracking results in the working face of the mould becoming covered with a network of fine cracks, especially in the middle of each working face where the temperature remains high for a longer period. When grazing has commenced it is cumulative, and the net-work on the mould surface develops to such an extent that the surface of the ingot is impaired, necessitating the mould being discarded.

As shown by Pearce,[2] cast iron has a maximum resistance to grazing when it is mechanically hard, that is pearlitic or cementitic, close grained, low in carbon and silicon, and hence contains a small amount of graphite which should be in a fine state of division. Growth is greater when the carbon and/or silicon content of the iron is high, when the graphite is greater and coarser. It is also influenced by a longer or more frequent heating, and a higher temperature. The use of hot-tops, the re-use of moulds before cooling down, the casting of ingots at high temperatures, steel of a highly penetrating character, and the pouring of ingots in close proximity to each other, all tend to promote growth and shorten mould life.

Bacon[1] enumerates the various factors influencing mould life adversely as follows:

(1) Too short an interval between use.
(2) Ingots not taken from the mould as soon as solidified.
(3) High carbon steel.
(4) High degree of superheat on the metal poured.
(5) Fast teeming.
(6) Direct instead of bottom pouring.
(7) Inadequate stripping facilities, which involve bumping and dropping the mould.
(8) Mould design, such as:
    (a) Wall thickness.
    (b) Method of incorporating the lifting lugs.
    (c) Outside contour.
    (d) Feeder head arrangement.
(9) Chemical composition of the mould material.
(10) Physical condition of the metal.

The effect of most of these factors can be readily appreciated and undoubtedly a large number of ingot moulds are ruined by careless usage within the casting pit. The cooling of moulds by quenching or the too frequent use of very hot moulds are very common causes of reduced mould life.

The composition of the material employed for the production of the mould appears to have an important bearing on its life. Typical analyses of ingot moulds are shown in Table XL. The phosphorus content has been the subject of many discussions on mould life. Many operators prefer to keep this content under 0·050 per cent., whilst others allow it to approach 0·080 per cent. There does appear to be evidence to demonstrate that a high phosphorus content reduces the resistance of the mould to high temperature conditions, and promotes premature failure. In Germany, however, ingot moulds with a phosphorus content as high as 0·20 per cent. have given excellent performances. This aspect is discussed by Pearce.[2]

The subject of *mould design* has an important bearing on the life of the mould, but an even more important effect on the quality of the ingot produced. This question is more fully discussed in conjunction with ingot defects later in this chapter.

TABLE XL

ANALYSIS OF INGOT MOULDS

Example	C	Si	S	P	Mn
A	3·76	1·41	0·047	0·050	0·59
B	3·60	1·75	0·050	0·048	0·48
C	3·58	1·83	0·065	0·038	0·74
D	3·74	2·06	0·061	0·037	0·48
E	3·70	1·93	0·046	0·044	0·68
F	3·77	1·95	0·049	0·046	0·76

Methods of Teeming

The method of teeming has also a decided effect on the quality of the ingot produced. As shown by Kilby[3] the speed at which an ingot is teemed affects the subsequent behaviour during mechanical working. It appears to be the temperature and/or speed with which the ingot is teemed, but these aspects are dealt with in more detail when discussing ingot defects. The rate of teeming or the rise in the ingot mould can be controlled by varying the method of pouring. There are three methods of teeming:

(1) *Direct pouring*, in which the metal is teemed directly from the ladle into the ingot moulds. In this case the rate of pouring can be regulated to some extent by the use of different sizes and designs of nozzle bricks. The normal practice is to employ a nozzle having a 1-in. dia. opening, and these nozzles are usually made of fireclay. This type of nozzle is inclined to erode badly during the teeming of the heat so that the ingots teemed towards the end of the cast are poured relatively faster than those at the commencement of pouring. The enlargement of the nozzle is to some extent neutralized by the drop in the ferro-static head within the ladle.

In order to overcome this erosion several types of special nozzles have from time to time been introduced, one interesting design being the Bagnall-Bethel[4] nozzle, shown in Fig. 140. In practice teeming is commenced with the correct size of secondary nozzle in position, and when a certain number of ingots have been teemed this secondary nozzle can be removed so that teeming is

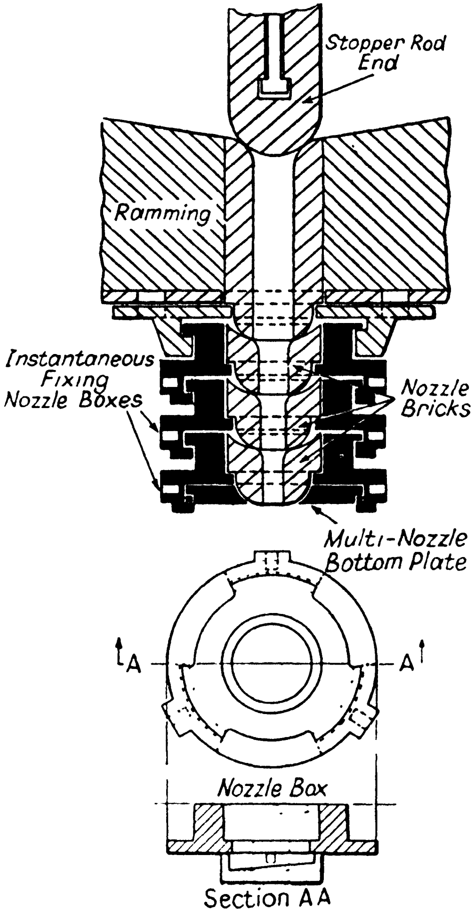


FIG. 140. Bagnall-Bethal nozzle.

continued through the primary nozzle, or another secondary nozzle of the desired size can be inserted. The design of the equipment is very simple and allows the secondary nozzle to be very quickly changed. This method has not been extensively adopted.

Another method of ensuring a more constant rate of teeming was the introduction of magnesite and graphite nozzles. The graphite nozzles were not extensively employed, but many plants still use the various types of magnesite nozzles shown in Fig. 136. Some interesting notes on the relative rates of teeming were given by Jackson,[5] whose test curves are reproduced in Fig. 141.

Naturally the size of nozzle employed depends to some degree on the type of steel being teemed. When teeming a soft heat, which has a more viscous nature, a larger nozzle is required than when teeming a high carbon and therefore more fluid steel. At some plants, where the standard practice for medium and high carbon steel is to use a 1-in. nozzle, a 1¼-in. nozzle is used for the softer varieties.

(2) *Tundishing.* Another method of controlling the rate of flow from the ladle to the ingot mould is to insert a tundish, similar to the

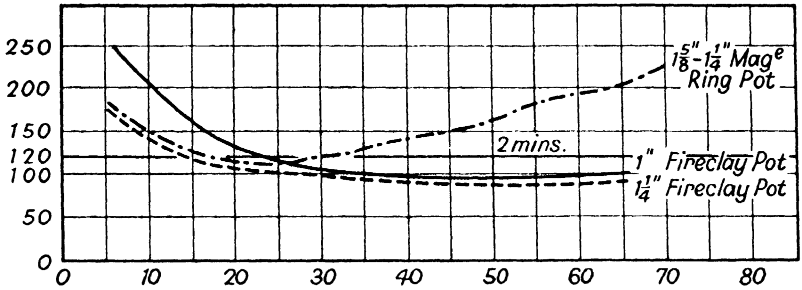
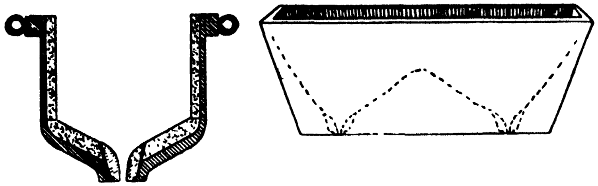


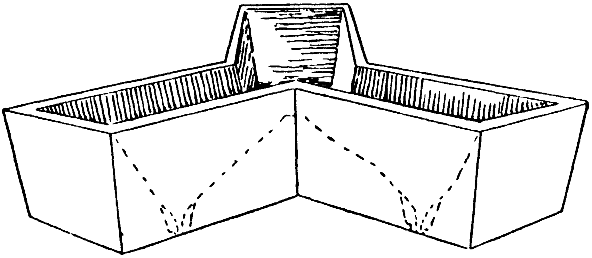
FIG. 141. Effect of nozzles on teeming speed.

design shown in Fig. 142, between the ladle and the mould. These tundishes may be two-, three- or four-way, which allow the steel to flow into two, three or four moulds at the same time, with a corresponding reduction in the rate of teeming or the rate of the rise in the mould.

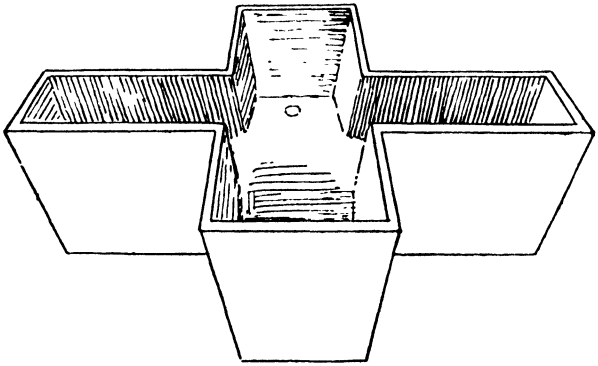
The size of the nozzle employed in the tundish is important and is generally much larger than the nozzle in the ladle, due to the smaller ferro-static head in the tundish as compared with that in the ladle. Even when great care is taken to ensure that the tundish is set level there is a tendency for the metal to flow unevenly and it is necessary to control the flow through the different tundish nozzles by means of a stopper similar to that shown in Fig. 136. This stopper can be slightly inserted into any nozzle which is filling too quickly, whereby the stream is slowed down; or the stream can be completely stopped by the complete insertion of the stopper. This method of teeming is extensively employed for large ingots for forging and for certain classes of special alloy steels.



(*a*) Single Nozzle (*b*) Two Nozzles



(*c*) Three Nozzles



(*d*) Four Nozzles

FIG. 142. Tundish design.
(*By courtesy of the Iron and Steel Institute.*)

(3) *Bottom pouring* is probably the most popular method of controlling the teeming speed, and is extensively employed. The steel from the ladle runs down a trumpet, which is an iron casting lined with fireclay pipes, and through fireclay runners set in a mould bottom plate. One trumpet can be designed to supply steel to two, three and up to six moulds, set on one bottom plate and surrounded by moulds as shown in Fig. 143. Great

care should be taken to see that the joints between the bricks are good, so as to prevent the metal breaking out. The trumpet and runner bricks should be thoroughly dried before use and preferably blown out with compressed air to remove any dust or clay that may remain from the joining.

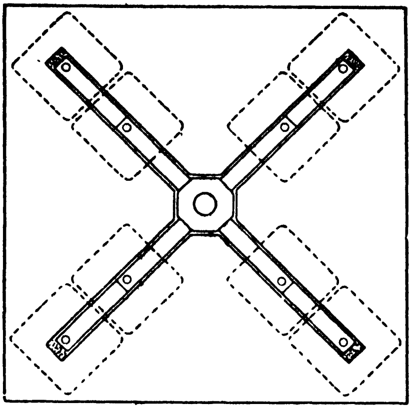


FIG. 143. Bottom-pouring arrangement.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

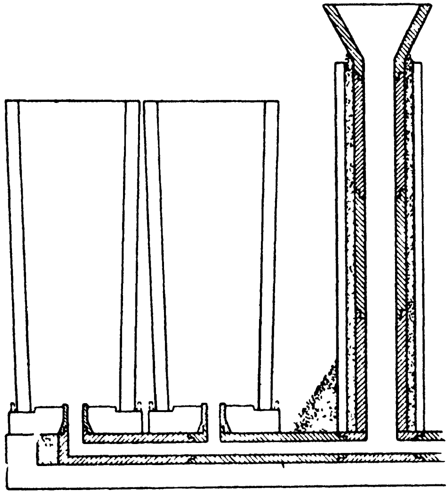
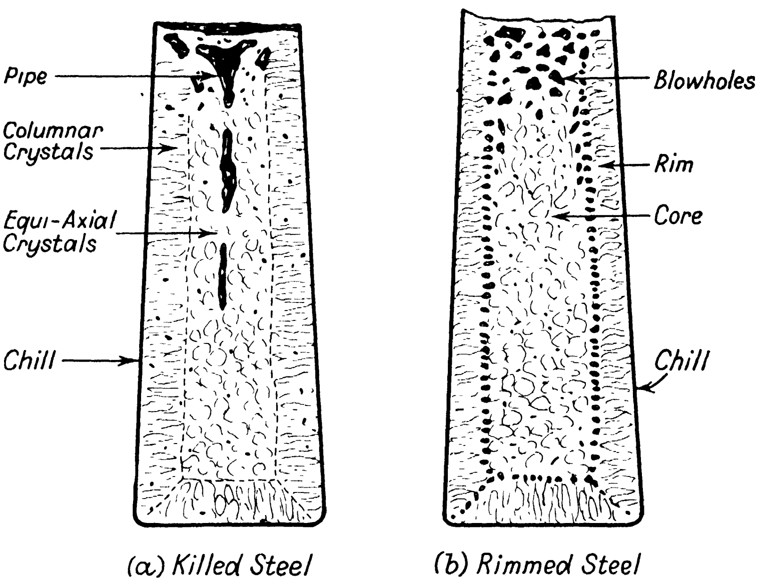


FIG. 144. False mould bottom plate on a bottom-pouring arrangement.
(From " Practical Steelmaking ", by W. Lister, by courtesy of the author.)

It will be noted from Fig. 144 that a false bottom plate is employed over the main bottom plate in which the runner bricks are set. This false bottom plate prevents the metal coming into contact with the refractory runner bricks, reducing the risk of non-metallic inclusions from that source. This bottom may be designed to engage the trumpet and to give it a solid foundation. It may be cast in one part, but more frequently it is made up in sections so as to reduce the weight of replacement, since it is seldom that the bottom wears away evenly over the entire surface. These bottom plates are made of haematite iron and the maintenance of the sections is considerably lower than the replacement



(a) Killed Steel (b) Rimmed Steel

FIG. 145. The crystallization of an ingot.

of the mould bottoms necessary when direct or top-pouring is adopted. The cost of the refractory tubes for the trumpets and the runners, however, represents an appreciable extra, but this extra cost is repaid by the higher quality of the ingot produced.

Types of Ingot

When molten metal is poured into an ingot mould it solidifies in three distinct phases. First a very *thin skin of fine-grained or chill crystals* is formed by the initial chill of the mould. This skin will be of approximately the same composition as the steel in the ladle from which it is teemed. As the steel cools further a second set of crystals begins to grow practically perpendicular to the mould wall, and these *columnar crystals* continue to grow inwards as long as influenced by the wall of the mould. From these columnar crystals off-shoots grow approximately at right angles giving a dendritic formation. In most cases the columnar crystals cease to grow before the centre of the mould is reached and a large number of nuclei appear giving rise to a zone of equi-axed crystals. These phenomena of solidification have been discussed by several authors including Desch.[6] These three zones are shown in Fig. 145.

Since the steel contracts on cooling, a shrinkage cavity is formed at the top of the ingot as shown in Fig. 146. This only applies, however, to a killed steel. If the steel is not killed, or only semi-killed, the crystal formation is disturbed and blowholes persist in the ingot. The position and number of these blowholes depend on several factors, but all these factors are related to the degree of deoxidation or degasification. A complete series of commercial steels is shown in Fig. 146, taken from a paper by the author.[7]

On the left of this figure four examples of fully killed steels are shown, in which the advantage of employing wide-end-up moulds is visible. The use of this type of mould confines the shrinkage cavity to the top of the ingot, whilst in the narrow-end-up mould evidence of secondary piping is seen.

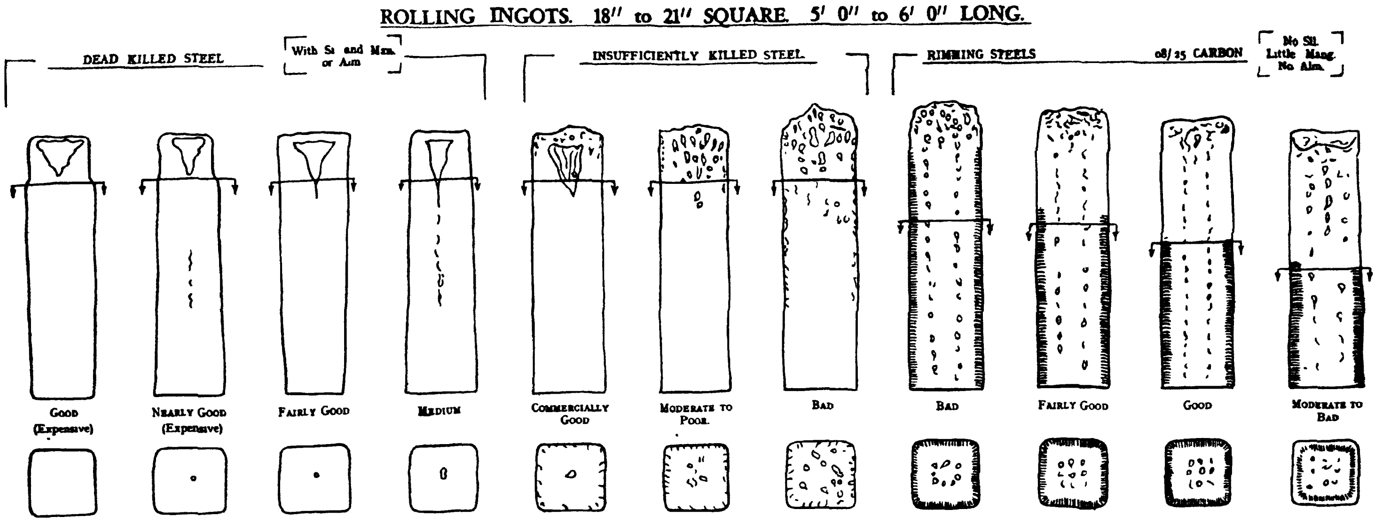
The four ingots shown on the right of this illustration are typical examples of commercial rimming steels. It will be noted that these ingots contain blowholes at varying depths beneath the skin of the ingot. The methods of producing this type of ingot will be discussed more fully in Chapter 15. They represent an important class of steel which possesses properties highly suitable for certain types of employment.

Between these two extremes there is a series of semi-killed or balanced steels, in which the position of the blowholes again varies.

A similar series of ingot types was discussed by the Heterogeneity Committee,[8] and these types are reproduced in Fig. 147. These ingots can be summarized as follows:

Type 1 is a killed steel ingot, properly made, being completely deoxidized, free from blowholes, and having a well-controlled pipe.

Type 2 is a true rimming steel with deep-seated blowholes and a flat top. It should be pointed out that the production of this type of ingot usually demands a carbon content not exceeding 0·15 per cent. and a manganese content of not over 0·50 per cent., with no silicon. In low carbon steels the ratio of FeO in solution to carbon is high and the rim is usually substantial and free from blowholes. Where the carbon



Type 1A	Type 1B	Type 2A	Type 2B	Type 3	Type 4	Type 5	Type 6	Type 7	Type 8	Type 9
Pipe 12½% Confined to Head. No Blowholes. Billet Yield 80%. All Good Sound Billets. 82% Clean Surface.	Top pipe as 1A. Some Secondary Pipe Lower Down. No. Blowholes. Billet Yield 80%. Practically all Sound Billets. 80% Clean Surface.	Top Pipe 20% No. Blowholes. Billet Yield 70%. 85% Clean Surface.	Top Pipe 30%. Some Secondary Pipe. No Blowholes.	Pipe Spreading. A few Blowholes at Top End.	Rise, 1 or 2 in. Fair number of Blowholes with a few on skin. Billet Yield 77½% Unpiped. 5% Rolled up Pipe. 2½% Seamy. 85% Fairly Seamy.	Rise, 2 to 4 in. Large amount of Blowholes. Many on Skin.	Rise of 6 to 9in. No Pipe. Blowholes Bad on Skin. *Billet Yield* Bad 40% to 60%.	Liquid Rise 3 to 4 in Blowholes ½ in. off the Skin (*Thin Skin* Ingot). Billet Yield Good 88% to 90%. (If not long in Soakers).	Liquid Rise 0 to 1½ in. Blowholes ¾ to ½ in. off Skin (*Thicker Skin* Ingot). Billet Yield Good 90% to 92%.	Top, Sunk Top Hat Ingot. Liquid Sink. Slaggy Pipe at Top. Blowholes ¼ to ¾ in. off Skin. Billet Yield Good, Outside Surface Often 92%.

FIG. 146. A series of commercial steels.

(By courtesy of the Staffordshire Iron and Steel Institute.)

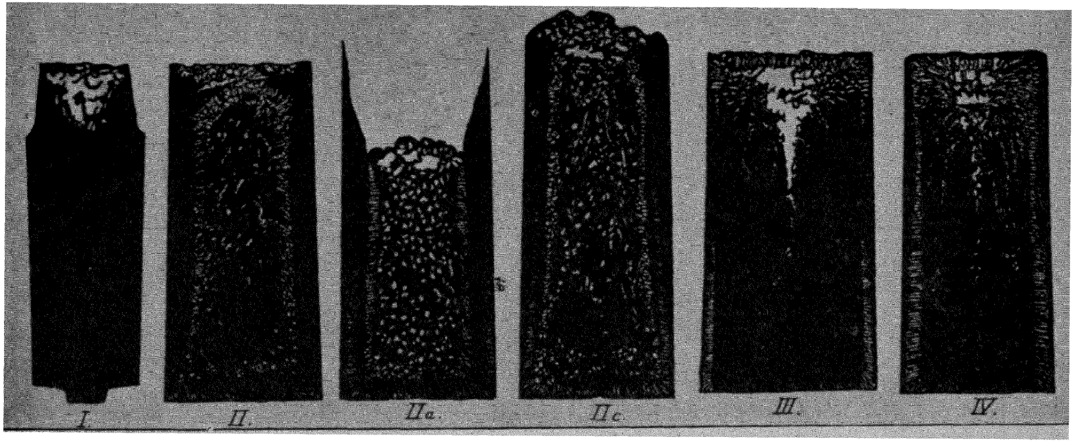


FIG. 147. Ingot types.

(*By courtesy of the Iron and Steel Institute.*)

content approaches 0·25 per cent. and where therefore the ratio FeO/C is low, the rim is less substantial and less defined, and frequently blowholes are found near the skin over the whole length of the ingot. As the ratio FeO/C still further decreases the thickness of the rim decreases and blowholes become more numerous and nearer to the skin.

Type 2A is a "box-hat" ingot of the rimming type and is apparently produced in the manufacture of low-carbon steels when the FeO/C ratio is particularly high. It will be noted that the top of the ingot has sunk considerably from the height to which it was teemed. Subsequent ingots from such a heat could be adjusted to form ingots like Type 2 by addition of aluminium to the ladle.

Type 2C is a rising steel frequently produced when the FeO/C ratio is low.

Type 3 is a semi-killed steel in which a shrinkage pipe or cavity can be discerned together with blowholes in the lower portion of the ingot.

Type 4 is a balanced ingot in which the pipe is practically absent and the blowholes present are fairly deep-seated.

The mechanism of the ingot production as far as killed steels are concerned has been discussed in detail by Brearley and Brearley.[9] Fig. 148 shows the progressive stages in the solidification of a killed steel ingot, and the gradual formation of the shrinkage cavity is clear. As the degree of deoxidation or degasification is decreased this cavity diminishes, and blowholes begin to form and gradually increase in number.

Two theories have been advanced to account for the formation of blowholes:

(1) The Reaction Theory, which assigns the formation of blowholes to the production of CO formed by the interaction of FeO and carbon present in the steel:

$$\mathrm{FeO} + \mathrm{C} \rightleftharpoons \mathrm{Fe} + \mathrm{CO}$$

(2) The Solution Theory, which attributes the presence of blowholes to hydrogen, carbon monoxide and the other gases dissolved or occluded in the steel.

Muller[10] shows that the principal gas occluded in blowholes was hydrogen, together with smaller quantities of nitrogen and a little carbon monoxide. The blowholes near the surface of the ingot frequently contain a higher percentage of carbon monoxide. It has been suggested that these blowholes may be due to the interaction between the carbon in the metal and the oxidized surface of the ingot moulds.[7]

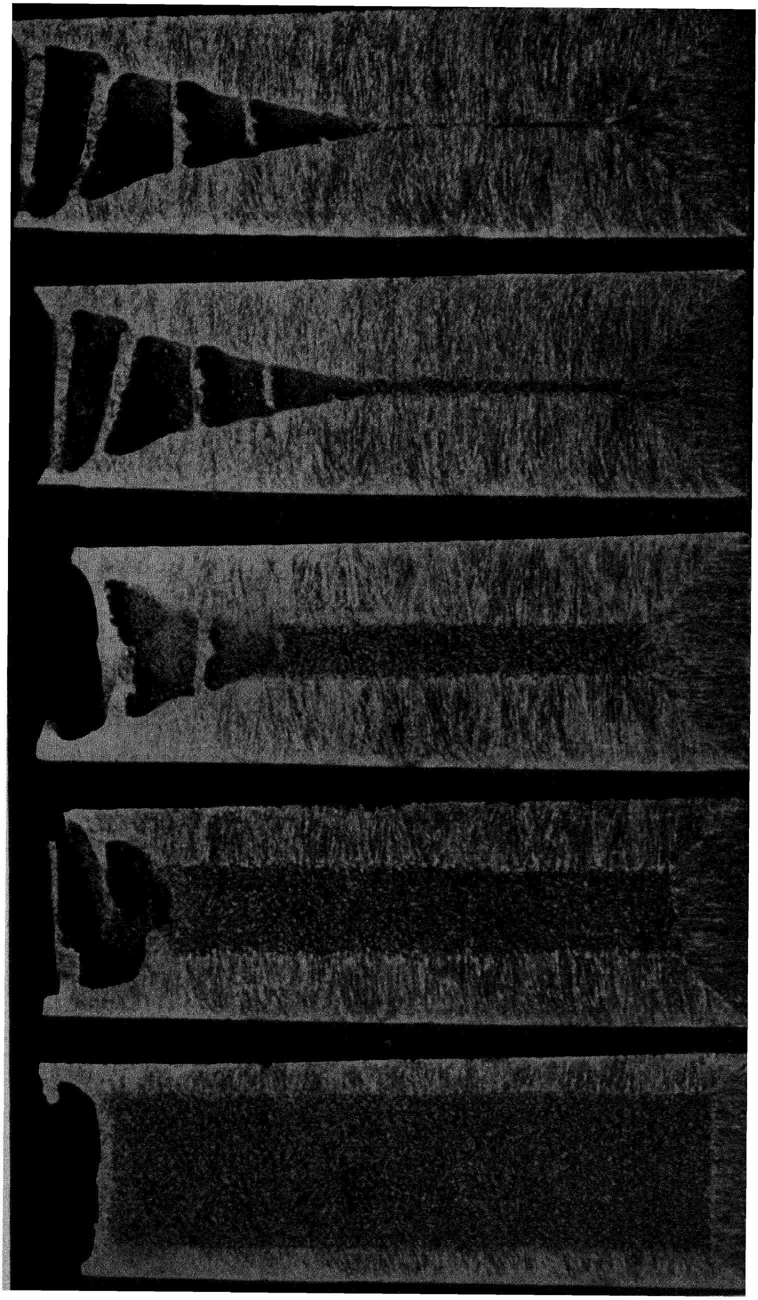


FIG. 148. Progressive stages of ingot solidification.

(*Brearley and Brearley—by courtesy of the Iron and Steel Institute.*)

McCance[11] argued that blowholes were dependent on the concentration of FeO in solution in the metal. An equilibrium will be established, therefore, between the FeO and the CO or CO_2 formed. It is possible that a high concentration of FeO will result in the amount of gas present at the time of solidification exceeding the solubility of steel for that gas, thereby causing blowholes to be formed.

The prevention of blowholes on the other hand can be achieved by reducing the FeO concentration, thereby bringing the CO or CO_2 within the limits of saturation at the point of solidification. This consideration explains the action of deoxidizers in the prevention of blowholes in the light of the reaction theory.

Since this reaction:

$$FeO + Fe_3C \rightleftharpoons 4Fe + CO$$

is endothermic, it will be influenced by the temperature and will proceed with the greatest velocity, whilst the ingot is at its maximum temperature. The velocity of the reaction will gradually decrease as the temperature falls.

These thermochemical considerations suggest that blowholes resulting from this reaction would be near the skin of the ingot. Other factors, however, must be considered. As shown by Hultgren and Phragmen,[12] gas globules formed at the interface of the solid-liquid phase may be swept away if the rate of gas production is rapid and the advance of the solid face is slow. When the gas evolution is slow and/or the growth of the solid face is rapid, the gas globules may be trapped, forming blowholes.

The argument of Hultgren and Phragmen[12] explains the behaviour of rimming steels. When the gas formation is rapid, the liquid is in a state of agitation and the gas globules are swept away until a later stage of solidification, when a solid crust on the top of the ingot hinders or prevents further escape of the gas, and deep-seated blowholes are formed. A slower rate of gas formation, accompanied by the earlier crusting of the ingot surface, prevents the gas globules being removed, with the formation of blowholes near the skin.

This theory would also account for the formation of blowholes near the skin in the lower part of the ingot, where a high ferro-static head retards the rise of the gas globules.

It follows that the weaker the rate of gas formation and the more rapid the cooling of the ingot, especially the crusting over of the ingot top, the nearer will the blowholes be to the skin of the ingot. These facts explain the position of the blowholes in semi-killed and balanced steels.

The study of blowhole formation is complicated by the fact that many of the elements used to deoxidize steel also affect the solubility of steel for those gases normally occluded by the steel.

Baker[13] showed that killed steel contained 1·33 cc. of gas per grm. of steel, whereas unkilled steel contained only 0·66 cc. per grm. of steel. It might be assumed that the difference of 0·67 cc. of gas per grm. is evolved during the solidification of the steel. If the oxygen content of steel is taken at 0·02 per cent., as suggested by Pickard and Potter,[14] the reaction:

$$FeO + Fe_3C \rightarrow 4Fe + CO$$

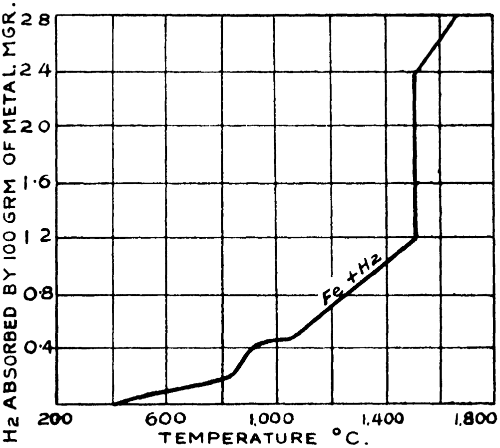


FIG. 149. Solubility of hydrogen in steel. (Sievert.)
(*By courtesy of the Staffordshire Iron and Steel Institute*).

would evolve 0·22 cc. of gas per grm. of steel. Therefore, an oxygen content of 0·05 per cent. would be necessary to evolve the difference indicated by Baker.[13] Both these volumes require correction for temperature.

Sievert[15] proved the solubility of hydrogen in steel and produced the solubility curve shown in Fig. 149. This diagram demonstrates the effect of temperature on the solubility and also the fact that a considerable increase in that solubility occurs at the melting point. As argued by the author,[7] when liquid steel which is saturated with hydrogen solidifies hydrogen will be thrown out of the solution immediately. Under favourable conditions this gas may escape, but under adverse conditions it may be trapped, forming blowholes.

If the hydrogen content is low, its escape may be prevented by the formation of a solid crust at the surface of the ingot and it may be dissolved by the still liquid metal. This concentration would continue until the saturation point was reached and such conditions would favour the formation of deep-seated blowholes.

It may therefore be suggested that the position of blowholes in an ingot will depend on the solubility of the particular steel for hydrogen and the relative amount of hydrogen actually in solution.

McKune[16] showed that manganese increased the solubility of steel for hydrogen and nitrogen, whilst Donaldson[17] showed that silicon and manganese increased the solubility for hydrogen, but decreased the solubility of carbon monoxide in steel. The percentage of manganese and silicon present, therefore, will necessarily affect the position of blowholes.

Brinnell[18] drew attention to the effect of silicon, manganese and aluminium on the presence and position of blowholes, as early as 1902. He stated that "the percentage of carbon and the casting temperature which has until now been considered the principal if not the only factor responsible for the presence and position of blowholes, could be regarded as a secondary but by no means unimportant influence. The principal factor controlling the presence and position of blowholes is the amount of silicon, manganese, and in some cases aluminium, present in the ingot metal at the time of casting".

He also suggested that the formation of blowholes to a large extent depended on whether the evolution of gas took place before or after the solidification of the ingot-top. Any factor, therefore, which advanced or retarded the evolution of gas, such as aluminium, silicon and/or manganese, would affect the presence and position of the blowholes.

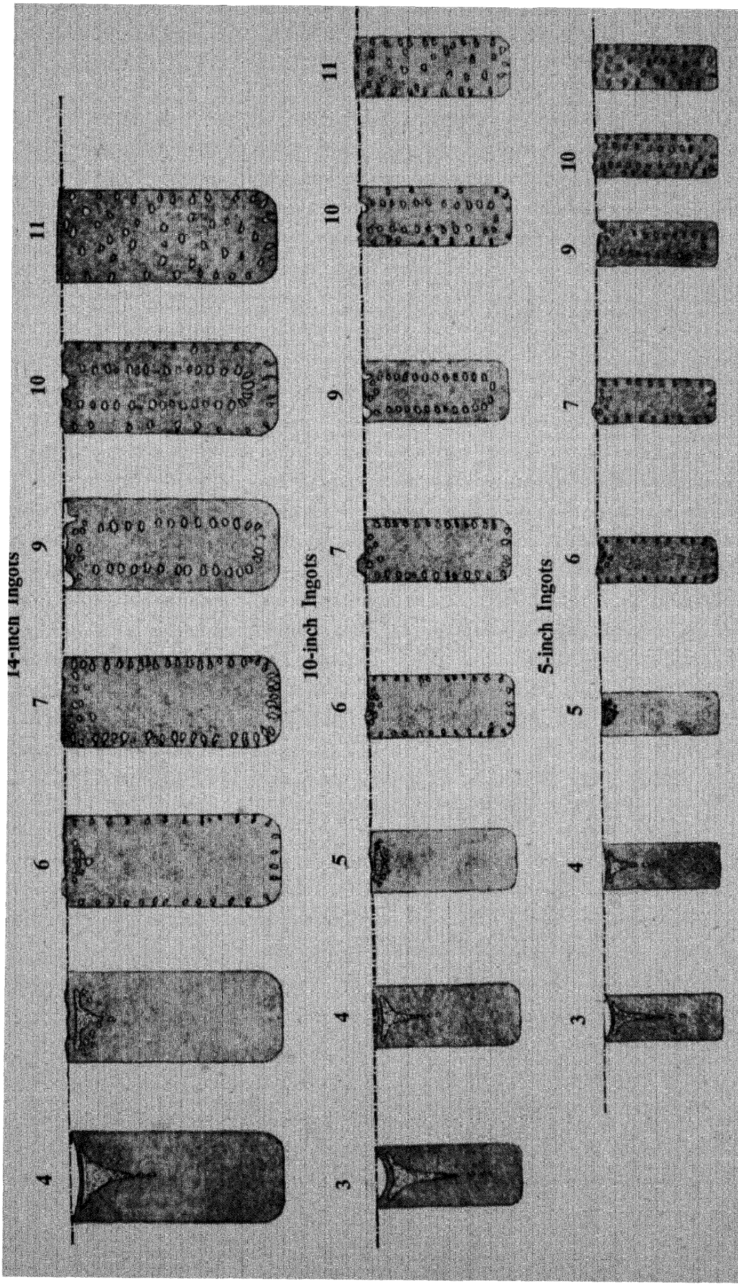
He expressed this relationship in the formula:

$$T = \text{Mn per cent.} + 5{\cdot}2\ \text{Si per cent.} + 90\ \text{Al per cent.}$$

where T was termed the density quotient. By applying this formula to a series of steels containing varying percentages of silicon and manganese, the degree of soundness and the type of ingot structure bore a definite relationship to the value of T.

For 10-in. square ingots it was found that:

When T = 2·05 the ingots were free from blowholes but contained considerable pipe.
When T = 1·66 the ingots showed no blowholes and minimum piping.
When T = 0·28 the ingots contained deep-seated blowholes.



Note: The horizontal dotted lines indicate the height of ingot within the mould immediately after casting.

FIG. 150. Brinnell's ingot series.
(*By courtesy of the British Iron and Steel Research Association.*)

Similar results were obtained when making 14-in. and 5-in. ingots. The value of T necessary to produce a definite type of ingot depended on the mould size, being greater for the larger cross-section of the mould. Typical ingot series are shown in Fig. 150.

Swinden[19] gave an example of the application of this formula to a series of steels containing 0·55 per cent. carbon and 0·70 per cent. manganese, which were cast into 25-in. octagonal moulds. The T value was varied by alteration of the silicon content and the following results were obtained:

Si%	T Value	% Unsound
0·12/0·15	1·40	100
0·15/0·18	1·41/1·60	57·8
0·18/0·20	1·61/1·80	13·8
over 0·20	over 1·80	none

Although this formula is very empirical the author[7] considers it a very useful practical guide. The value of T necessary to produce a definite type of ingot appears to vary with the mould size and design, local practice employed, the casting temperature and the rate and method of teeming. The value of T necessary to produce a definite ingot type must be determined for each particular practice or set of conditions.

Considerable information on blowhole formation has been given by the Ingots Committee in their various reports.[8, 20–27] In their fourth report[8] there is an extremely interesting section (Section IV) in which the principles involved in the manufacture of rimming steel are discussed. They define the characteristics of a typical rimming ingot, which is divided into four distinct zones:

(1) A very thin outer skin of "chilled" crystals of approximately the same composition as the bath. It is suggested that the depth of zone is dependent chiefly on the casting temperature, the specific heat of the mould and the mass of the mould.

(2) Underlying the chilled crystals is a zone composed of material purer than the average, which in practice may contain certain pencil-like blowholes normal to the surface of the ingot.

(3) A narrow zone of comparatively impure material associated with globular blowholes, usually termed intermediate blowholes.

(4) A centre portion less pure than the average, containing large globular blowholes which are generally termed central blowholes.

TABLE XLI

Details of McNair's Ingot Series.

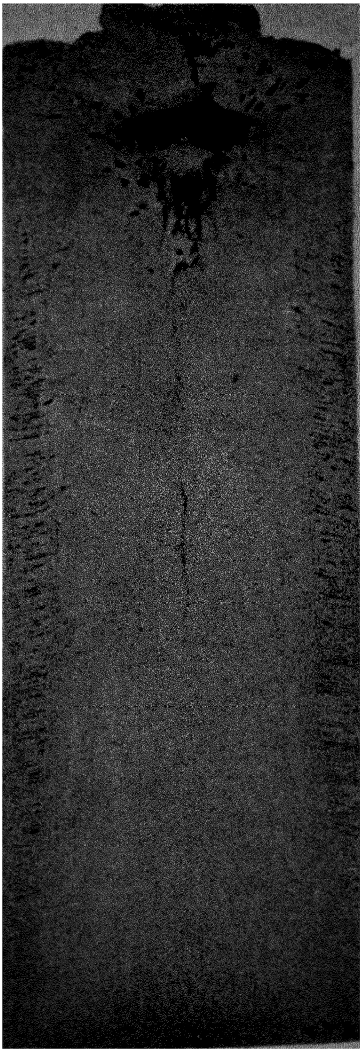
Ingot No. and Type	Additions to Mould	Time Before Top Solid min	Weight of Ingot cwt.	Setting Action in Mould	Thickness of Solid outer skin· in·		Primary Blowholes	Secondary Blow-holes	Central Blowholes
					Top	Bottom			
0. Box-hat.	24 oz. Iron ore	15	17¼	Sink of 5 in leaving box-hat. Rimmed. Formed cauliflower top	2 to secondaries.	0 6	In bottom quarter only.	Full length	Scattered throughout.
1. Level rimmer.	8 oz. Iron ore	13½	18	Rimmed level with no rise or sink. No cauliflower top	2¼ to secondaries	0 5	In bottom half only.	Full length.	Scattered more particularly at top and bottom.
2. Slightly rising rimmer.	Nil	14	18½	Rise of 3½ in Rimmed. No cauliflower top.	2¾ to secondaries.	0 35	In bottom half only.	Full length	Scattered more particularly at top and bottom.
3 Rising rimmer.	3 oz. Al.	15	19½	Rise of 6 in Rimmed Formed cauliflower top reaching 11½ in. above teeming level.	0 1	0 2	In bottom two-thirds only	Full length	Scattered throughout.
4. Rising.	6 oz. Al.	14½	19	Rise of 12½ in. No apparent rimming. Formed cauliflower top.	0·1	0 2	Almost full length.	Full length.	Scattered throughout.
5. (Fully) Rising.	9 oz. Al.	13½	19¼	Rise of 14 in. No apparent *rimming*. Formed cauliflower top	0 1	0 2	Almost full length, and above teeming level at which no discontinuity.	Only in top half.	Only in top half.
6. Semi-killed.	12 oz. Al.	Less than 10 sec.	19¼	Set flat at teeming level	Solid Ingot.		Nil	Nil	Only at top in pipe zone.
7. Killed 8. 9.	16 oz. 32 oz. 80 oz. Al.	less than 10 sec.	19¼ 19¼ 19½	Set flat at teeming level	Solid Ingot.		Nil	Nil	Nil. Shrinkage cavity.

In this report it is suggested that the skin blowholes are associated with hydrogen, whilst the deep-seated blowholes are formed by the interaction of FeO and carbon.

The influence of the method of teeming on the ingot structure is also discussed. The solid rim is considerably thinner in the lower portions of the ingot when top-pouring is adopted. Bottom-pouring, however, reduces this difference in the thickness of the solid rim. There is a tendency for the rim to be thicker in ingots obtained from the later periods of teeming from a given ladle.

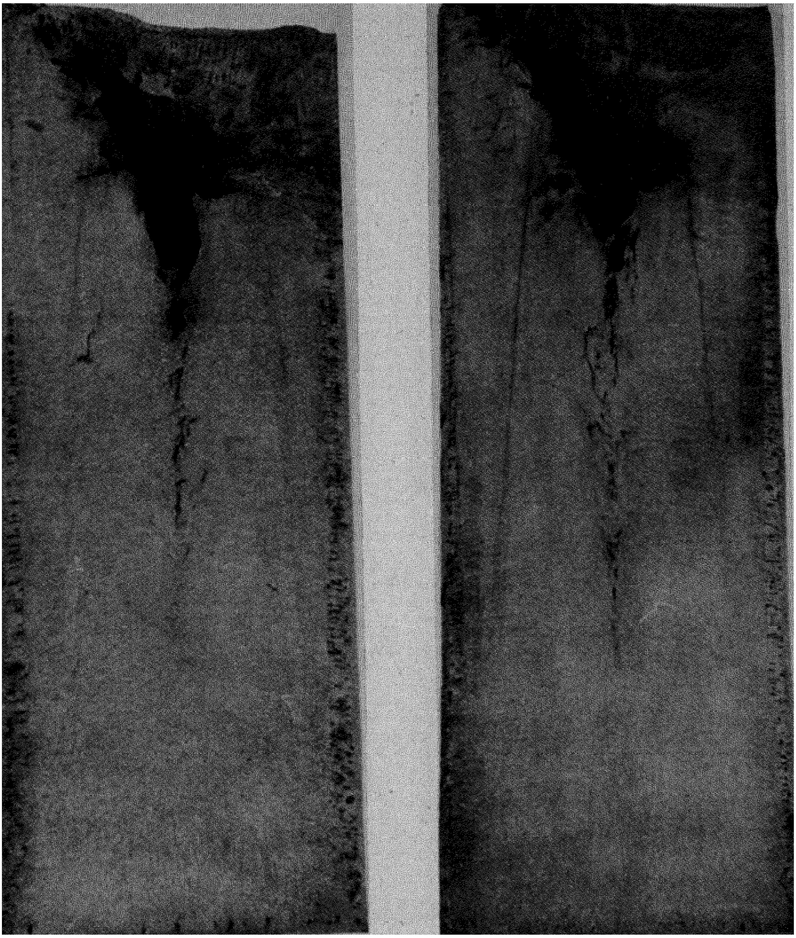
In the first section of this report three examples of ingots containing copper are given. These ingots are reproduced in Fig. 151. Their examination demonstrates the quietening effect of copper additions and the sulphur prints of such ingots indicate that copper tends to bring the blowholes nearer to the skin.

McNair[28] has recently produced a series of ingots ranging from box-hat, through rimming and rising steels, to fully killed steels from the same cast. These various types are produced by suitable iron-oxide or aluminium additions to the ingot moulds. The details of the method of production and the characteristics of these ingots are given in Table XLI taken from his paper. The structures obtained are reproduced in Fig. 152.



(*a*) Copper free

At the time of preparing this MS. the gas contents of the various ingots are not available and further work is in progress. Nevertheless,

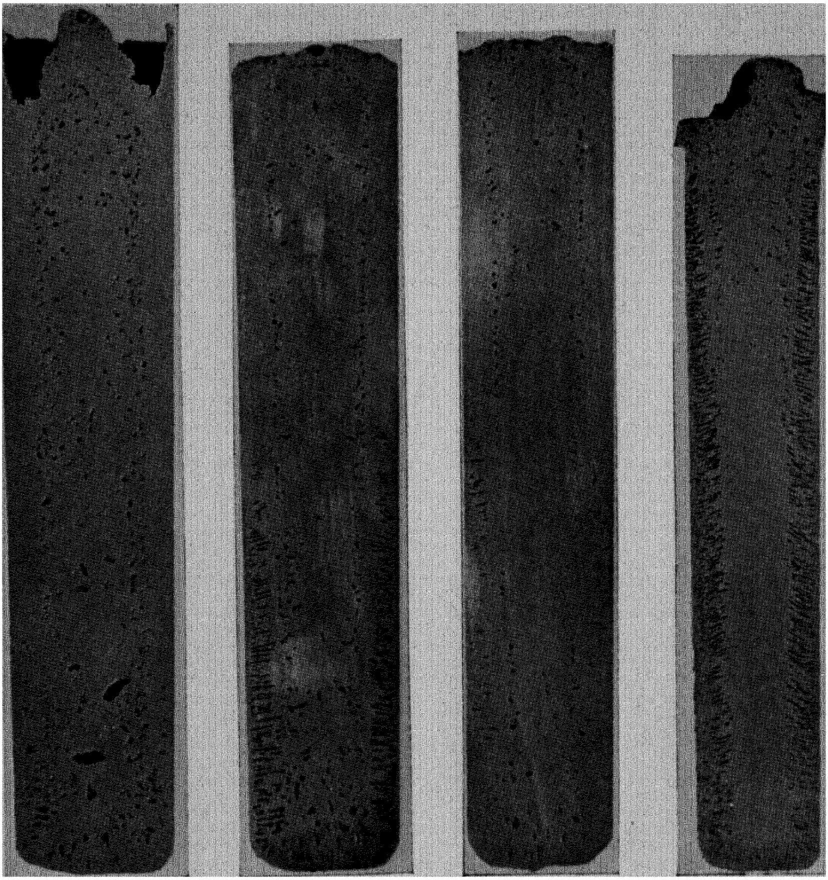


(*b*) Copper 0·25 % (*c*) Copper 0·50 %

FIG. 151 (*a*, *b*, *c*). Effect of copper on blowhole formation.
(*By courtesy of the Iron and Steel Institute.*)

the results obtained so far do support the theory that the presence and position of blowholes are influenced by the FeO/C ratio, and the interaction between FeO in solution and the C present in the steel.

Against this view, however, must be recorded the remarks made by Fleming,[29] whose work is discussed more fully in Chapter 15, pages 420. Referring to aluminium additions to the ingot mould, he adds:



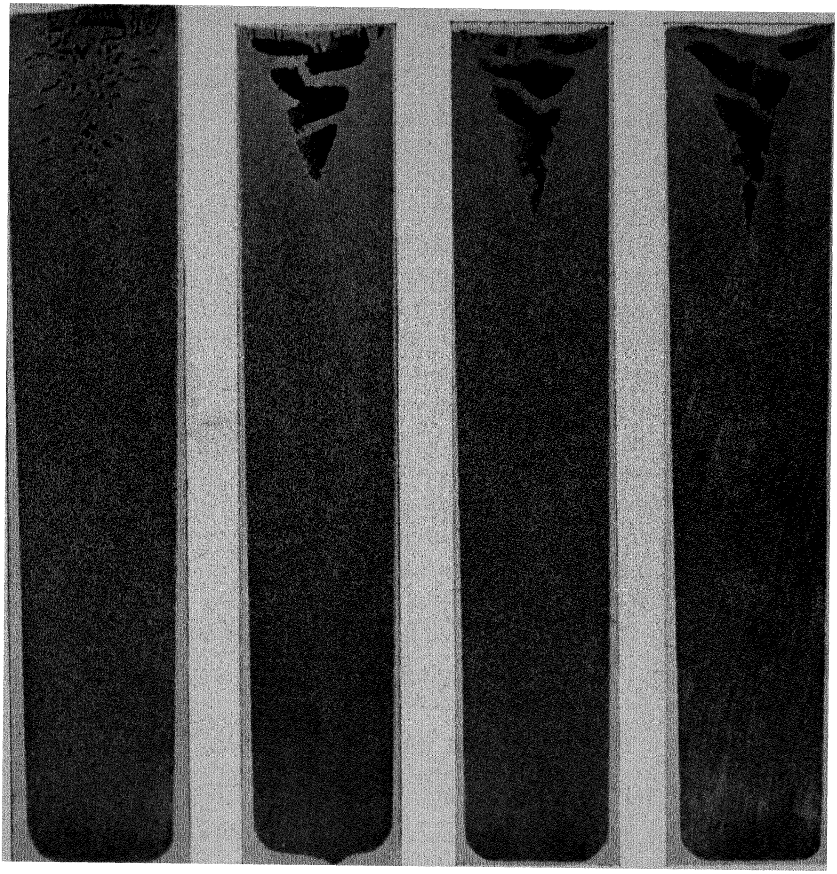
Ingot No. 0 **No. 1** No. 2 No. 5

Fig. 152. McNair's ingot series.

"If the liquid steel in an ingot mould happens to be impregnated with finely divided gases, an ounce or two of aluminium will cause it to sink instantly several inches. This phenomenon cannot be attributed to deoxidation. In some way it greatly increases the solubility of the entrapped gases in the steel."

Ingot Defects

A clear conception of ingot types and the mechanism of ingot formation produces a better understanding of ingot defects and their prevention. The chief ingot defects can be classified as follows:



Ingot No. 6 No. 7 No. 8 No. 9

(By courtesy of the Iron and Steel Institute.

(1) Ingot cracks.
(2) Piping.
(3) Segregation.
(4) Surface defects.
(5) Blowholes.
(6) Non-metallic inclusions.

Ingot Cracks

Cracks form one of the chief defects in steel ingots. These cracks mainly develop during the mechanical working of the ingot, although in most cases they are present in the ingot soon after solidification. The fundamental causes of these cracks are:

(1) The contraction of the skin of the ingot on cooling.
(2) The expansion of the ingot mould.
(3) The ferro-static pressure within the ingot.

When steel is poured into an ingot mould a solid skin is formed due to the initial chilling of the steel. The depth of this skin depends on several factors, which are complex. Immediately this skin has been formed the ingot begins to contract and at the same time the ingot mould begins to expand, resulting in a gap between the mould and the ingot skin. The skin must withstand the ferro-static pressure exerted by the still liquid metal within. If the skin is insufficiently strong rupture will occur and inherent cracks, which develop during the subsequent mechanical working of the steel, are formed.

Field[30] deduced from theoretical considerations that the depth of this skin at any time could be determined by the formula:

$$D = K\sqrt{T}$$

where D is the thickness of the metal in inches,
T is the time in minutes after the metal is in contact with the mould,
K is a co-efficient which varies between 0·9 and 1·22 depending on the mould size and shape.

This formula has since been verified experimentally for killed steel by Nelson,[31] and for rimming steel by Chipman and Fondersmith.[32]

These cracks can be conveniently divided into three groups:

(1) Transverse and/or longitudinal cracks, which occur in the initially formed skin immediately after solidification. These cracks are due to the inability of the skin to withstand the stresses arising from the normal contraction of the ingot during solidification and cooling and the ferro-static pressure exerted on the steel envelope by the still liquid metal.
(2) Subcutaneous cracks, which may or may not break through to the surface of the ingot, which are also developed during solidification.
(3) Restriction cracks, which may take the form of basal, fin, hanger or pulls, and are due to some obstruction in the mould preventing the free contraction of the ingot.

The type of cracks falling in Group 1 may be sub-divided into:

(a) Transverse facial cracks, which are probably the most common, and which result from the vertical contraction of the skin, the internal pressure and the friction with the mould face. This type of crack appears to be accelerated by increasing the ratio of the height of the mould to the cross-sectional area.

(b) Longitudinal facial cracks, which are less common and result from the lateral contraction, friction with the mould face, and the internal pressure. The tendency for the formation of this type of crack appears to increase with the cross-section of the mould and is more frequent in alloy than in plain carbon steel.

(c) Longitudinal corner cracks which are due to the lateral contraction, withdrawal from the corners followed by heating up from within, and internal pressure. In some cases these cracks are aggravated by vertical lines of segregate weakness at the intersection of the corner crystals.

The subcutaneous cracks in Group 2 are similar to the longitudinal corner cracks, but do not come to the surface and cannot be seen until the steel is mechanically worked. They are caused by the same factors as the longitudinal corner cracks. This type of defect is associated with alloy steel more than with plain carbon steel.

In addition to the type of cracks already discussed, there are two more classes, "clinked ingots" and "hairline cracks".

"Clinked ingots" is a term applied to internal cracks which arise from different causes, such as:

(1) Too rapid re-heating, which results in the centre of the ingot being unable to withstand the tension set up by the expansion of the outer portion of the ingot, giving rise to internal rupture. Certain classes of alloy steels are very susceptible to this thermal shock, owing to their coarse and weak crystal structure in the "as cast" condition.

(2) Too rapid cooling of the ingots after stripping from the ingot mould. This results in the rapid contraction of the exterior of the ingot and strain being set up between that exterior and the hot centre of the ingot.

The avoidance of this type of defect depends on the care taken to prevent too rapid cooling or reheating of the ingot. Frequently in practice it is found advisable to strip the ingots fairly early and to transfer them as quickly as possible to a soaking-pit, which is at approximately the same temperature as the ingots. These ingots should then be rolled without being allowed to cool below the critical range, to the smallest possible size. This method of treatment is also helpful in the prevention of "hairline cracks".

"Hairline cracks" are also due to stresses set up within the ingot, and a considerable difference of opinion exists on their true causes. Thompson and Campbell[33] show that these cracks can only be detected by the use of magnetic testing, as shown in Fig. 153, or by macro-etching. Micro-examination will not show these defects unless the specimen is very deeply etched. They are not necessarily connected with any form of segregation, are not confined to any particular part of the ingot, and do not have any specific direction.

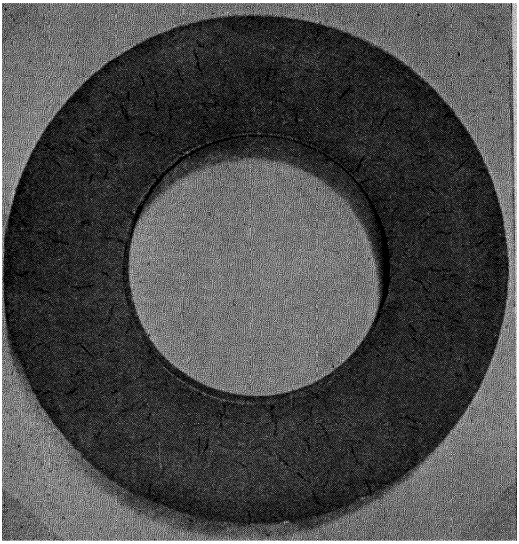


FIG. 153. Detection of hair-line cracks by magnetic testing.
(By courtesy of the West Scotland Iron and Steel Institute.)

As shown by Robertson[34] and many other workers, plain carbon steels are not regarded as being susceptible to this type of defect, although the higher ranges of carbon steels do not appear to be immune. The greatest trouble is experienced with alloy steels, especially nickel-chromium, nickel-chromium-molybdenum and high manganese-molybdenum steels. The general theory advanced to account for this type of crack attributes its presence to hydrogen in solution in the metal, and probably the effect of the alloying elements on its solubility.

The latest work by Andrews and others[35] indicates the following general conclusions:

(1) Hydrogen is the fundamental cause of "hairline cracks".

(2) The hydrogen which subsequently causes cracks is not in solid solution, but for a time is held in the form of a hydrogen-rich constituent which is formed on rapid cooling through the g–a change.

(3) Breakdown of the constituent releases the hydrogen, some of which diffuses into voids and builds up a disruptive pressure.

(4) In addition to hydrogen, the cracks contain methane, even at room temperature, and the presence of this methane is the cause of some of the difficulties encountered in the industrial prevention of "hairline cracks".

(5) The influence of composition upon the susceptibility of a given steel to "hairline crack" formation is explained by the effect of alloying elements upon the stability of the resulting hydrogen-rich constituent.

(6) Once the constituent has been formed, its breakdown at low temperatures must lead to "hairline crack" formation. On the other hand "hairline cracks" will not result if breakdown of the constituent can be brought about at a sufficiently high temperature.

In addition to the precaution already mentioned, the process of steelmaking appears to influence the development of this type of defect. Steel produced by the basic electric process is more susceptible than that produced by the basic open-hearth, whilst acid open-hearth steel is the least susceptible.

As pointed out by Thompson and Campbell[33] there is a considerable school of opinion which suggests that by assuring that the ingot from the time it is cast is never allowed to cool rapidly below a temperature range of 300/350 deg. C., which appears to be a critical range, the tendency to form "hairline cracks" is inhibited, if not entirely eliminated.

Since this type of defect is attributed to hydrogen, any method of controlling this element will be beneficial. Care should be taken to ensure that raw materials used in the steelmaking practice are free from moisture. On many occasions severe "hairline cracks" have been traced to the use of damp feeder-heads. The avoidance of moisture in the casting pit must be regarded as an important factor.

The prevention of the type of cracks referred to in Group 1 was the subject of an interesting paper by Gathemann.[36] The best method of prevention is to ensure that a sufficiently strong skin is formed before the mould wall and ingot skin draw apart. When the chilling effect is high, the skin formed is strong and there is less tendency for cracks to develop.

Round moulds are found to increase the susceptibility to cracks of this type. In a cylinder the ratio of the perimeter to the cross-sectional area is at a minimum and therefore the initial chill and the subsequent rate of cooling are reduced. The skin of a cylindrical ingot cannot contract laterally without reducing the cross-sectional area and either the pressure within the mould increases to force the liquid higher up the mould or the skin ruptures. Any departure from the cylindrical type of mould will, therefore, reduce the tendency for cracks to develop.

In an ingot mould six feet high the pressure one foot from the base is five times the pressure one foot from the top of the ingot. In order to withstand this extra pressure the skin at the base of the ingot should be proportionally thicker. This fact explains the tendency for this type of crack to form at the bottom of the ingot. It also provides a reason for the advantages of bottom-pouring, whereby the skin at the bottom of the ingot is allowed to assume substantial proportions before it has to withstand the full ferro-static pressure from within.

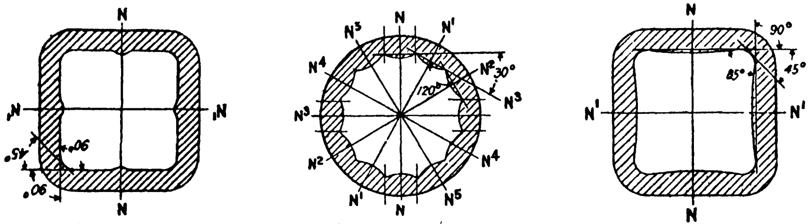
It is, therefore, obvious that several causes can affect the degree to which ingots tend to crack and these causes may be summarized under the following headings:

Mould design. In the design of ingot moulds flexibility of the perimeter, chilling effect, taper, wall thickness and corner radius appear to be the more important features.

As already pointed out any departure from the cylindrical form provides greater *flexibility* and enables the skin of the ingot under stress to accommodate itself more easily to the new conditions prevailing. A square or a fluted ingot mould allows the skin of an ingot to contract with less tendency than the simple cylinder to reduce the cross-sectional area and thereby facilitates the relief of stresses within the ingot.

Gathemann[36] showed that the purpose of corrugating moulds was to increase their perimeter and to provide more salients, which would allow the initial skin to solidify more quickly. The corrugations would also resist the tendency for the skin to tear during solidification. It is obvious, therefore, that flexibility and *chilling effect* are closely associated.

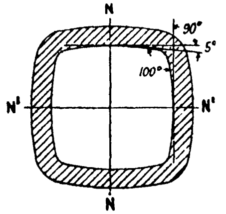
In designing an ingot mould the first two considerations are the freedom of the ingot to contract without binding on the mould wall and the assurance that the salients and corners of the mould are of such an angular contour that they will not be decarburized during reheating. With these general principles in mind, Fig. 154 shows the development of the Gathemann mould. The first mould shown is the Mooney patent of 1873, which was probably the first attempt at applying these considerations to mould design. The octagon and dodecagon "b" provided a better surface solidification than the older types of ingot moulds, but they were never extensively employed owing to the need for more passes in the mill in squaring up the cross-section, which resulted in loss



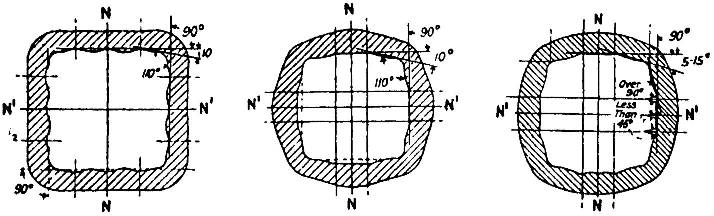
(*a*) —Earliest Rectangular, Corrugated Mould

(*b*) —Round-Dodecagon Corrugated with Sharp Salients.

(*c*) —Early Type of Rectangular Mould, with Narrow Corners and Convex Sides



(*d*) —Good Corner Construction with Concave Sides.



(*e*) — Improvements over the Early Forms of Rectangular Corrugations.

(*f*) —Design Yielding a Flexible Ingot.

(*g*) —Variation of the Gathmann Principle

FIG. 154. Development of the Gathemann mould.

of output. The third example, "c" is a rectangular mould which has been extensively employed. It has four inwardly convex side walls, which allow moderate adjustment of the skin during solidification. Sharp corners, however, frequently overlap during cogging and may result in serious surface defects.

Example "d" is the antithesis of example "c" and has been employed widely. The initial skin of the ingot is bowed outwardly and cannot readily adjust itself to the initial contraction and expansion of the mould chamber during solidification. This results in numerous shrinkage cracks and incipient stresses being found in the skin of the ingot.

Example "e" has a rippled design with broad corners which will not readily lap in light rolling passes. The free sides, however, are heavily stressed during the initial passes. Gathemann suggests that the width of the primary sides cannot spread in a side-wise direction and must elongate the ingot, which causes overstressing of the unworked sides.

Examples "f" and "g" which were developed and recommended by Gathemann, have been extensively adoped in America. He feels that the cross-sections shown possess the necessary features of good mould design for both normal and inverted practice. The lines connecting the corners of the mould matrix are so corrugated as to offer no obstruction to free contraction of the metal towards the neutral lines of shrinkage. The corrugation of the secondary corners renders the ingot surface flexible so that it can yield without breaking or cracking during the period of separation of the mould and ingot, subsequent to the expansion of the mould. The corners are broad, so that they are not likely to be burnt during reheating or to lap during the rolling operations. The projecting primary sides of the ingot are spread in a direction which is approximately a tangent with the axis of the rolls in the early stresses, whereby the free unworked sides are not unduly stressed.

It is impossible, however, to review all the excellent work which has been done in this country and abroad on the question of ingot mould design, but reference is made to this work in the bibliography at the end of this chapter. It is sufficient to summarize the chief features as follows:

(1) *Relation of periphery to cross-section area.* In a 10-in. round mould the periphery is 31·42 in., whilst the cross-sectional area is 78·54 sq. in. which gives a ratio of 0·40. A square mould with the same cross-sectional area would be 8·86 in. in dimension and would have a periphery of 35·40 in. This mould would have a ratio of periphery to area of 0·45. The greater this ratio the

greater will be the chilling effect and the greater the flexibility in the adjustment of the skin.

(2) *Corner radius.* The corner radius appears to have a distinct effect on the development of corner cracks. If the radius is too sharp external cracking will occur, whilst when the radius is too great internal cracks may result. The corner radius appears to vary from ¾ to 3-in. with satisfactory results. It has been suggested that on a 20-in. mould this radius should be between 1½ and 2 in., but the author has found that on 21-in. square moulds the most satisfactory results were obtained when this radius was only ½ in. Local practice may be the best guide within the limits suggested.

(3) *Mould taper.* However clean and smooth the interior of the mould, insufficient taper will tend to promote lateral cracks. If the ingots are stripped early insufficient taper will also develop major pulls. Excessive taper, however, introduces manipulative difficulties in the rolling operations. The general practice is to allow between ¼ and ½ in. per foot. The author has experienced a slightly less taper giving very satisfactory results in narrow-end-up moulds, where the taper was reduced to minimize the piping.

(4) *Wall thickness.* McCance[37] has shown that wall thickness does not affect the thickness of the initial chill. This fact is deduced from theoretical considerations in a very able manner in the paper mentioned.

Although it has no effect on the initial chill, the wall thickness does affect the expansion of the mould. The rate at which an ingot cools is affected by the temperature on the interior and exterior mould face. The rate of heat flow will be expressed by:

$$Q = \frac{(T' - T'') \times K}{d},$$

where T′ is the inner face temperature,
T″ is the outer face temperature,
K is the thermal conductivity of the mould,
d is the thickness of the mould wall.

Whilst the mould is in contact with the ingot skin the inner face will approximate the same temperature as the ingot skin and the heat flow will be represented by the diagram in Fig. 155 taken from the paper by McCance. When the mould face and ingot skin have pulled apart, the air gap between them will act as an insulator and will cause a drop in the rate of heat transfer, as demonstrated by McCance and shown in Fig. 156. Mathematically McCance[37] also shows that thick moulds do not separate so quickly from the

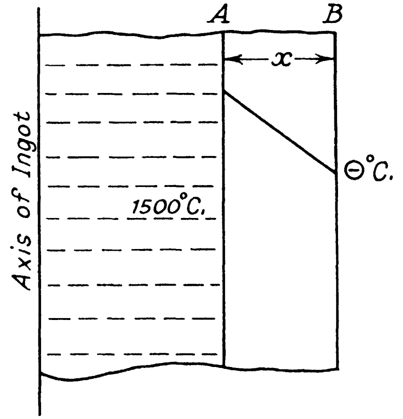


FIG. 155. Heat flow through mould wall. (McCance.)
(By courtesy of the West Scotland Iron and Steel Institute.)

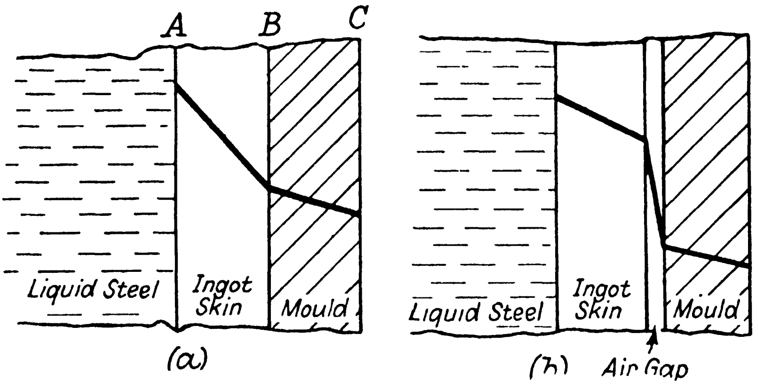


FIG. 156. Effect of air gap in ingot cooling. (McCance.)
(By courtesy of the West Scotland Iron and Steel Institute)

ingot skin as do thin moulds, consequently they offer more support to the frail ingot skin when it is first formed. This statement is based on the argument that if the breadth or diameter of the mould is L and the average temperature is θ, then the expansion will be:

$$L \times \theta \times a,$$

where a is the co-efficient of expansion. This expression can be simplified so that the function is actually:

$$\frac{\text{Time}}{(\text{Thickness})\ 2}$$

Russell[38] shows that the rate of solidification is accelerated by increasing the mould thickness until the mould ratio is about 0·8 to 1·0. This ratio of mould weight/ingot weight does appear to have a considerable bearing on the tendency to cracked ingots, and most operators would advocate mould ratios between the figures quoted.

Teeming speed. A very definite relationship has been established between the rate of teeming and the incidence of cracking. At constant temperatures an increase in the speed of teeming will tend to promote cracking. This factor was demonstrated by Kilby[3] as early as 1916. The speed of teeming may be controlled as already mentioned on page 273 by:

(1) Special nozzles and direct pouring.
(2) Tundishing.
(3) Bottom pouring.

It is obvious that the rate of rise in the ingot mould will materially affect the thickness of the skin before it has to withstand the ferro-static pressure due to a considerable head of metal, and will therefore affect the tendency for the ingot to develop cracks. The thicker this skin, due to a slower rate of rise, the less will be the likelihood for cracks to be formed.

Since the chilling effect of the ingot mould decreases with increasing the diameter of the mould, the safe rate of rise must be proportionally reduced as the moulds increase in size. This rate of rise, however, must be related to the *temperature* of the metal during the actual teeming. This feature is demonstrated by Reeve.[39]

The rate of teeming and the temperature of the metal during pouring must, therefore, be a compromise. If the metal is teemed too fast or too hot, cracks will develop. If on the other hand the temperature is too low, or the rate of rise in the ingot mould is too slow, the ingot surface may be rippled, oxide inclusions may be trapped and the surface of the ingots may become defective, leading to other types of surface defects. The optimum rate of rise for each particular type of steel must be determined in relation to the mould size, the casting conditions and local practice.

Composition of the steel. The composition of the steel has also been shown to have a decided effect on the tendency to produce cracked ingots. Heavily killed steels have a greater tendency to crack than other types of steel, and this tendency is probably connected with the greater contractional effects produced in such ingots.

Blowholes existing too near the surface of the ingot due to incorrect deoxidation may develop into cracks during the cogging of the ingot.

Several workers, including Reeve,[39] have demonstrated the adverse effect of *sulphur* on the mechanical working of steel. It has been shown that sulphur has a pronounced effect in increasing the susceptibility of steel to cracks both in the ingot as cast and in the subsequent cogging of the ingot. Norris[40] in a paper dealing with the factors affecting red-shortness indicated the relationship between the sulphur, oxygen and manganese contents of the steel.

It is suggested that the low solubility of FeS in gamma iron causes precipitation of this compound at the grain boundaries, resulting in red-shortness. The tendency to develop red-shortness can, however, be diminished by a higher manganese content, in which case the sulphur is converted into MnS. It is shown that a high FeO content favours red-shortness due to:

(1) The decreased solubility of FeS in gamma iron.
(2) The oxidation of the Mn, thereby reducing the amount of MnS.

The general conclusions of this work[40] appear to be:

(1) That the chief cause of red-shortness is sulphur and oxygen is a secondary factor. The manganese content necessary to prevent red-shortness due to this cause depends on the sulphur and oxygen present, and on the cooling conditions previous to mechanical work.
(2) That if the steel is worked in the red-short range immediately after solidification the ratio required is 6·46.
(3) That the ratio (Mn + 0·048) (S + 0·130) is related to the forging properties.
(4) That if the ratio is below 3·30 the steel will be red-short.
(5) That if the ratio is between 6·46 and 3·32 red-shortness can be prevented by cooling to room temperature after solidification and before heating for mechanical work above the red-short range.

The rolling properties are also affected by the *residual metals*, such as tin and copper, present in the steel. It would appear that these effects become pronounced when the tin exceeds 0·05 per cent. and the copper is over 0·20 per cent.

It has been found by many operators that steels containing 0·15 to 0·22 per cent. carbon are more susceptible to cracking than other ranges of carbon steels. Andrews and Binnie[41] drew attention to the *peritectic change* which must be considered when dealing with the freezing of mild steels. They point out that the peritectic change mentioned was not a sluggish change, but that the thermal point on cooling was sharp and suggestive of the sudden freezing of a liquid. This peritectic change, although still present, diminishes in intensity in steels over 0·30 per cent. carbon. This consideration probably accounts for the marked tendency of steels within the 0·16 to 0·25 carbon range to develop cracks.

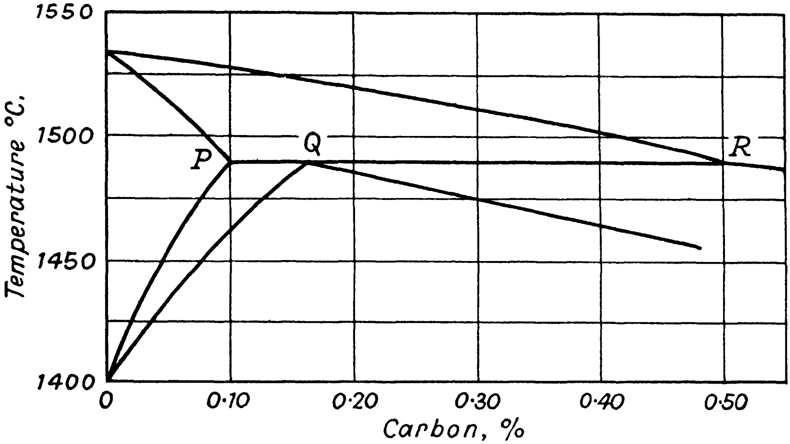


FIG. 157. Delta-gamma range diagram. (Binnie.)

Binnie[42] shows that the maximum amount of steel freezing at the peritectic temperature occurred at the extreme limit of the delta-gamma range, Fig. 157. This maximum falls rapidly towards zero at 0·10 per cent. carbon and decreases more gradually towards zero at 0·51 per cent. carbon. Steels within the range of 0·15 to 0·22 per cent. carbon were, therefore, the most susceptible to the influence of the peritectic change, a range in which snake-like cracks are prone to develop during the rolling of the ingots. It is probable that these cracks are actually present during the cooling of the ingots and are developed during the mechanical working of them. It is possible that they are formed because of the volume change which takes place at the point of solidification within this range. The effects of this change will be affected by the rate of cooling.

Casting-Pit Practice. There are several features of casting-pit practice which can greatly affect the possibility of cracked ingots. *Hot moulds* will naturally promote cracking owing to the reduction of the chilling effect of the moulds and the retarding of the early formation of a substantial skin. If the moulds are very hot the ingots may adhere to the moulds, producing increased friction between the mould face and the ingot skin, which will aggravate cracking.

It is generally recommended that the moulds should be at a blood temperature, because although a cold mould increases the chill effect it introduces the possibility of moisture which would lead to other troubles.

Moulds should be carefully examined and cleaned. *Dirty and worn moulds* tend to reduce the chilling effect, prevent freedom in the contraction of the ingot skin and cause roughness on the surface of the ingot. When feeder heads are employed care should be taken to ensure that no restriction prevents the free contraction of the ingot, otherwise major pulls will result.

Serious *surface defects* can also be caused by splash, due either to opening out the stream too quickly or by teeming from an excessive height. The increased pressure of the stream causes a spraying effect on the side of the mould, producing a double skin with probably an oxide film between the two layers of metal. This aspect of surface defects has recently been discussed by Walker.[43] The effect of splash can be minimized by opening out more slowly or by the use of splash pans or buckets. These splash buckets can be constructed of thin steel sheets or some material such as cardboard or roofing felt. In the case of the metal type they are dissolved by the steel when a sufficiently deep well of metal has collected in the mould to prevent the splashing of the mould wall. The cardboard and felt types allow sufficient metal to collect before they are consumed.

Friction between the mould face and the ingot skin can be considerably reduced, with very beneficial results as regards crack formation, by suitable preparation of the mould face. Several mould varnishes are now on the market and in many cases have replaced the older method of tarring the mould surface. The method of applying these mould dressings, however, demands careful attention, as careless application can produce serious troubles.

Where the mould dressing is applied too thickly hot spots may be formed, which result in skin weakness and crack formation. Spraying of the dressing is advocated so as to obtain an even coating. Properly applied, suitable dressings reduce the friction between the mould face and the ingot skin, minimize the effect of the splash and produce a cleaner and smoother skin.

The more important mould dressings are: anhydrous tar, bituminous paint, graphite wash, Montan wax, pitch, lime wash and aluminium paint. The chemical composition of the mould dressing, however, should be considered in connection with the type of steel being made. If low-carbon steels, which have not been completely deoxidized, are poured into tarred moulds, interaction between the FeO in solution and the carbon in the tar may give rise to blowholes. In the reverse manner a similar blowhole formation may result from interaction between an oxidized mould surface and the carbon present in the steel.

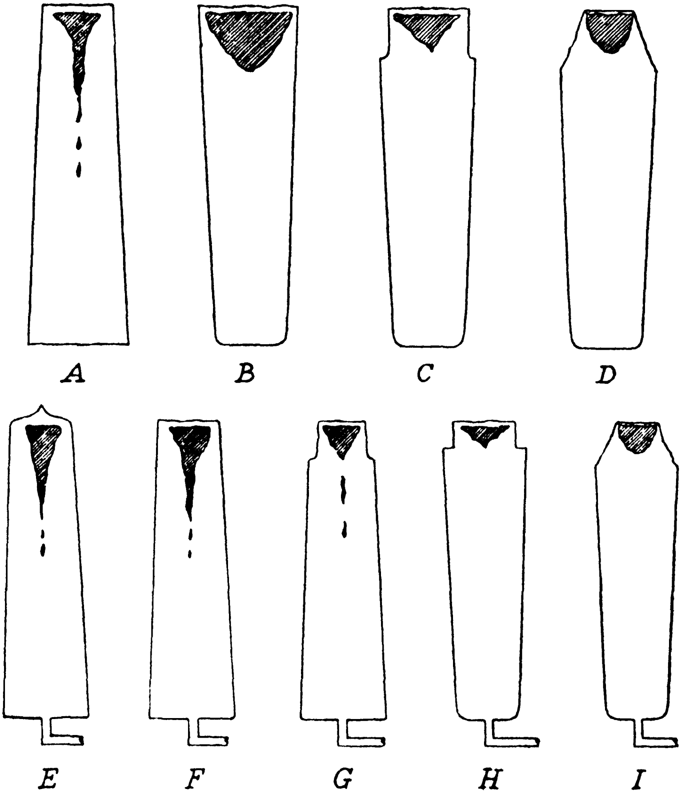


FIG. 158. Kilby's ingot series.
(By courtesy of the Iron and Steel Institute.)

Piping

The shrinkage cavity which results from the volumetric contraction of the steel during solidification and cooling is termed piping. This shrinkage cavity amounts to between 2·5 and 3·0 per cent. of the total ingot volume, according to the degree to which the steel is deoxidized or degasified. In certain classes of steel such as rimming steels, this piping is absent owing to the shrinkage cavity being replaced by a blowhole formation as shown in Fig. 147. In other classes of steel, however, it is possible to have both a central shrinkage pipe and internal porosity due to blowholes, which are shown in the same illustration.

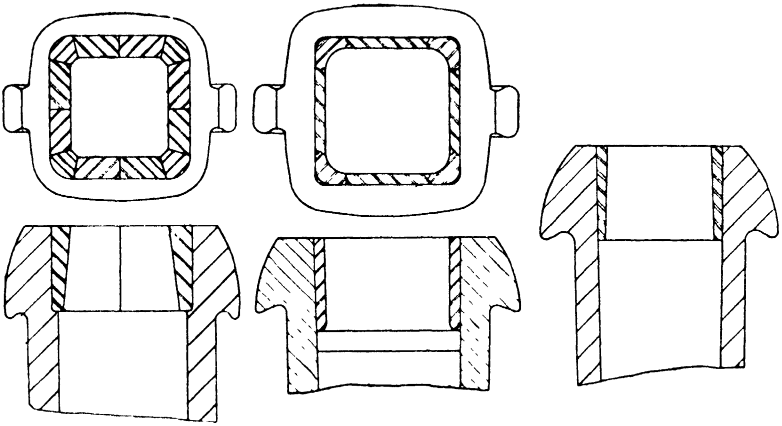
Although this shrinkage cavity occupies a comparatively small percentage of the total volume of the ingot, it may be so dispersed that a large portion of the ingot must be discarded to ensure the production of sound blooms or billets, as shown by Kilby[44] and reproduced in Fig. 158. The details of these ingots and the position and extent of the pipe are given in Table XLII.

TABLE XLII

Details of Ingots (Kilby[44])

Ingot	Type of Mould Used	Average Pipe and Segregate Zone	Surface	
A	Top-poured, narrow end at top, no brick head, usual Bessemer mould.	Up to 50 per cent. of the ingot. Liable to secondary pipe.	Cracks and splash.	Practically free from occluded fluxed brick.
B	Top-poured, wide end at top, no brick head.	Up to 35 per cent. of the ingot. Traces of secondary pipe.	Same as A	Same as A
C	Same as B, with brick top	Under 12 per cent. of ingot.	Same as A	Same as A
D	Same as B, with " sand " head.	Same as C.	Same as A	Same as A
E	Bottom poured, narrow end at top, closed top.	Up to 80 per cent. of ingot. Always has secondary pipe. Worst possible ingot for pipe and segregation.	Good	Liable to be occluded fluxed brick.
F	Bottom-poured, narrow end at top, no brick top, open at top.	Up to 60 per cent. of ingot. Always has secondary pipe.	Good	Same as E
G	Same as F, but with brick head.	Up to 40 per cent. of ingot. Nearly always has secondary pipe.	Good	Same as E
H	Bottom cast, wide end at top, adjustable brick head.	Under 10 per cent. No secondary pipe.	Good	Same as E
I	Bottom cast as H, but with " sand " heads.	Same as H.	Good	Same as E

When steel is cast into an ingot mould a solid skin is formed on contact with the mould face. This skin has the same approximate dimensions as the ingot mould. As cooling, solidification and further cooling proceed, contraction of the metal must take place in a dead-killed steel, and therefore piping is inevitable. The effect of piping can, however, be reduced by the employment of wide-end-up moulds, which of course necessitate more work in the casting pit. Even the use of this type of mould still renders necessary the discarding of a fair portion of the ingot to assure sound blooms.



Small-end-up mould (22½ in. sq.) with bricked head.

Wide-end-up mould (23 in. sq.) with fitted bricks.

Wide-end-up mould (20 in. sq.) with loose tile bricks.

FIG. 159. Bricked topped moulds.
(By courtesy of the British Iron and Steel Research Association.)

The serious effects of piping can be further reduced by the use of a refractory head or hot-top, various designs of which are shown in Figs. 159 and 160. A refractory head or hot-top consists of a refractory or insulating lining placed at the top of the mould which retards the transfer of heat from the mould and thereby forms a reservoir of molten metal which feeds the shrinkage cavity as it forms. In this manner the cavity or pipe is restricted to the feeder head or top portion of the ingot and the discarding of this head should ensure that the remainder of the ingot is perfectly sound. Usually this head is about 14 to 15 per cent. of the total volume of the ingot mould.

The original type of head employed appears to have been the refractory tiles, which was fixed into position by means of supports.

When these tiles had been wedged into position the supports were removed. The chief disadvantage of this type of head is the fact that it must be renewed after each heat, and is, therefore, costly.

The superimposed feeder-head shown in Fig. 160 is more effective. This type of head consists of an iron casting lined with a suitable refractory. This refractory lining may consist of specially shaped bricks or it may be rammed with ganister or other suitable material. In the case of the rammed head, ganister or similar material is rammed around a former head which is removed when the ramming has set. Usually these heads are faced with a fireclay, graphite or fireclay-graphite wash and are capable of being used several times before being re-bricked or re-rammed.

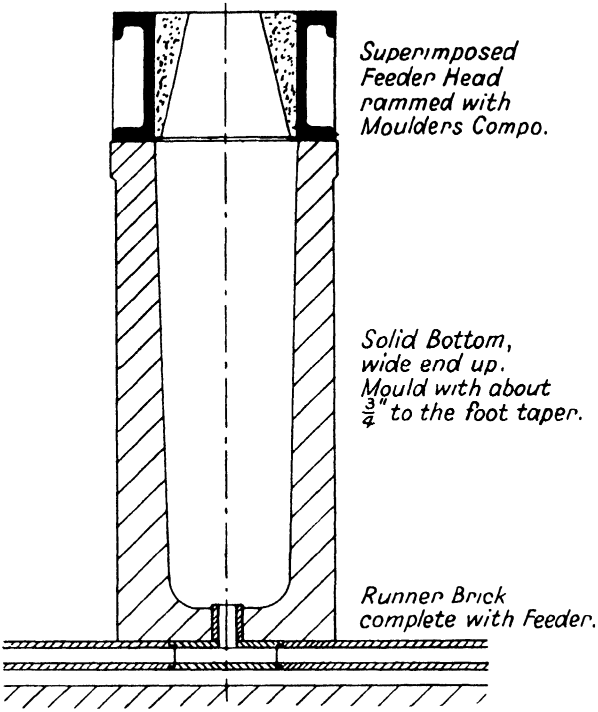


FIG. 160. Superimposed feeder head.

Although the superimposed feeder-head appears to be the more effective, great care must be taken to ensure good joints between the mould and the head, otherwise serious pulls may result. Metal will find its way into any bad joints and crevices between the head and the mould, causing a fin, which will prevent the free contraction of the ingot and give rise to major pulls.

The employment of refractory heads retards the lateral diffusion of head, but unless the top of the ingot surface is insulated, heat will be dissipated vertically and this will affect the extent of the piping. Attention was drawn to methods for preventing this dispersion of heat by Hadfield and Burguss,[45] who produced a very sound ingot. The ingot top can also be insulated by a layer of ashes thrown on the ingot surface as soon as teeming is completed.

In recent years several anti-piping compounds have been introduced to increase the effectiveness of the hot-top and to prevent the dissipation of the heat vertically to the atmosphere. The main object of the employment of these anti-piping compounds is to confine the primary pipe within the feeder-head and thus to increase the proportion of sound material.

These compounds consist essentially of carbonaceous matter and irreducible oxides. When they are placed on the surface of an ingot, the carbon and some of the other elements slowly oxidize, producing heat. The non-metallic matter remains on the surface of the ingot as a residue which insulates the molten metal underneath. This combination of heat generation and heat insulation delays the freezing of the metal in the feeder-head or hot-top and renders the feeding of the pipe more effective.

TABLE XLIII

COMPOSITIONS OF PROPRIETARY ANTI-PIPING COMPOUNDS (Gregory[46])

Number	1	2	3	4	5
Al_2O_3 per cent.	41·65	62·20	7·56	13·43	13·09
SiO_2 per cent.	4·59	18·20	22·50	22·10	18·60
Fe_2O_3 per cent.	2·52	2·60	3·50	6·10	1·71
CaO per cent.	2·09	3·50	—	0·80	—
MnO per cent.	0·17	—	0·44	—	0·09
MnO_2 per cent.	—	—	—	—	—
Na_2O per cent.	4·24	—	0.72	—	0·636
K_2O per cent.	—	—		—	
CO_2 per cent.	6·02	—	—	—	—
Mg per cent.	—	—	—	—	0·52
MgO per cent.	0·52	2·89	0·21	0·65	0·23
CuO per cent.	—	1·20	—	—	—
NaCl per cent.	—	2·18	—	—	—
KCl per cent.	—	0·36	—	—	—
Carbonaceous matter per cent.	21·33	6·80	59·56	48·17	60·45
Volatile matter per cent.	—	—	—	8·50	—
Loss on ignition, per cent.	—	—	—	56·82	63·50

In the earlier experiments it was found that some carbon was absorbed by the steel, but that disadvantage does not appear general, especially when compound No. 5 is employed. Typical analysis of these anti-piping compounds is given in Table XLIII.

A very thorough investigation into the merits and demerits of these compounds was recently conducted by Gregory.[46] The primary object of these experiments was to determine the extent of the piped portion of blooms produced from:

(1) Ingots cast in the normal method into a feeder-head mould.

(2) Ingots in which the surface of the metal in the feeder-head had been treated with an anti-piping compound.

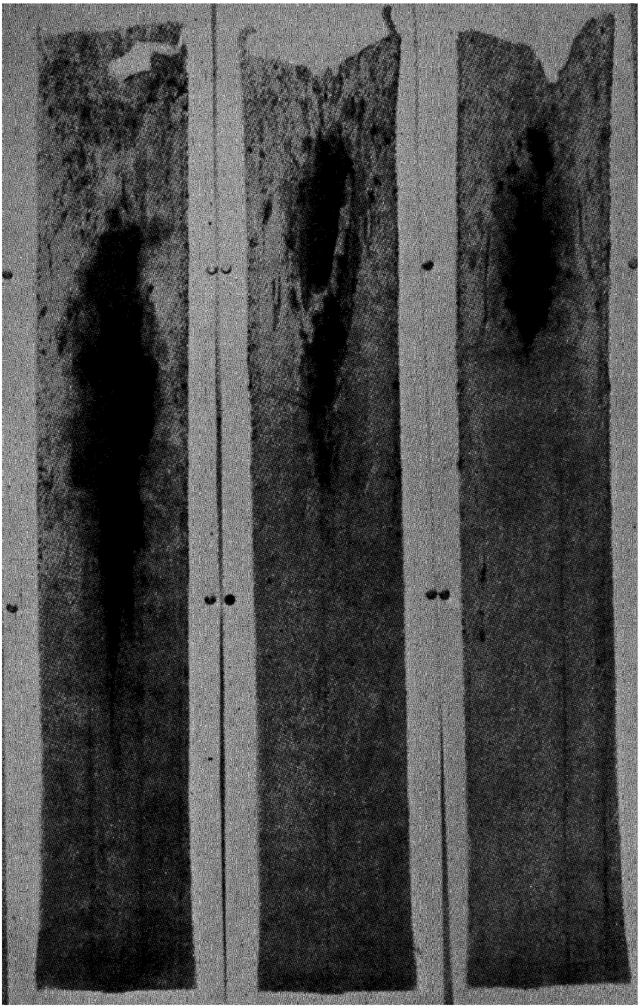
(3) Ingots in which the surface of the metal in the feeder-head had been treated with an anti-piping compound and which had subsequently been "after-teemed". The term "after-teemed" refers to the practice of refilling the feeder-head with molten metal after several other ingots have been cast.

It was found that when using anti-piping compounds alone, a period of one hour elapsed between the casting of the ingot and the final freezing of the surface of the liquid metal in the feeder-head. During this time the level of the metal in the head sank from 8 to 10 in. When both the anti-piping compound was employed and the ingot was "after-teemed" the liquid remained molten for a period of one and a half to two hours. The position and the extent of the piping in a 9-in. bloom rolled from experimental ingots are shown in Fig. 161 and demonstrate the advantages to be gained from the use of these compounds.

Segregation

Segregation is due to the differential solidification or freezing of the elements present in the steel. The carbon, manganese, sulphur and phosphorus in the liquid steel form compounds which have lower melting points than the pure iron, and which, therefore, remain in the liquid portion of the metal which is the last to solidify. This differential solidification causes the impurities to segregate to certain parts of the ingot which are the last to solidify, and this phenomenon is termed "positive segregation".

Positive segregation is usually confined to the upper portions of the ingot. Although segregation is not as uniform as the structural changes



(*a*) (*b*) (*c*)

Sulphur Prints of 9-in. Blooms rolled from 67-cwt. ingots, cast H456. (*a*) Cast with feeder-head only. (*b*) Covered with anti-piping compound after casting. (*c*) After casting, covered with anti-piping compound and then after-teemed.

FIG. 161. Effect of anti-piping compounds. (Gregory.)

(By courtesy of the " British Steelmaker ".)

in the ingot, it does follow a general pattern. One zone of positive segregation closely follows the position of the shrinkage cavity or pipe and is known as the V segregation. The other zone of positive segregation, which also occurs at the top of the ingot, consists of parallel bands of segregates slightly inclined to the axis of the ingot. These are shown in Fig. 162.

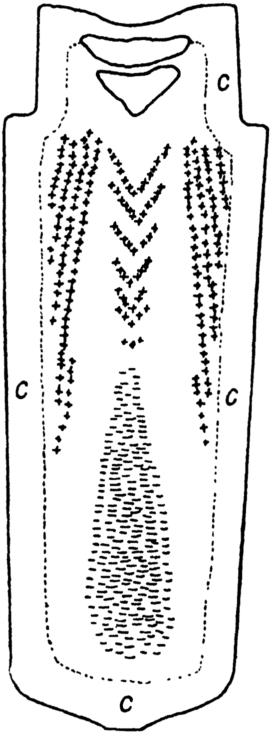


FIG. 162.
Zones of segregation.
(*By courtesy of "Iron and Steel".*)

Owing to the fact that some parts of the ingot contain positive segregation, other portions of the ingot will contain less than the average amount of impurities and will exhibit "negative segregation". Negative segregation is confined to the lower part of the ingot. As the metal solidifies from the bottom of the ingot, the crystals formed are more pure than the average and the less pure material segregates to the upper portion of the ingot. This zone of negative segregation is also shown in Fig. 162.

A considerable amount of work on segregation in ingots has been undertaken by the Ingots Committee,[20, 27] and their results have been published in a series of reports. Typical examples of segregation taken from their reports are shown in Figs. 163 and 164. It is apparent that the most important single factor determining the amount of segregation is the weight of the ingot. It has been shown that the tendency to segregate increases with the ingot weight and a useful graph showing the relationship between ingot weight and the degree of segregation is given in Basic Open-Hearth Steelmaking.[47]

The tendency for the individual elements to segregate can be expressed as a ratio of the percentage of the element in the first metal to solidify to the percentage of that element in the last metal to solidify. This ratio can be determined from the equilibrium diagram of the element and iron. Hayes and Chipman[48] have determined these ratios, or distribution constants K, for a number of the elements present in steel. The tendency to segregate is directly proportional to $1 - K$, and these values are shown in Table XLIV.

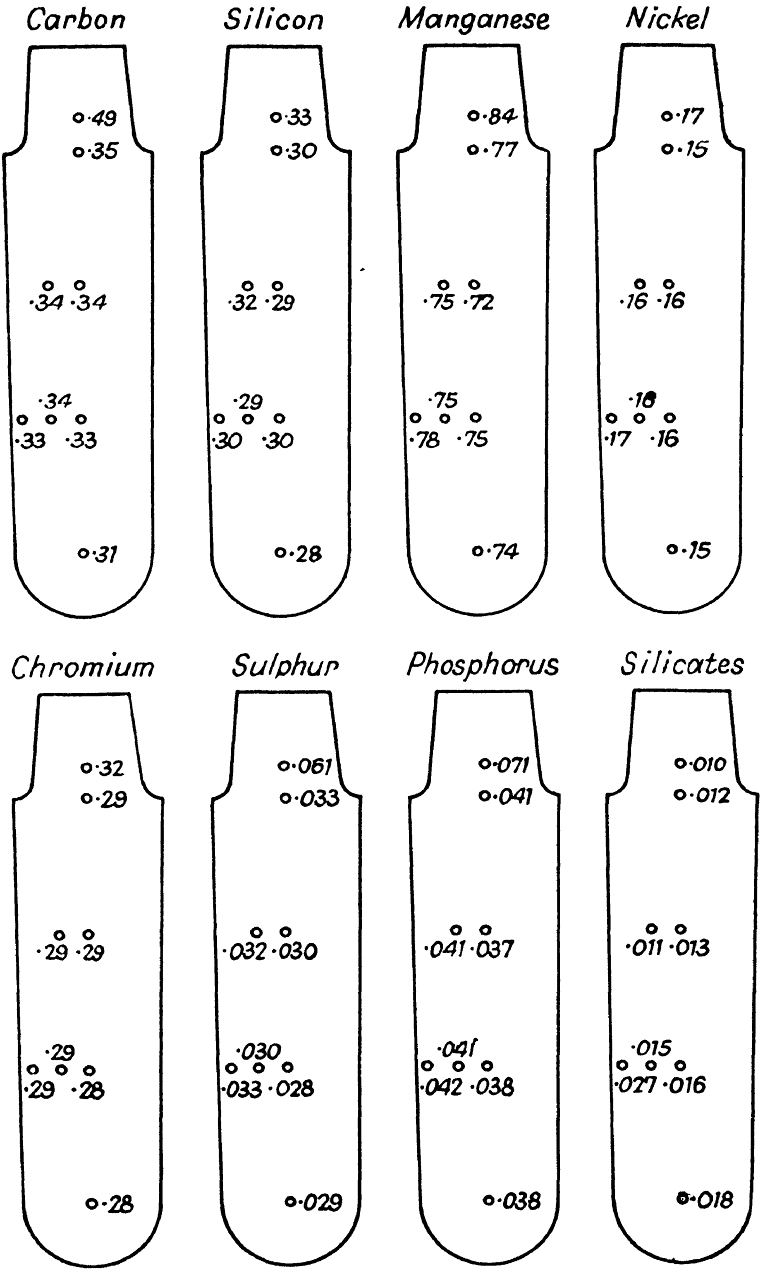


FIG. 163. Typical segregation (killed steel).
(Fourth Ingot Report—by courtesy of the Iron and Steel Institute.)

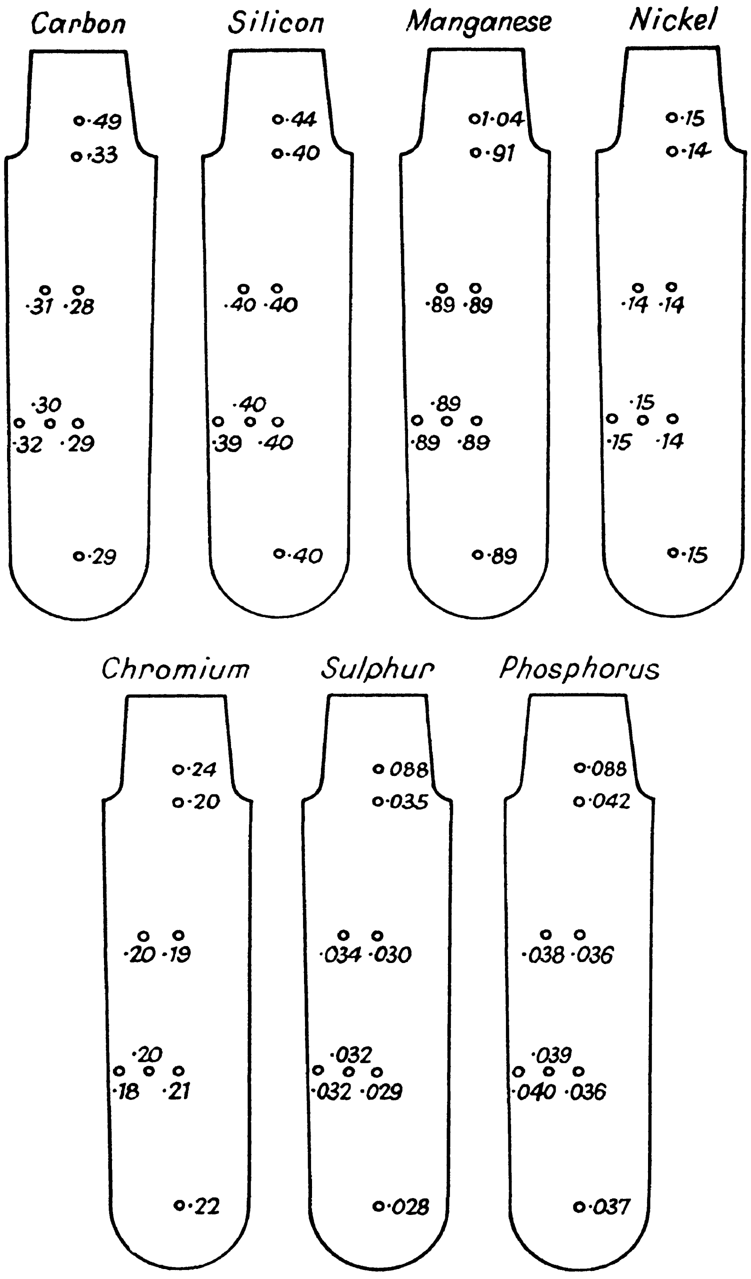


FIG. 164. Typical segregation—(killed steel).
(Fourth Ingot Report—by courtesy of the Iron and Steel Institute.)

TABLE XLIV

Element	K	1−K
Sulphur	0·05	0·95
Oxygen	0·10	0·90
Carbon	0·13	0·87
Phosphorus	0·13	0·87
Silicon	0·66	0·34
Manganese	0·84	0·16

Although segregation cannot be prevented, its *effects* can be reduced by:

(1) The rapid cooling of the ingot during solidification, which lowers the temperature gradients.

(2) The use of aluminium reduces the effect of segregation, probably due to the fact that the alumina particles act as nuclei during the crystal formation.

(3) The employment of feeder-heads or hot-tops, which concentrates the pipe in the upper portion of the ingot, and tends to confine the greatest degree of segregation to this area. In this manner, by discarding the feeder-head portion of the ingot, not only is the piping removed but the more highly segregated parts of the ingot are rejected.

Gregory[46] demonstrated that the delayed freezing due to the use of anti-piping compounds also affected the position of major segregation. Sulphur prints showed that the zone highest in impurities was nearest the top of the ingot in those ingots which had been treated with anti-piping compounds and which had been "after-teemed". This zone of impurities was lower and more elongated in ingots which had not been treated in this manner.

In addition to major segregation *microsegregation*, or the variation of composition over comparatively small areas, may occur. These changes in composition are not shown by chemical analysis, but are disclosed by sulphur prints. They are most pronounced in the columnar zone, where there is a tendency for the less pure liquid to be trapped between the columnar crystals in the course of their growth. To a large extent the microsegregation of carbon is neutralized by the diffusion of that element. Elements, like sulphur and phosphorus, which diffuse slowly will, however, exhibit microsegregation even after repeated reheating and rolling.

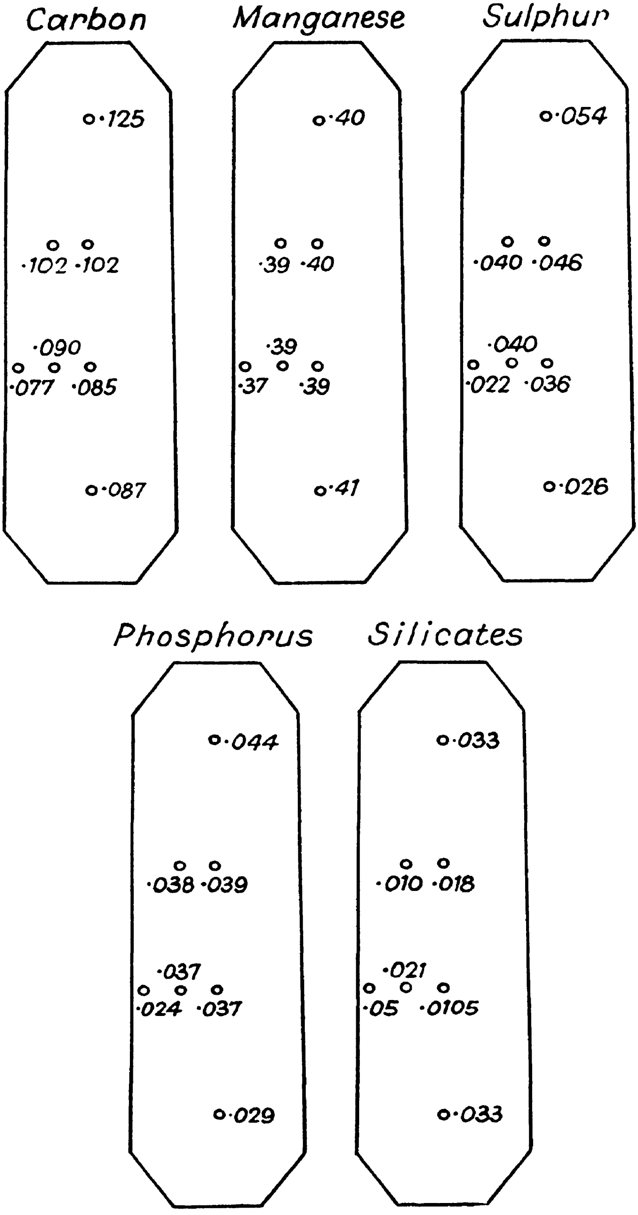


FIG. 165. Segregation in rimming steel.

(*Fourth Ingot Report—by courtesy of the Iron and Steel Institute.*)

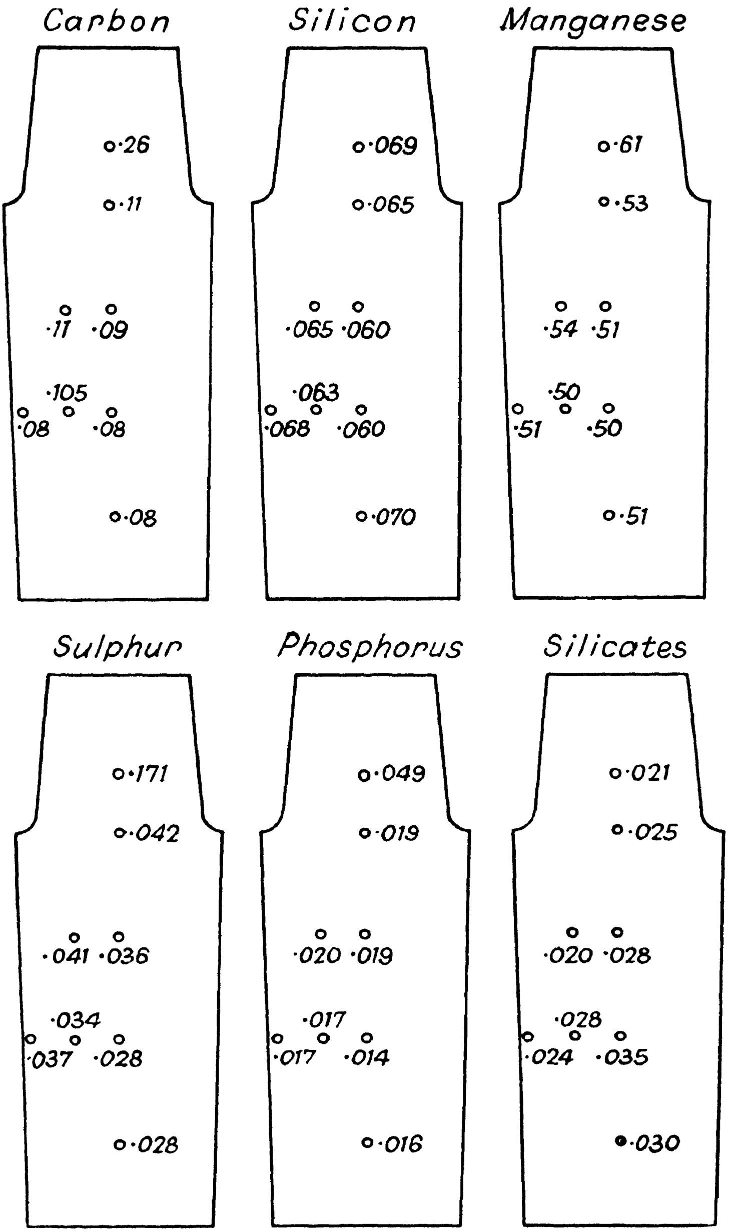


FIG. 166. Segregation in semi-killed steel.

(Fourth Ingot Report—by courtesy of the Iron and Steel Institute.)

The position and extent of segregation is also affected by the gas content of the steel and the liberation of this gas during solidification. In rimming steels, as explained on page 420, the zones of positive and negative segregation are displaced by evolution of gas and assume a distinctive pattern. Typical examples of segregation in rimming and semi-killed steels are shown in Figs. 165 and 166.

Surface Defects

Surface defects are frequently associated with ingot cracks and have been referred to under that heading. Surface defects arising from splash, bad mould surfaces, dirty and worn moulds, have been discussed on page 306. Major pulls, caused by some restriction to the free contraction of the ingot, have been mentioned in connection with ingot cracks and in conjunction with the use of feeder-heads, on page 294.

Another type of surface defect which may cause serious trouble in the production of chromium, vanadium or high-silicon steel, is termed "lappiness". Kilby[44] showed that these laps or folds on the surface were due to bottom-pouring too slowly or at low temperature. Owing to a low casting temperature or a very slow rate of pouring a solid crust is periodically formed on the surface of the ingot. As teeming proceeds, liquid metal breaks through this crust, entrapping some of the oxide film formed, giving rise to a lappy appearance.

Blowholes

The factors affecting the presence and position of blowholes have been discussed on page 281. Provided these blowholes are clean and deep-seated they are welded up during mechanical working and cause no further trouble.

Subcutaneous blowholes, occurring very near the skin of the ingot, may become oxidized during reheating, resulting in the formation of "roaks" and seams in the finished bars or blooms. It appears that this type of blowhole may also burst during mechanical work, resulting in similar defects.

Non-Metallic Inclusions

Non-metallic inclusions form probably the most difficult problem in steelmaking, and constitute a serious type of ingot defect. It should be realized that entirely clean steel has never been made and it is doubtful whether perfectly clean steel will ever be made. The cleanliness of steel is therefore a comparative term, indicating freedom or otherwise of the steel from non-metallic inclusions.

These inclusions fall into two classes:

1. Indigenous, which arise in the course of steelmaking.
2. Exogenous, which are caused by non-metallic matter entrapped by the steel, as a result of its contact with the furnace hearth or in the course of its transfer from the furnace to the ingot mould.

Indigenous are usually oxides or sulphides, although in certain cases carbides and nitrides play an important role. Practically all the sulphur present in the molten steel is precipitated during solidification as sulphides of the metals present. The best prevention for this type of inclusion is to ensure that the sulphur in the finished steel is as low as possible. Sulphide inclusions are usually a mixed sulphide of iron and manganese.

The oxide inclusions result from the various phases of steelmaking and their subsequent deoxidation. Their mode of formation, and preventative measures to counteract their formation, have been discussed in connection with the steelmaking processes on pages 26, 172, 189.

Normally nitrides are not precipitated in steel of the ordinary composition. If the nitrogen content of the steel is very high, nitrides may be precipitated, providing low temperatures and slow rates of cooling persist. The addition of such elements as zirconium, titanium or vanadium, however, causes the formation of stable nitrides. These nitrides have a slight solubility in the liquid steel, but practically no solid solubility. They are, therefore, precipitated during solidification as more or less distinct cubic crystals.

Carbides are not generally regarded as non-metallic inclusions, although the addition of some elements such as titanium to a sufficient quantity will result in the formation of stable carbides, which are practically insoluble in solid steel even at temperatures near to melting point.[47]

The mode of reducing the amount of inclusions of this type has, however, been discussed in the chapters dealing with the various steelmaking processes. To a large extent this procedure depends on:

1. Slag control.
2. Control of the state of oxidation.
3. Steelmaking procedure.
4. The correct choice of deoxidizers.

Exogenous inclusions are more easy to control and to prevent. They arise from long contact with the furnace lining or from contact with unsatisfactory furnace linings. As the metal runs from the ladle to the ingot mould, it must pass over a refractory lined chute, into a refractory lined ladle and in some cases the metal comes into contact with runner bricks in the case of bottom pouring, or the firebrick lining of a tundish when tundishing. These refractories may break and pieces may be entrapped in the finished steel.

In most cases, however, the trouble is due to poor-quality refractory materials, which fuse as a result of their contact with molten steel. This fused material is carried forward into the ingot, where it may be trapped as a non-metallic inclusion. The erosion of the ordinary fireclay ladle nozzle during the course of teeming is a good example of this erosion and the possibility of non-metallic inclusions being formed.

This type of inclusion can be prevented by good casting-pit practice, with special attention to cleanliness and the careful selection of casting-pit refractories. Additions of aluminium to the ingot moulds during the course of teeming should be avoided.

There is an old adage that hot steel is clean steel and there appears to be a logical basis for this statement. The hotter the steel the more fluid the metal and the greater the possibility of inclusions rising to the surface in accordance with Stokes's Law. This remark will apply particularly to the larger inclusions, and especially to the exogenous inclusions, which are usually large. The segregation of inclusions can also be assisted by holding the metal in the casting ladle for a period between tapping and commencing to teem the metal into the ingot moulds.

TABLE XLV

RATE OF RISE OF NON-METALLIC INCLUSIONS

Size of inclusion cm. × 10^{-4}	Max. rising velocity ins/min.	Least time required to rise through	
		30 inches of Open-Hearth Bath Mins.	12 feet of Metal in Ladle Mins.
5	0·061	493·00	2,360·00
10	0·245	122·00	587·00
50	6·120	4·93	23·60
100	24·450	1·22	5·87
500	612·000	0·05	0·24

This Table shows that if the steel is held in the ladle for a period of 15 minutes most of the inclusions will rise to the surface, especially if silico-manganese is used as the deoxidizer.

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CHAPTER 11

SIDE-BLOWN CONVERTER PRACTICE

A large number of side-blown converters are employed in foundries for the production of steel castings. In a few places this type of converter is also used for the production of steel ingots, but this practice is somewhat unusual.

Normally these converters are acid-lined and refine haematite pig iron, which is usually melted in cupolas. At works producing steel ingots by this process, the cupolas are of special design and the charge

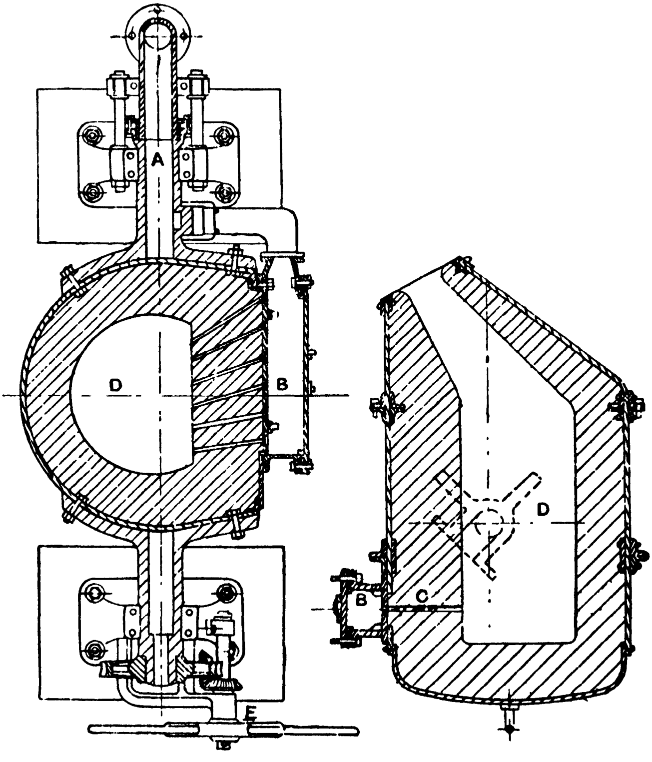

FIG. 167. Typical side-blown converter showing wind-box.
(By courtesy of Manc. Assoc. Engineers.)

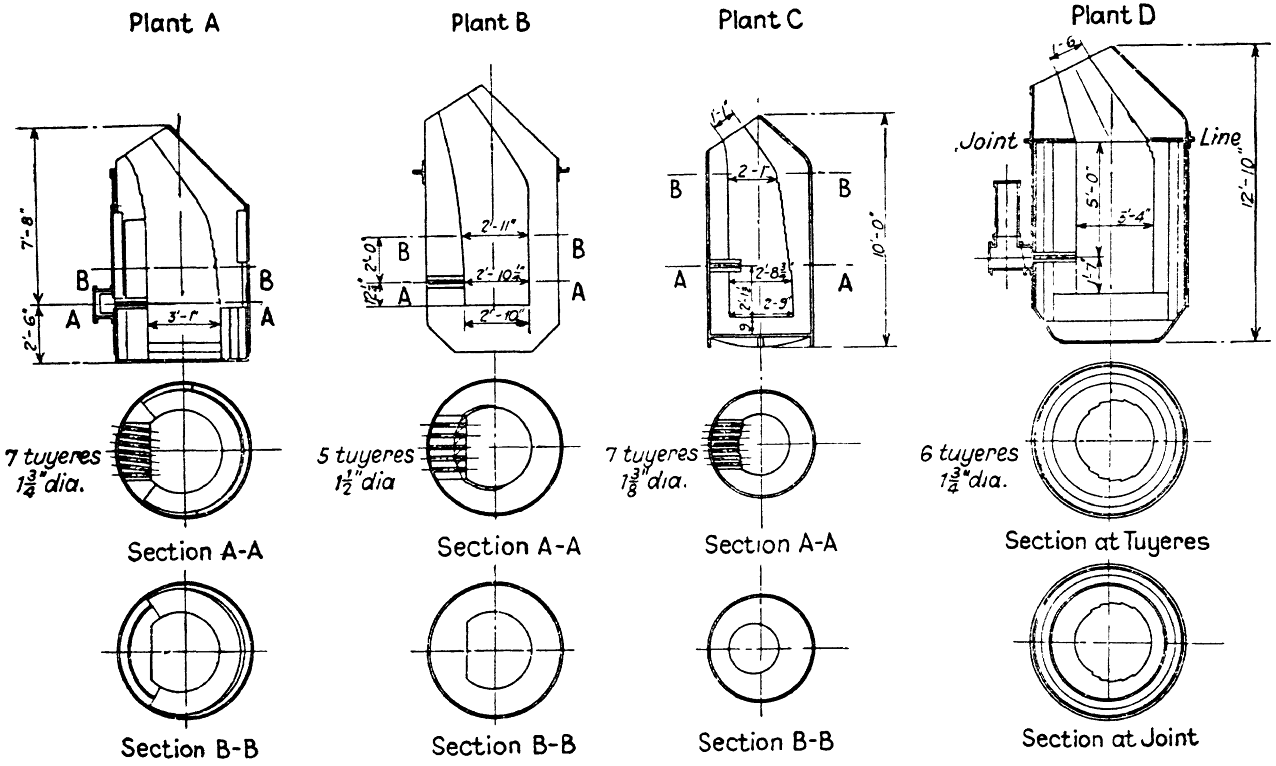

FIG. 168. Four types of side-blown converters.
(*By courtesy of the Iron and Steel Institute.*)

employed frequently contains high percentages of steel scrap, which is partially recarburized before blowing in the converter.

General Design of Converter

Side-blown converters vary in size, but are generally about 3 tons capacity. Frequently they are provided with two rows of tuyeres, which can be operated independently. It is claimed that this double row of tuyeres assists the production of a higher percentage of carbon dioxide in the gases issuing from the converter, thereby raising the temperature of the metal.

The general design of a side-blown converter is shown in Fig. 167, whilst Fig. 168 shows the design of four converters recently discussed by the Steel Castings Research Committee.[1] The details of these converters are given in Table XLVI, whilst Table XLVII gives the normal operating conditions adopted.

In the true side-blown converter the blast is admitted through tuyeres in one side of the vessel. These tuyeres are either above or at the surface of the metallic bath. Usually they are just above the level of the bath and are arranged in the shape of a fan, so that the blast is evenly distributed over the whole of the converter.

Owing to the acid nature of the normal charge these converters are lined with acid materials. Three types of lining can be employed:

(1) Silica rock or ganister, which can be rammed around a suitable former. A lining of this type will frequently give a life of 100 to 120 heats, although in some cases as many as 230 heats have been obtained. With linings of this type the tuyeres may either take the form of specially shaped tuyere-bricks or the tuyeres may be rammed by means of formers. In most cases these rammed linings are backed with a 4½-in. layer of brickwork.

(2) A similar type of lining may be prepared by employing a mixture of sand and fireclay, in which case the fireclay acts as a bond to the more refractory sand.

(3) The more modern method is to line the converter with high-grade silica bricks, in a manner similar to that described in Chapter 2 for the preparation of the Acid Bessemer Converter. When a brick lining is employed, the tuyeres are invariably formed of specially shaped bricks.

The lining usually requires some patching after each heat, although in many plants the converter is operated for 7 to 12 heats, after which

TABLE XLVI

DETAILS OF SIDE-BLOWN CONVERTER PLANTS. (*Steel Castings Research Comm.*[1]).

Plant	A	B	C	D
Normal Capacity, cwt.	60	47	50	55
Shape of Vessel	D-shaped.	D-shaped.	Circular.	Approx. Circular.
Number of tuyeres	7	5	7	6
Diameter of tuyeres, in.	1¾	1½	1⅜	1¾
Capacity/Tuyere area, cwt./sq. in.	0·28	0·19	0·21	0·26
Tuyere area, sq. in.	16·8	8·8	10·4	14·4
Arrangement of tuyeres	Six inclined, One straight.	At right angles to tuyere box.	At right angles to tuyere box.	At right angles to tuyere box.
Depth of Well	1 ft. 10 in.	1 ft. 0 in.	2 ft. 1½ in.	1 ft. 7 in.
Area at tuyere level, sq. in.	1,250	1,134	755	1,250
Area/Capacity, ratio, sq. in./cwt.	21	24	15	23
Area/Depth of Well ratio	57: 1	94: 1	30: 1	66: 1
Area/Tuyere area, ratio	75: 1	130: 1	72: 1	87: 1
Type of Blower	Rotary.	Compressor.	Rotary.	Rotary.
Max. output of Blower, cu. ft./min.	4,380	2,500	2,400	5,000

TABLE XLVII

Average Operating Conditions of Side-Blown Converters. (*Steel Castings Res. Comm.*[1])

Plant.	A	B	C	D
Composition of the metal:				
Carbon %	2·5	3·3	3·0	3·1
Silicon %	1·0	1·0	1·2	0·1
Manganese %	0·3	0·4	0·4	0·4
Sulphur %	0·06	0·04	0·03	0·04
Phosphorus %	0·04	0·04	0·06	0·06
Temperature of metal charged. ° C.	1,320	1,300	1,320	1,450
Blast volume, cu. ft./min.	2,400–2,800	2,200	1,600–1,800	3,500
Blast pressure, lb./sq. in.	3	5	4	4
Blowing time in minutes	25	19–20	30	9–12
Blowing angle. deg.	20–10	14–10	20–12	13–8
Additions during the blow:				
Ferro-silicon 45% Si	—	—	60 lb.	—
75% Si	80 lb.	75 lb.	—	75 lb.
Deoxidation and alloying	Metal poured into a ladle containing 6 lb. Al per ton, then into a second ladle containing 1 to 1½ lb. per ton Ca, Si, Mn, and alloys.	Slag raked off; 7 lb. per ton Al rabbled in. Alloys added in the vessel.	Ferro-silicon, silicon-mang. and ferro-manganese added in the vessel. Then poured into a ladle containing 3 lb. per ton Al.	Slag raked off 5 lb. per ton Al rabbled in. Alloys added in the vessel.
Interval between heats	30 min.	15 min.	30 min.	5 min.

such repairs as are necessary to the lining are made. The chief erosion occurs in the vicinity of the tuyeres and in order to facilitate repairs to this portion several types of drop-bottom converters have been designed.

In some works two converters are provided and these converters are operated on alternate days. When this policy is adopted a converter will produce about 10 or 12 heats and the following day the necessary repairs will be made. At other plants the two converters are operated continuously for about five days, after which extensive repairs are carried out. Frequently the top section or nose of the converter is removed to facilitate these repairs being performed.

The position of the wind or blast box is shown in Fig. 167. The method of supplying this blast and the mechanism for rotating the converter are similar to that employed in connection with the bottom-blown converters. Since the blast is admitted above the metallic bath a lower blast pressure is employed. This blast pressure varies from 3 to 5 lb. per sq. in.

Reaction in the Converter

In normal converter practice the reactions taking place can be divided into three stages:

(1) When the blast impinges on the surface of the metal a layer of iron oxide is formed. Some silicon and manganese may be oxidized at this stage and their oxides combine with the iron oxide to form a ferrous-manganous silicate slag. The slags formed initially, however, are chiefly FeO.

(2) When the surface of the metal is completely covered with slag, the oxidation of the silicon and manganese is accelerated. The oxidation of these elements is essentially a slag/metal reaction and is similar to those reactions which occur in the acid open-hearth furnace.

(3) When the silicon has been removed, or very largely removed, the elimination of carbon becomes rapid. The temperature increases considerably at this stage, frequently reaching 1,400 to 1,450 deg. C. At still higher temperatures the reduction of SiO_2 and MnO by carbon may take place.

The oxidation of carbon by the FeO or other oxides in the slag results in the formation of carbon monoxide at the bath level. This carbon monoxide is converted to carbon dioxide by the excess of air within the converter. This feature is the essential difference between the

TABLE XLVIII

ANALYSIS OF GASES ESCAPING FROM THE CONVERTER

Samples taken 9 in from nose end.

	First Heat.			Second Heat.			Third Heat.		
Weight of Charge. Cwt.	75			65			65		
Silicon content. %	0·18			0·32			0·6		
Duration of blow. Min.	12			12			17		
Gas analyses after min.	1	5	9	2	7	11	1	7	11
Carbon dioxide. %	4·0	9·8	12·8	12·0	12·2	4·2	4·2	10·0	9·8
Oxygen. %	10·4	2·6	2·8	2·4	0·9	10·2	5·6	4·4	5·0
Carbon monoxide. %	0·2	0·2	0·2	0·2	Nil	Nil	0·1	0·1	Nil
Nitrogen. %	85·4	87·4	84·2	85·4	86·9	85·6	90·1	85·5	85·2
Blast volume. Cu. ft. per min.	4,200	200	4,000	4,300	4,500	4,500	4,200	4,200	4,200

TABLE XLIX

ANALYSES OF CONVERTER GASES 4 FT. ABOVE THE TUYERES.

	First Heat.			Second Heat.			Third Heat.			
Weight of Charge. Cwt.	56			56			60			
Carbon content. %	2·92			3·14			2·98			
Silicon content. %	0·46			0·31			0·66			
Duration of blow. Min.	13			11			11			
Gas analyses after min.	3	6	10	3	6	10	1	4	8	10½
Carbon dioxide. %	4·0	6·2	14·2	8·0	6·6	10·8	6·4	12·0	12·8	12·4
Oxygen. %	10·6	3·0	1·6	0·8	4·2	0·4	0·3	0·1	Nil	0·6
Carbon monoxide. %	1·0	0·4	2·6	0·8	0·4	1·4	0·9	0·7	0·2	0·1
Nitrogen. %	84·4	90·4	81·6	90·4	88·8	87·4	92·4	87·2	87·0	86·9
Blast volume. Cu. ft. per min. ..	3,500	3,000	3,000	3,580	3,050	3,050	3,200	2,900	2,900	2,900

bottom- and side-blown converter practice. In the bottom-blown converter all the blast passes through the metal and all the oxygen is absorbed.

This conversion of the carbon to the higher oxide results in the production of considerably more heat per unit of carbon in the original charge than occurs in the bottom-blown practice. This aspect was noted by Fassotte[2] and typical analyses of gases escaping from the converter quoted in his paper are given in Tables XLVIII and XLIX.

Fassotte gives an interesting log of a normal blow, which is reproduced in Table L. This table shows the changes taking place in the metal and slag during the progress of the blow.

TABLE L

METAL AND SLAG ANALYSES DURING THE BLOW
(Tropenas Plant)

Converter charge: carbon 2·54 per cent., silicon 1·46 per cent., sulphur 0·079 per cent., phosphorus 0·047 per cent., manganese 0·27; 75 cwt. Temperature of iron: 1,280° C. Total blowing time: 48 min.

	Time from commencement of blow				
	11 min.	18 min.	28 min.	37 min.	Finish of blow
Analysis of metal:					
Carbon per cent.	2·66	2·44	2·18	1·38	0·05
Silicon per cent.	1·26	1·08	0·50	0·34	0·06
Manganese per cent. ..	0·19	0·15	0·08	0·06	0·03
Sulphur per cent.	0·078	0·070	0·067	0·059	0·054
Phosphorus per cent.	0·047	0·050	0·052	0·049	0·045
Analysis of slag:					
Silica per cent.	38·1	40·95	46·15	55·35	66·65
Ferrous oxide per cent.	54·5	52·28	47·61	37·67	27·09
Alumina per cent.	3·9	3·20	2·80	3·70	3·60
Manganous oxide per cent. ..	3·29	3·36	3·36	3·07	3·39
Lime per cent.	0·50	0·40	0·30	0·30	0·30
Magnesia per cent. ..	Trace	Trace	Trace	Trace	Trace
Temperature (immersion thermocouple). ° C. .. .	1,310	1,370	1,560	1,685	1,690

In normal side-blown converter practice it is necessary to have sufficient silicon in the molten metal supplied to the vessel to raise the temperature of the metal to 1,400 deg. C. before satisfactory elimination of carbon will occur. When the metal has attained this temperature carbon elimination proceeds rapidly.

TABLE LI

Composition of Metal and Slag at Progressive Stages of the Blow at Plants A and D

Plant A							Plant D						
Time Min.	Metal			Slag			Time Min.	Metal			Slag		
	C%	Si%	Mn%	SiO₂%	FeO%	MnO%		C%	Si%	Mn%	SiO₂%	FeO%	MnO%
0	2·6	1·2	0·15	—	—	—	0	3·1	0·4	0·4	—	—	—
8	2·6	1·1	0·12	39·9	53·8	2·0	2	2·9	0·2	0·2	55·4	29·9	10·1
17	2·6	0·7	0·1	39·4	54·5	2·0	4	2·6	0·05	0·05	54·2	29·4	12·2
34	2·2	0·2	0·03	—	—	—	6	2·0	0·05	0·05	54·0	39·3	12·0
44	0·9	0·04	0·02	50·9	42·9	2·5	8	1·0	0·05	0·05	56·0	28·8	10·9
49	0·04	Trace	0·02	54·4	30·5	2·5	14	0·08	0·05	0·05	60·0	25·5	10·7

Side-blown Converter Practice

In some cases the molten metal is transferred direct from blast furnaces or from metal mixers, but in most cases the metal is melted in cupolas. The cupola charge consists chiefly of haematite pig iron although at some plants a considerable amount of scrap metal is employed. Where high percentages of scrap are employed the silicon content of the molten metal is adjusted by means of ferro-silicon additions. These additions of ferro-silicon may be as the metal is transferred to the converter, or in some cases they are made to the converter according to the progress of the blow. The amount of ferro-silicon added depends on the silicon content of the molten metal and the temperature of this metal. If the temperature is low, the silicon content is increased in order that the oxidation of the element will generate sufficient heat to ensure satisfactory blowing procedure.

When the metal has been refined, which is indicated by the drop of the carbon flame as in the case of the acid Bessemer process, the blown metal is poured into a ladle. In some cases the slag is removed before the metal is poured from the converter and any alloy additions are made to the vessel. At other plants these alloy additions are made in the ladle. Typical examples are given in Table LI, which is taken from the work of the Steel Castings Committee.[1]

Owing to the small capacity of these converters a much more rigid control of the blow is necessary to avoid over-oxidation of the metal. The successful application of the spectroscope to the control of the side-blown converter has been discribed by Jazwinski,[4] whose work is reviewed in Chapter 3. He claims that the adoption of spectrographic control has resulted in a considerable reduction in the number of "off-grade" heats produced.

Modification of Side-Blown Practice

The work of Fassotte[2] has already been mentioned. He carried out some large-scale experiments on the design and use of side-blown converters. These experiments were really initiated because of the war-time shortage of haematite pig iron normally employed in this type of converter. Instead of melting up the usual charges of haematite and scrap in cupolas and subsequently blowing this metal in the side-blown converter, he melted up a charge of low-silicon iron in an electric furnace and then converted it into steel.

The results of this work demonstrated that:

(1) In the side-blown converter the bulk of the carbon is transformed

into carbon dioxide, provided that there is an adequate blast volume. Typical analyses are given in Tables XLVIII and XLIX.

(2) Irons containing from 2·7 per cent. carbon upwards, with very low silicon contents, can produce a temperature increment of over 200 deg. C.

(3) Silicon in the iron may be dispensed with, providing the iron is sufficiently hot at the commencement of the blow.

(4) It was found that when blowing low-silicon iron severe erosion of the lining occurred. This trouble could, however, be counteracted by a sand addition to the converter before the introduction of the molten iron. This sand addition amounted to 70 lb. of sand to a 70-cwt. charge.

Typical changes in the analysis of the slag during a low-silicon blow are shown in Table LII. These experiments led to the evolution of a modified side-blown practice, which consisted of melting up high scrap charges in a cupola, superheating this metal in a rotary furnace and subsequently converting it in a side-blown converter.

TABLE LII

SLAG ANALYSES DURING THE BLOW (LOW-SILICON IRON)

Converter charge: carbon 2·7 per cent., silicon 0·2 per cent., manganese 0·35 per cent.; 70 cwt. sand (70 lb.) put into the converter prior to the introduction of the iron. Total blowing time: 14 min.

	Time from commencement of blow				
	2 min.	4 min.	6 min.	8 min.	14 min.
Silica per cent.	55·4	54·2	54·0	56·0	60·0
Alumina per cent.	3·6	3·5	3·7	3·6	3·1
Ferrous oxide per cent.	23·2	25·7	25·1	25·2	22·5
Ferric oxide per cent.	7·5	4·1	4·7	4·0	3·3
Manganous oxide per cent. ..	10·1	12·2	12·0	10·9	10·7

As pointed out by Fassotte[2] the advantage of this practice is the use of cheap scrap in the cupola charges, but these economies are off-set by:

(1) A considerable increase in the capital cost of the plant necessary, which involves mixers and auxiliary equipment.

(2) High coke ratios are required to melt the cupola charges.

(3) There is greater wear on the cupola refractories.

(4) Extra labour and maintenance costs are necessary.

(5) There is the additional cost of firing and maintaining the mixers or rotary furnaces.

In spite of these considerations this practice has been successfully applied on a commercial scale to a limited extent. The cupola charges consist of 100 per cent. steel scrap but the cupola used is of the balanced-blast type. The coke ratio is about 6 to 1 coke and a ferro-silicon addition of about 2 lb. of ferro-silicon per 200 lb. of metal charge. The temperature of the cupola metal varies from 1,400 to 1,450 deg. C. Unless the sulphur content of this metal is low, it is advisable to treat the metal in a basic-lined ladle with soda ash. This treatment will, of course, reduce the temperature of the metal.

This metal is transferred to a rotary furnace where it is superheated to a temperature of about 1,500 deg. C. The superheated metal is subsequently converted in a side-blown converter. Owing to the temperature of the molten metal supplied to the converter carbon elimination begins immediately and the blow progresses quickly.

This particular practice is mentioned by the Steel Castings Committee[1] in the report already referred to, details are given in Tables XLVII and LI under example D. In this plant 100 per cent. scrap charges are melted in a cupola and this metal is superheated in a rotary furnace before being charged into the converter. In order to prolong the life of the converter lining sand is added to the vessel before the molten metal is charged. This sand provides an acid oxide for combination with the large quantities of iron oxide formed during the early stages of the blow. The relatively high temperature of the metal charged into the converter results in the rapid oxidation of carbon at the beginning of the blow. The temperature of the metal actually charged into the converter averages 1,400 to 1,450 deg. C.

They include an interesting note on the gas content of the steel produced by this practice. Samples analysed for oxygen and nitrogen show:

(1) That the desulphurization of the metal with soda ash tends to decrease the nitrogen content about 50 per cent., from about 0·015 to 0·008 per cent. N.

(2) During conversion there is a further decrease in both the nitrogen and oxygen contents. The gas content falls to approximately 0·005 per cent. N and 0·06 per cent. O.

(3) During the deoxidation and alloying treatment there is a slight increase in the nitrogen content to about 0·075 per cent., which is slightly higher than the nitrogen content of open-hearth steel, but which is lower than that of electric or bottom-blown converter steel. In this period the oxygen content drops to about 0·004 per cent., which compares favourably with steel made by the other processes.

Metallic Yield

The blowing losses can be divided into two classes:

(1) Chemical.

(2) Mechanical.

The chemical losses depend on the analysis of the iron. These losses are due to the carbon, silicon, manganese and iron oxidized during the blow. To some extent the amount of iron oxidized depends on the composition and weight of the slag produced. These losses can only be controlled by selection of the original pig iron and correct slag formation.

The Steel Castings Committee[1] show that the mechanical losses are chiefly due to the blowing time and the silicon content of the pig iron blown. Any increase in the blowing time results in a longer period during which ejections from the vessel occur, with a corresponding increase in metallic loss from this cause. A prolonged blowing period also results in increased erosion of the converter lining, which will cause more silica to pass into the slag requiring more FeO to approach equilibrium conditions.

There also appears to be a definite relationship between mechanical loss and the silicon of the original pig iron. This loss is also affected by silicon additions made during the blow. The silicon content causes an increase in the mechanical loss. It is possible that this effect is connected with the increased temperature caused.

It is a well-known fact that the ejections from the converter increase with the increasing temperature of the metal. This temperature will be a function of the temperature of the initial metal and the silicon removed during the blow.

It is, therefore, obvious that the mechanical loss will depend on the blowing time, the silicon content of the blow and the temperature generated during the blow. The total blowing loss will vary from 5 to 12 per cent. depending on the particular process. This total loss will be composed of the chemical loss, which will vary from 3·5 to 5 per cent. according to the composition of the original pig iron, and the mechanical loss, which may vary from 2 to 8 per cent. When operating on high scrap charges, which are low in silicon, these losses may be as low as 3·5 per cent. for the chemical loss and 2 to 4 per cent. for the mechanical loss, giving a total loss of 5 to 7·5 per cent.

Thermal Balance Sheets

In the normal side-blown practice from 38 to 42 per cent. of the total heat introduced into the converter is in the form of the sensible heat of the metal charged. Where high scrap charges are melted in the cupola and superheated in a rotary furnace before being charged into the converter, the sensible heat in the molten metal charged represents about 50 per cent. of the total heat input.

The heat produced by the oxidation of carbon, silicon, manganese and iron accounts for 54 to 62 per cent. of the heat input. If working low-silicon mixtures this figure may decrease to about 49 or 50 per cent. Since the volume of slag formed is small, the heat increment from this source seldom exceeds 1 per cent.

The sensible heat in the finished steel represents from 45 to 55 per cent. of the heat absorbed, whilst the heat lost in the escaping gases and fume and the heat lost by radiation amounts to about 40 to 50 per cent. of the total heat expended.

The following figures taken from the Steel Castings Committee's report give a typical heat balance sheet of a normal blow.

Heat Input	kg. cal.	per cent.
Sensible heat in molten metal	805,140	42.9
Oxidation of C, Si, Mn and Fe	1,066,740	56·8
Heat of slag formation	6,440	0·3
Total heat input	1,878,320	100·0
Heat Absorbed		
Sensible heat in the blown metal	936,320	49·8
Sensible heat in the slag	60,500	3·2
Heat in the ejections	51,300	2·8
Heat in the waste gases, radiation losses, etc.	830,200	44·2
Total heat expanded	1,878, 320	100·0

The "Stock" Oil-Fired Converter

As a small economical unit the "Stock" oil-fired converter described by Lange[5] is of considerable interest. The general design and arrangement of this type of converter is shown in Figs. 169 and 170. It is a side-blown converter, whose main feature is that it is also designed to melt the charge by means of oil burners. During charging and melting the converter is turned down into a horizontal position and oil pipes or burners are introduced through the blowing tuyeres. The nose of the converter is turned towards an air-heater, which consists of a series of cast-iron pipes contained in a firebrick structure.

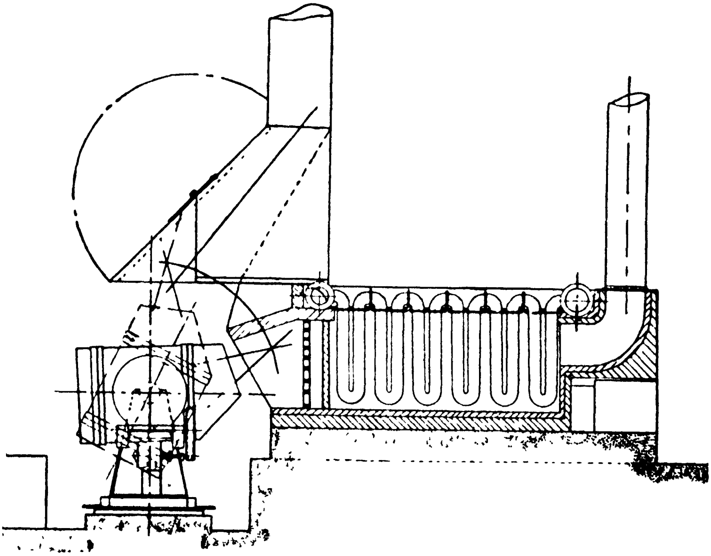

FIG. 169. Stock converter.
(By courtesy of Manc. Assoc. Engineers)

The air required for burning the oil is supplied, pre-heated through this heater, at a pressure of $\frac{3}{4}$ lb. per sq. in. The gases issuing from the converter are drawn through this heater by chimney draught. The oil is delivered to the converter at a pressure of 30 to 33 lb. per sq. in.

When the solid charge is melted, the oil burners and pipes are withdrawn, the vessel is turned up and the blowing operation conducted in the usual manner. The blast pressure normally employed is 2 to $4\frac{1}{2}$ lb. per sq. in.

The chief advantage of this type of converter appears to be the fact that it dispenses with the need for cupolas for melting the solid charge and provides a finished metal which is sufficiently fluid for intricate foundry work.

Oxygen Enrichment of the Blast

One of the most important steps in recent years has been the application of oxygen enrichment of the blast. Work on this subject has been proceeding for several years on the Continent. Haag[6] shows that with a 35 per cent. enrichment of the blast the blowing time was halved without any increase in the blowing loss. The properties of the steel produced were normal. In a more recent work Eilender and Roeser[7] claimed that with total oxygen contents up to 35 per cent., the recovery of manganese and the removal of sulphur were improved, a greater control of temperature was possible, whilst a lower nitrogen content in the finished steel resulted.

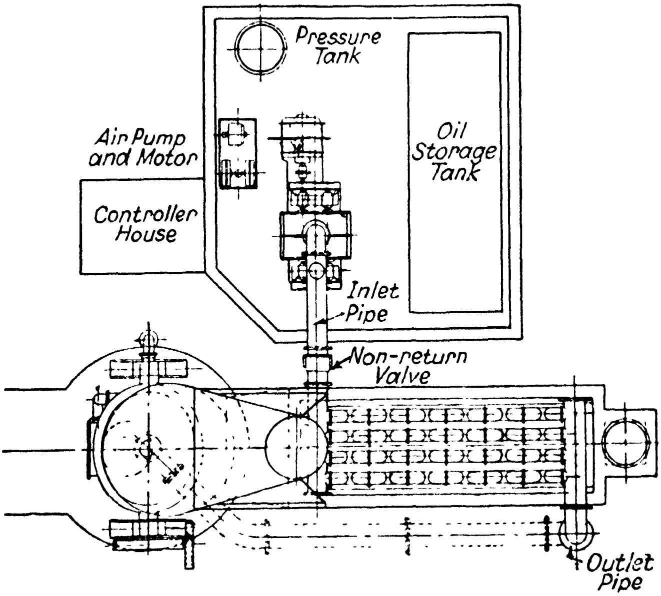

FIG. 170. Stock converter layout.
(By courtesy of Manc. Assoc. Engineers.)

Recently a classical work has been published by Harrison, Newell and Hartley[3] in which they describe the application of oxygen enrichment on a commercial scale over a long period. The converters used in these investigations were originally operated as "Stock" converters, although they are now operated as normal converters, the original oval shape and dimensions remaining unchanged. The converter shell has a siliceous monolithic lining about 13 in. thick. Owing to the shortness of the nose and body of the converter the blowing loss is rather higher than the modern-designed side-blown converters. This blowing loss averages about 10 per cent. The converter is provided with six 2-in. diameter silica-brick tuyeres, spaced 10 in. from the bottom of the vessel. The vessel has a capacity of 40 cwt. when new increasing to 50 cwt. when the lining becomes worn.

It has been found that very little wear occurs below the tuyere level, and in the vessels where the bottoms are made of silica bricks, these bottoms remain in position about six times as long as the lining.

The blast is supplied by a positive-displacement Rootes blower at a pressure of 3 to 3½ lb. per sq. in. The normal blast rate is 2,700 to 3,000 cu. ft. per min., but this rate can be controlled by regulating the speed of the blower to give from 1,600 to 4,000 cu. ft. per min.

In the normal melting and converting practice, the cupola charges consist of 20 per cent. haematite iron and 80 per cent. steel scrap, with an addition of ferro-silicon sufficient to raise the silicon content of the metal to 1·0 to 1·2 per cent. The temperature of this metal averages between 1,320 and 1,340 deg. C. After blowing in the converter this temperature is raised to 1,660 to 1,680 deg. C. If a further addition of ferro-silicon is made to the converter, equivalent to about 0·4 per cent. Si, this temperature can be increased to 1,740 to 1,750 deg. C.

TABLE LIII

OXYGEN-ENRICHED BLAST. (Harrison, Newell & Hartley[3])

Total rate of flow. cu. ft./min. ..	2,700	2,700	2,700	2,000	2,000
Total oxygen per cent.	20·5	30	40	30	35
Rate of air blast cu. ft./min. ..	2,700	2,400	2,000	1,750	1,650
Duration of blow. Mins. ..	16·10	8·10	5·31	8·38	8·10
Total O added cu. ft.	Nil	2,600	4,100	2,400	2,900
Composition of Metal					
Carbon per cent.	3·14	2·82	2·82	3·10	3·85
Silicon per cent.	0·66	0·66	0·60	0·65	0·61
Manganese per cent.	0·55	0·40	0·37	0·40	0·38
Composition of Blown Metal					
Carbon per cent.	0·09	0·09	0·08	0·08	0·08
Silicon per cent.	0·06	0·10	0·08	0·08	0·12
Manganese per cent.	0·05	0·07	0·06	0·08	0·09
Wt. of blown metal cwt.. ..	45½	44	46	41	43
Blowing loss per cent.	8·5	11·1	22·0	10·2	10·3
Cupola metal temperature °C. ..	1,325	1,371	1,370	1,340	1,361
Temperature of blown metal °C.	1,640	1,780	1,844	—	1,805

When oxygen-enriched blast was employed the larger temperature increment obtained rendered the addition of ferro-silicon to the cupola charges unnecessary. In these trials the silicon content of the molten metal averaged between 0·5 and 0·7 per cent.

Several trials with varying degrees of oxygen enrichment were made. In some cases the rate of the blast was maintained at 2,700 cu. ft. per min., whereas in other cases the rate of flow was reduced to 2,000 cu. ft. per min. This reduction in the rate of flow appeared to reduce the blowing losses. The average of several blows, taken from their work, is shown in Table LIII.

The general conclusion advanced by the authors of this paper is that oxygen enrichment of the blast is likely to be widely applied to side-blown converters in the future owing to the following technical advantages:

(1) A higher steel temperature is possible, with its attendant advantages in the foundry, such as increased fluidity and reduced skulling loss.

(2) A shorter blowing cycle is possible, thereby permitting a higher steel output from a given plant.

(3) A lower temperature of the initial iron can be accepted, thus allowing some economy in the fuel employed in the cupola.

(4) A higher percentage of scrap may be employed in the cupola charges, thereby reducing the pig iron consumption and raw material costs.

(5) A reduction in the need for ferro-silicon additions to the cupola during melting or to the converter during the blowing operation.

(6) Some reduction in the blowing loss is noted.

(7) An easier control of the end-point is claimed and it is shown that the final product is more consistent in composition.

Against these advantages the cost of the oxygen must be taken into consideration. This cost will depend on the size of the installation, which materially affects the cost of producing oxygen.

During these trials no decrease in the average life of the converter linings was noted. The final steel showed a lower nitrogen content than usually found in the normal side-blown converter steel. It is suggested that oxygen enrichment of the blast of side-blown converters will not only find increasing employment for foundry work, but also for bulk ingot-steel production.

Dimensions of Side-Blown Converters

In discussing the determination of the dimensions of small converters, Goldsbrough and Throssell[8] point out that, independently of capacity, the depth of the bath is standardized at 21·6 in. or 1·8 ft. From this depth the external and internal diameters can be calculated.

On a 2-ton converter, since one ton of metal (molten) occupies

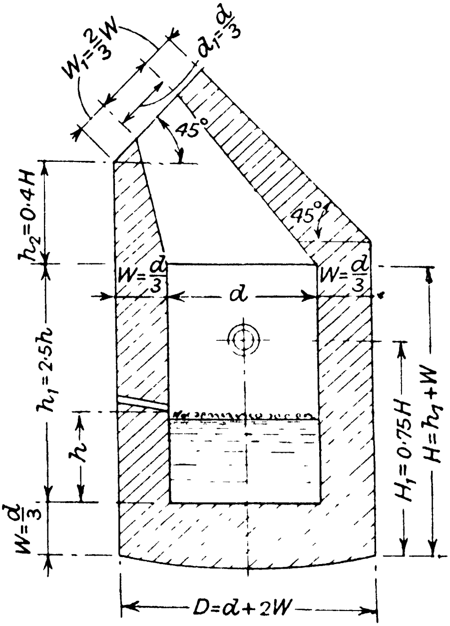

FIG. 171. Goldsbrough and Throssel converter design diagram.
(By courtesy of the " British Steelmaker ".)

approximately 4·944 cu. ft., a total capacity of 9·888 cu. ft. is required. Therefore the diameter, d, will be:

$$d^2 \times 0{\cdot}7854 \times 1{\cdot}8 = 9{\cdot}888$$

or $$d^2 = \frac{9{\cdot}888}{0{\cdot}7854 \times .1{\cdot}8}$$

$$= 2{\cdot}6446 \text{ ft.}$$

In other words an internal diameter of about 2 ft. 9 in. or 3 ft. would be selected.

The other dimensions can be determined from the formula given by Goldsbrough and Throssell, shown in Fig. 171. Normally the plates forming the converter shell vary from $\frac{7}{16}$ to $\frac{1}{2}$ in. in thickness. In larger converters this plate thickness may be increased to 1 in.

The air required depends on the composition of the metal being blown, but the same authors[7] suggest that this requirement will average about 17,720 cu. ft. per ton of metal treated. This figure is made up of:

Theoretical air required	12,637 cu. ft.
Excess necessary	5,083 cu. ft.
Total air required	17,720 cu. ft.

Therefore on a 5-ton converter the total air requirement will be:

$$5 \times 17{,}720 = 88{,}600 \text{ cu. ft. of air.}$$

Since the blow will take approximately 22 minutes

$$\frac{88{,}600}{22} = 4{,}027{\cdot}27 \text{ cu. ft. of air will be required per minute.}$$

Therefore the blower must have a capacity of at least 4,100 cu. ft. per min.

It is suggested that the velocity of the blast in the mains should not exceed 50 feet per second. If the blower is delivering 4,100 cu. ft. per minute, the diameter of the blast main should be:

$$A \times 50 = \frac{4{,}100}{60}.$$

Therefore:

$$A = \frac{4{,}100}{50 \times 60} = 1{\cdot}366 \text{ sq. ft.}$$

That would mean a diameter of approximately 16 in.

The blast pressure employed in side-blown converters is much less than that employed in bottom-blown converters. A pressure of 80 to 100 in. W.G. is sufficient for converters up to 2 tons capacity. For a 5-ton capacity vessel it is suggested that this pressure may be increased to 120 inches W.G

The velocity of the blast leaving the tuyeres must be sufficient to prevent them being slagged up, that is, it must be enough to blow the slag away from the tuyeres. It is suggested that the total tuyere area should be 5·27 sq. in. per ton of converter capacity.

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[2] Fassotte. *J.I. & S. Inst.*, 1944, Vol. 2.
[3] Harrison, Newell and Hartley. *J.I. & S. Inst.*, 1948, July.
[4] Jazwinski. *J.I. & S. Inst.*, 1945, Vol. 1.
[5] Lange. *Trans. Manchester Engs.*, 1916–17.
[6] Haag. *Stahl und Eisen*, 1925, Vol. 45.
[7] Eilender and Roeser. *Stahl und Eisen*, 1939, Vol. 59.
[8] Goldsbrough and Throssell. *British Steelmaker*, January, 1944.

CHAPTER 12

MODIFICATION OF STEELMAKING PROCESSES

Sometimes the normal methods of steel production already described are modified to meet local needs or the raw materials available. Occasionally two or three of the standard steelmaking processes are employed in conjunction with each other as a duplex or triplex process. The activated mixers which have been mentioned are in one aspect a duplex process when operated in conjunction with open-hearth furnaces or Bessemer converters.

The chief modifications are as follows:

Mixer Practice

In most melting shops employing a high percentage of hot metal, mixers are employed to act as reservoirs for the blast-furnace metal and to provide the steel furnaces with a metal more uniform in composition and temperature. Frequently these mixers are active, inasmuch as a certain amount of refining of the metal actually takes place.

Active mixers are usually lined with magnesite or dolomite bricks and carry a basic slag. The metal is treated with lime and iron oxides, which results in a partial desiliconization and desulphurization.

The mixer described by Geary[1] is used in conjunction with basic open-hearth furnaces. The average burden of the mixer per ton of mixer metal supplied to the open hearth-furnaces is:

	cwt.
Blast furnace metal ..	16·39
Cold pig iron	0·64
Steel scrap	2·15
Old moulds, etc.	0·21
Skulls, etc.	0·61
Limestone	0·14
Ore	0·08
Dolomite	0·13

It will be noted that in this practice a large percentage of steel scrap is employed, whilst the iron ore charged is comparatively low. The average composition of the metal entering and leaving the mixer is quoted as:

	C %	Si %	S %	P %	Mn %
Metal entering the mixer ..	3·60	0·79	0·06	1·67	1·42
Metal leaving the mixer ..	3·20	0·46	0·053	1·54	1·22

It will be noted also that there is some reduction in the carbon, sulphur and manganese. The reduction in carbon is probably due to the dilution by steel scrap. The silicon is removed by the combined action of oxide and limestone, the silicon being oxidized and passing into the slag as a calcium silicate. The drop in the manganese and sulphur is most likely related, and can be expressed by the reaction:

$$\mathrm{FeS} + \mathrm{Mn} \rightleftharpoons \mathrm{MnS} + \mathrm{Fe}.$$

This reaction is reversible and is influenced by temperature as shown by McCance. A falling temperature will favour the reaction proceeding in the direction of the upper arrow. This condition prevails in the mixer, hence the removal of sulphur by the action of manganese is accelerated. The removal of sulphur may also be assisted by the calcareous slag maintained.

An interesting modification of mixer practice is described by Brooke.[3] This practice is adopted at the Normanby Park Steelworks, Scunthorpe, and involves the use of two mixers. These mixers, shown in Fig. 103, have a capacity of 400 tons each and are fired with pre-heated blast-furnace and coke-oven gas. At the beginning of each week the mixers are full. One mixer is emptied to charge the first round on the steel furnaces, whilst the second mixer is reserved for the second round charges. When the second mixer is being emptied the first mixer is being replenished with blast-furnace metal.

The technical advantages claimed for this two-mixer process are summarized as follows:

(1) The blast-furnace manager is able to concentrate on the production of a hot low-sulphur iron, irrespective of the silicon content.

(2) The iron coming from the mixers is uniform in quality, thus enabling standard charges of lime and ore to be used in the steel furnaces.

(3) The iron is kept hot.

In addition to these direct advantages, this two-mixer process, when reviewed in relation to steel production in general, has resulted in:

(1) The total output of the melting shop being increased more by the addition of a second mixer than by the addition of a further steelmaking unit.

(2) A reduction in the conversion cost in spite of the fuel required to fire the second mixer and the increased capital and maintenance cost involved.

(3) A greater control of quality has been made possible.

The prime object of charging ore into the mixer is to reduce the silicon content and the degree to which this practice is effective is shown by the following figures taken from this Paper:[3]

	C %	Si %	S %	P %	Mn %
Metal into the mixers .. .. (9,433 tests)	—	0·99	0·045	1·74	1·41
Metal out of the mixers .. (9,400 tests)	3·36	0·33	0·026	1·55	0·84

The amount of lime and ore added to the mixer depends on the silicon content of the iron received from the blast furnace. It is possible to deal with blast-furnace metal containing up to 2 per cent. silicon without any serious trouble in the subsequent steelmaking practice.

It will be noted that there is a considerable reduction in the sulphur and manganese content of the metal whilst in the mixer. This reduction is probably due to the manganese-sulphur reactions already mentioned.

Although the mixers mentioned above are about 400 tons capacity, they are frequently much larger, holding up to 1,000 tons. It is possible, however, that there are decided advantages in having two mixers of 500 to 600 tons capacity rather than one mixer having a 1,000 ton capacity. This method allows them to be operated alternately, as in the two-mixer process. The provision of two mixers also facilitates repairs, which are inevitable.

Bertrand-Thiel Process

This process consists essentially of two open-hearth furnaces operated in conjunction with each other—one known as the primary and the other as the secondary furnace. The primary furnace is charged with pig iron, oxides and limestone, and a low temperature and a highly basic slag is maintained. These conditions result in the desiliconization and partial dephosphorization of the metal. Some of the carbon is also removed and after about four to four and a half hours the partially refined metal is transferred to the secondary furnace. Great care must be taken to prevent the siliceous primary slag passing into the secondary furnace.

Prior to the introduction of the hot metal from the primary furnace, steel scrap, lime and oxide are charged into the secondary furnace and are brought up to a high temperature. On the addition of the primary metal to this partially melted mixture of steel, lime and oxide a violent reaction takes place. After about two to two and a half hours the metal in the secondary furnace is refined to the required limits and is tapped in the usual manner.

Typical analyses of the original pig iron and the primary metal produced, taken from the paper by Bertrand,[4] are:

	C %	Si %	P %	Mn %
Pig iron	3·8	1·0	1·6	1·0
Primary metal	2·1	0·09	0·4	0·05

In some cases the plant was designed so that the metal from the primary furnace runs down a chute into the secondary furnace. At other plants where this practice was adopted two furnaces alongside each other were employed. At the Round Oak Steel Works, who were one of the first firms in this country to adopt the process, the same furnace was used both as a primary and a secondary furnace. After the primary refining the metal was tapped into a ladle, where it was held until the steel scrap, lime and oxide had been charged. The primary metal was then poured into this mixture. Very good rates of output were obtained.

More recent developments in the open-hearth process have caused a discontinuation of this practice.

The Talbot Continuous Process

The Talbot continuous process was introduced to accelerate the rate of conversion of molten pig iron into steel. The process was described by its inventor—Benjamin Talbot[5]—in 1900, whilst further developments were discussed in 1903.

The process is carried out in a basic-lined tilting furnace, which permits the removal of any quantity of slag and/or metal. The normal capacity of these furnaces varied from 200 to 300 tons. The furnace was started up in the normal manner by charging steel scrap and pig iron, which could either be solid or molten metal. This charge was then melted and refined to the usual limits. When the charge had been worked down to the specification, from 10 to 33 per cent. of the refined metal was tapped and teemed in the ordinary manner. The balance was left in the furnace and formed part of the next charge.

Molten metal equivalent in weight to the metal tapped, plus an allowance for depreciation, was added, together with the necessary lime. Most of the refining slag was removed before the finished steel was tapped and a new slag made with the necessary lime additions. This continuous process resulted in a considerably increased rate of output and ensured a more regular supply of hot ingots to the rolling mills.

This process was adopted by several large steel works where a high percentage of hot metal was employed. The fundamental principles of the process are still employed with some modifications in the hot metal basic open-hearth process. The chief difference is that a smaller percentage of refined metal is left in the furnace.

Monell Process

The Monell process was another method suggested for the conversion of high percentages of molten pig iron. In this process, described by Monell,[7] either a fixed or tilting furnace could be employed. Limestone and iron ore were charged into the furnace and raised to incipient fusion before the introduction of the molten metal. A violent reaction took place and a quick rate of conversion was obtained.

Twynam Process

The Twynam process was really a direct method of producing steel from iron ore. It consisted of throwing iron ore and carbon briquettes into a bath of molten metal. There was a tendency for the banks and bottom of the furnace to grow and the amount of ore reduced was limited, so that the process was finally abandoned.

Perrin Process

Some aspects of the Perrin process are similar to the process for the dephosphorization of acid Bessemer steel described by Yocom[8] and discussed in Chapter 2, page 29. The general principles of the process, which were described by Perrin[9] and which are covered by patent rights, consist of treating molten steel by pouring it from a considerable height, about 20 ft., into a previously prepared synthetic slag contained in a casting ladle. The process can also be applied by churning the molten steel and the synthetic slag together in a suitable vessel before it is cast into ingot form. Perrin showed that his process could either be employed for deoxidizing or dephosphorizing the molten steel by varying the composition of the synthetic slag.

When the process is employed for *deoxidizing* the molten steel, a highly siliceous slag is prepared. The silica content of this deoxidizing slag should not be less than 55 per cent., whilst the iron oxide should be kept low. The total iron oxide, $FeO + Fe_2O_3$, should be under 1·0 per cent. It is claimed that when steel is deoxidized in this manner solid inclusions are not formed as is the case when deoxidizers are added to the molten metal. In the process described it is suggested that the ferrous oxide is removed in accordance with the reaction:

$$2FeO + SiO_2 = Fe_2SiO_4.$$

As this deoxidation is proceeding some of the silicon enters the steel due to the reduction of silica from the slag. This method was shown by practical results to reduce the non-metallic inclusions in the finished steel. Satisfactory deoxidation was obtained when the slag weight was about 3 per cent. of the weight of the metal treated.

When the process is applied for *dephosphorization* a highly basic slag is essential. In this slag the lime should be over 55 per cent. and the iron oxide ($FeO + Fe_2O_3$) not less than 25 per cent. Perrin claimed that when this synthetic slag is churned with molten steel, the phosphorus is oxidized to phosphoric acid and combines with the lime in the slag to form calcium phosphate according to the equation:

$$3CaO + P_2O_5 = Ca_3(PO_4)_2.$$

In the early experiments there was some difficulty in controlling the manganese content of the finished steel due to the fact that the ferro-manganese addition was made to the steel before the dephosphorization. In some cases this resulted in a considerable loss of manganese and attempts were made to increase the manganese content of the synthetic slag to prevent this loss occurring. Finally the difficulty was overcome by adding the ferro-manganese in a red-hot state to the casting ladle immediately before dephosphorization reactions had subsided. This practice appears to be successful and an interesting example taken from Perrin's paper quoted in Table LIV. The author understands that the experiments were considered sufficiently satisfactory for the process to be adopted by the Tata Iron and Steel Co.

TABLE LIV

EXAMPLE OF DEPHOSPHORIZATION BY THE PERRIN PROCESS

(*Perrin*[9])

Weight of iron ore	47 tons.
Weight of synthetic slag used	3 tons.
Time slag poured into casting ladle	2·49 to 2·51 p.m.
Time blown metal poured down through into casting ladle	2·57 to 3·04 p.m.
Reaction in ladle	Moderate owing to slow pouring of the metal. The reaction continued for four minutes.

700 lb. of 75 per cent. ferro-manganese added at 3.08 p.m. (This alloy was red-hot). Started casting ingots at 3.13 p.m.

ANALYSIS OF METAL SAMPLES

	C %	Si %	S %	P %	Mn %
Blown metal	0·07	0·038	0·024	0·278	0·05
Blown metal after dephosphorization and after addition of ferro-manganese	0·11	0·008	0·025	0·052	0·38

ANALYSIS OF SLAG SAMPLES

	Before Mixing	*After Mixing*
SiO_2	6·21	11·76
FeO	8·10	20·76
Fe_2O_3	24·43	7·30
MnO	1·03	2·90
CaO	50·60	45·80
MgO	5·34	4·61
P_2O_5	0·98	3·62

In some cases basic open-hearth slag low in P_2O_5 was employed instead of the synthetic slag, but the dephosphorization was not satisfactory. The process appears attractive when it is necessary to refine pig iron which is too low in phosphorus for the normal basic Bessemer process.

The suggested *plant* and the *proposed method of operation* is outlined by Perrin.[9] Blast-furnace metal of suitable composition (1·10 to 1·30 per cent. Si and 0·30 to 0·60 per cent. Mn) is delivered in 60-ton ladles to the Perrin plant. This metal is poured into a hot-metal mixer, which supplies a 25-ton Bessemer converter as required.

When producing low-carbon steels, the iron is blown until the carbon is about 0·10 per cent. The slag mixture is then transferred hot from a slag mixer to a casting ladle of about 50-tons capacity. This ladle is about twice the normal capacity to accommodate the violent reaction which occurs during the dephosphorization. The blown metal is then poured from a height of about 20 ft. into this ladle, when a violent reaction occurs. When this reaction has subsided, red-hot ferromanganese is added to the casting ladle and after about five minutes the metal is teemed in the usual manner.

When producing high-carbon and alloy steels, the Bessemer blow is stopped at a higher carbon content, depending on the specification required. The dephosphorized metal is then transferred to acid open-hearth furnaces where it is quality refined, and where the necessary alloy additions are made.

At the Ugine Steelworks in France, the synthetic slag was prepared in a 5-ton Girod-Heroult basic-lined electric arc furnace. The slag consisted of a mixture of lime, iron ore and fluorspar. About 2½ tons of Armco iron were charged into the furnace to produce the necessary circuit and on top of this material the following slag-making ingredients were added:

	cwt.
Lime (CaO 85 per cent.)	28
Iron ore (FeO 65 per cent.)	16
Fluorspar (CaF_2 85 per cent.) ..	3½

The object was to produce a slag of the following composition which had been found to give good results:

CaO %	$FeO + Fe_2O_3$ %	SiO_2, Al_2O_3, MnO, etc. %
60	30	10

Duplexing

In some parts of the world in order to meet local conditions or irregularities in the raw materials available, two of the standard processes of steel production are employed in conjunction with each other forming a duplex process. Frequently the two processes employed in this manner are the Bessemer and the Open-Hearth Process. This combination results in quick rate of conversion and the low scrap requirements of the Bessemer process being linked with the greater control of the open-hearth process. Duplex processes usually employ a high percentage of hot metal.

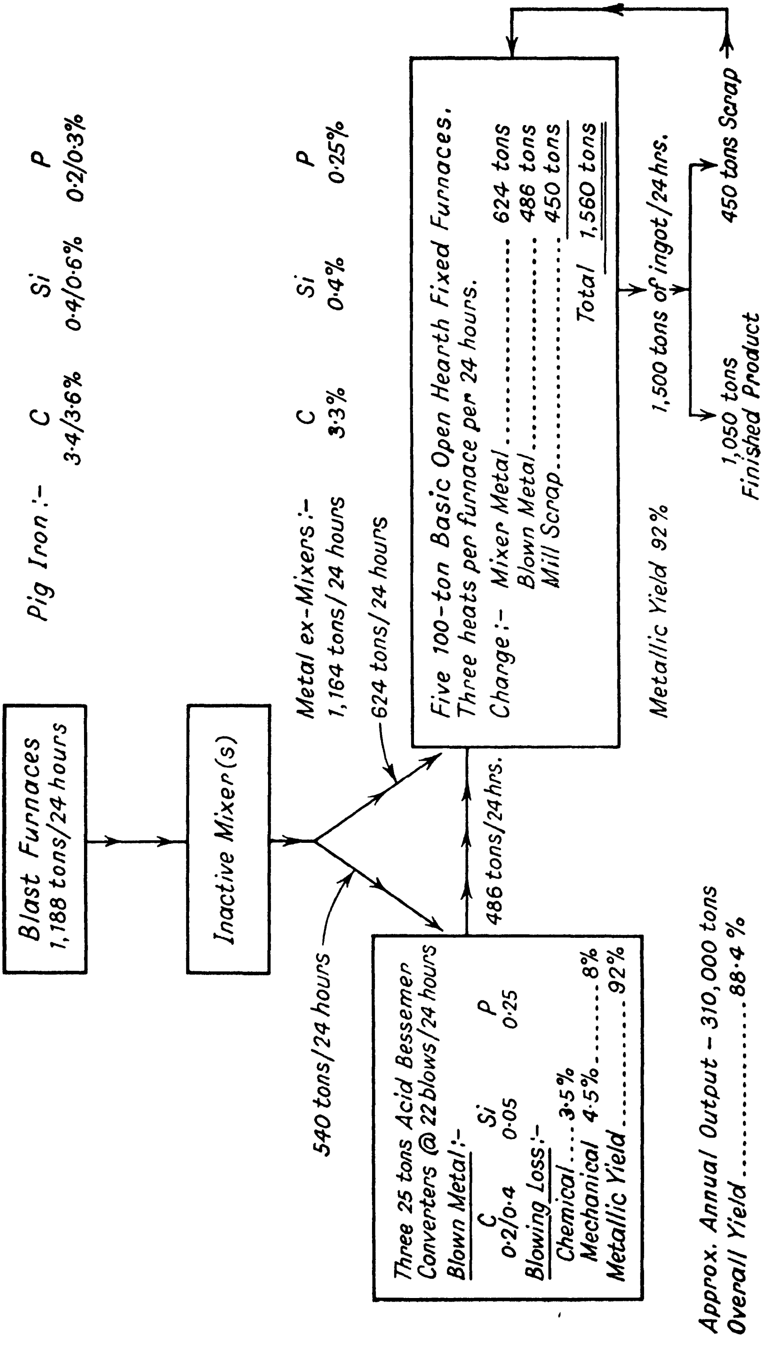

FIG. 172. Diagrammatic arrangement of a Duplex process.

The normal duplex process involves an all-liquid charge in the open-hearth furnace, which is frequently of the tilting type and operated in a semi-continuous manner. About two-thirds of the refined charge is tapped from the tilting furnace, and this furnace is then replenished with blown or partially blown metal from the Bessemer converter.

At other works stationary open-hearth furnaces are employed. The open-hearth charge may contain a certain portion of blown metal from the converters together with a percentage of high carbon or unrefined mixer metals. In such cases the blown metal replaces the steel scrap normally employed in the open-hearth charge.

In some parts of the world a very limited supply of steel scrap is available and this low percentage of low-carbon material would greatly prolong the conversion time in the open-hearth furnace. In order to increase the speed of conversion a certain portion of the open-hearth charge is made up of blown metal from the converter.

The combination of the open-hearth and the Bessemer converter involves increased capital cost and maintenance. Owing to the metallic loss which occurs in both units, the yield of metal is lower than when either unit is employed alone. The blowing loss in the converter, however, to some extent depends on the degree to which the metal is decarburized. The transfer of partially decarburized metal from the converter to the open-hearth furnace would tend to reduce the overall conversion loss.

Fig. 172 shows diagrammatically a duplex process suggested by the author for the conversion of a pig iron containing 3·5 per cent. C, 0·5 per cent. Si and 0·25 per cent. P. A pig iron of this composition is unsuitable for direct conversion in either the acid or basic Bessemer. Owing to the fact that only 30 per cent. of steel scrap, that is, the scrap arising in the works, is available, the rate of conversion in the open-hearth furnace would be low. A duplex process is, therefore, recommended. The use of partially decarburized Bessemer metal not only improves the ingot yield, but also assists the transfer of the metal from the Bessemer to the open-hearth furnace. The high carbon content reduces the tendency for the metal to skull the transfer ladles.

"Triplex" Processes

In a few localities where high-grade or alloy steels have to be produced from high percentages of pig iron, a triplex process is adopted. In a triplex process, the pig iron is partially refined in an acid Bessemer converter, where the silicon and part of the carbon are removed.

The blown metal from the converter is transferred to an open-hearth furnace, with or without the addition of mixer metal, where it is refined in the normal manner to the specification required. This refined metal is then transferred to basic electric arc furnaces where it is quality refined and the necessary alloy additions are made.

Cupola Open-Hearth Process

In a few localities the cupola is employed in conjunction with the open-hearth process. Instead of charging high percentages of steel and cast-iron scrap plus coke or anthracite into the open-hearth furnace, this steel and cast-iron scrap is melted in a cupola and the molten cupola metal is transferred to the open-hearth furnace. This method is adopted where there is a plentiful supply of cheap steel and cast-iron scrap.

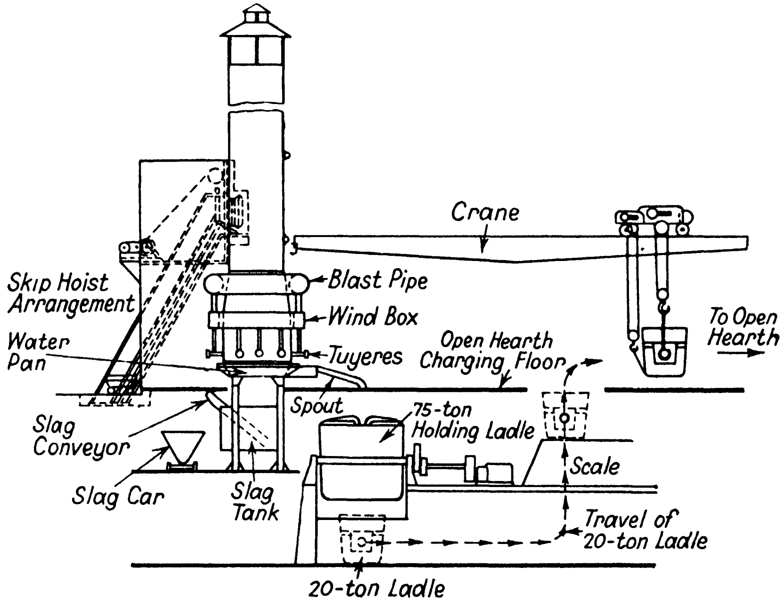

FIG. 173. Cupola plant.
(*By courtesy of " Iron Age ".*)

The cupola practice adopted is similar to that used in connection with the supply of molten metal for side-blown converters described in Chapter 11. The cupola metal replaces the molten pig iron employed in the hot-metal open-hearth process.

Recently the employment of cupolas in conjunction with open-hearth furnaces in America has been discussed by Kopecki[10] and the type of cupola employed and the general layout of the plant is shown in Figs. 173 and 174.

Owing to the fact that large quantities of high-sulphur cast iron are frequently melted with high-sulphur coke, the sulphur content of the cupola metal is often too high. This metal can, however, be treated with soda ash with a considerable reduction of the sulphur content. This treatment may take place in the ladle transferring the metal to the open-hearth furnaces, but better results can be obtained when a holding ladle is provided. It will be observed that a holding ladle, where the soda ash treatment is carried out, is shown in Fig. 173. This holding ladle enables the slag from the soda ash treatment to be separated from the metal before it is transferred to the open-hearth furnace. When no holding ladle is provided the soda ash treatment can be very successfully performed in a basic-lined ladle of the type shown in Fig. 175.

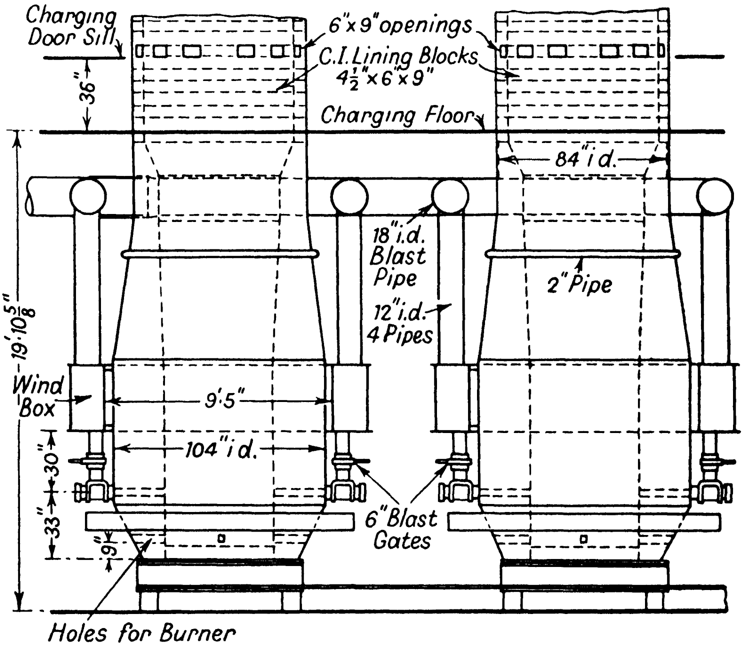

FIG. 174. Special design of cupola.
(*By courtesy of " Iron Age ".*)

Kopecki[10] quotes cases where the sulphur content of the cupola metal is as high as 0·15 to 0·20 per cent. This sulphur can be reduced by

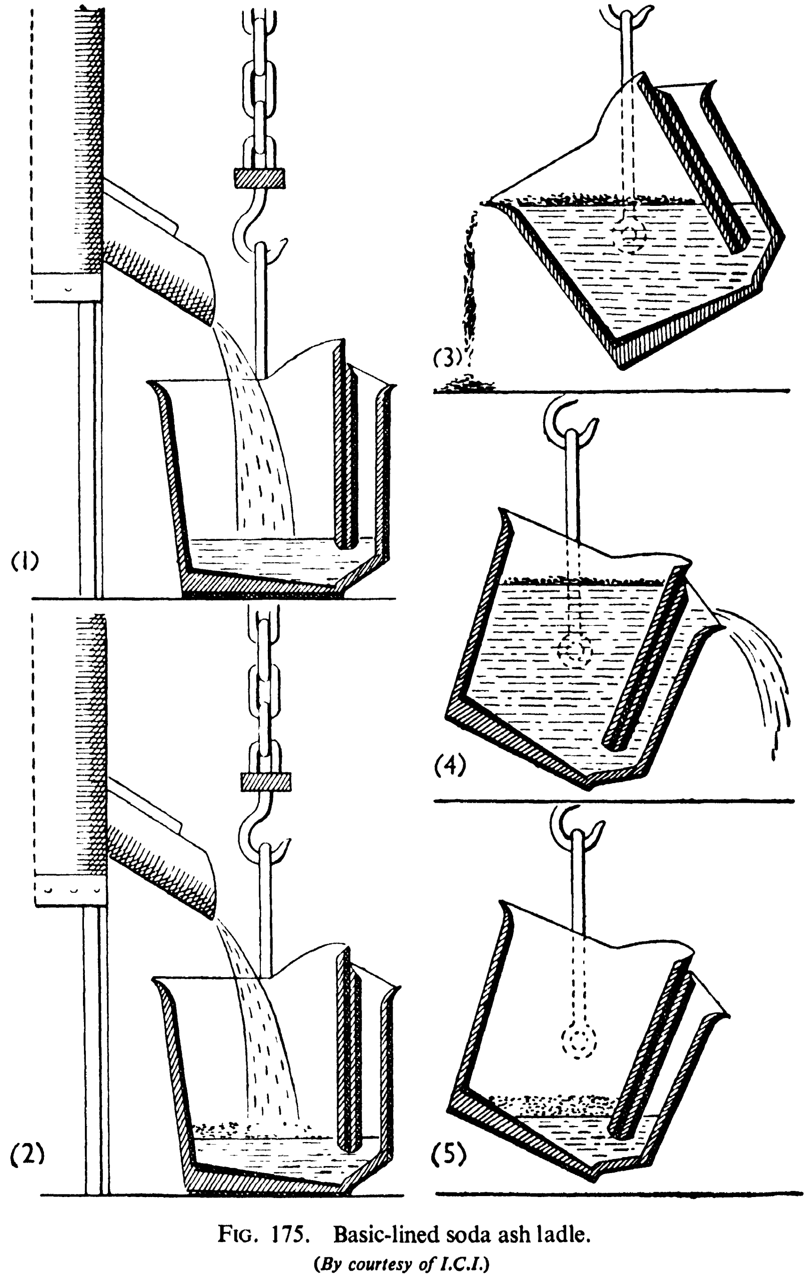

FIG. 175. Basic-lined soda ash ladle.

(By courtesy of I.C.I.)

soda ash treatment to 0·099 to 0·129 per cent. The normal range of composition of the cupola metal is:

C per cent.	Si per cent.	S per cent.	P per cent.	Mn per cent.
2·8/3·0	0·4/0·7	0·15/0·18	0·2/0·25	0·35

The cupolas described by Kopecki are capable of 72 hrs. continuous operation between repairs. It will be observed that the original lining is 24 in. thick and this thickness of refractory material reduces the diameter of the cupola at the melting zone to 56 in. Silica blocks are used in the melting zone above the tuyeres only. The blast is admitted through 12 tuyeres at a pressure varying from 8 to 24 oz.

At the Sheffield Steel Co., Kansas City, two cupolas are producing 11,645 tons of metal per month. These cupolas have a shell diameter of 108 in. and are provided with a 19-in. lining. Approximately 70 hrs. continuous operation between repairs is being obtained. The wall of the cupola to a point some distance above the tuyeres is lined with new silica blocks, whilst old silica bricks from the roofs of open-hearth furnaces are employed for the upper portions of the cupola.

The normal cupola charges appear to average 70 per cent. steel scrap and 30 per cent. cast-iron scrap. The coke ratio varies from 5 : 1 to 7 : 1.

It is obvious that the supply of molten cupola metal to the open-hearth furnace will considerably increase the rate of conversion in that furnace. Against this increased rate of conversion and its consequent economics, the capital, maintenance and operating cost of the cupola plant must be considered. Where steel scrap is cheap and plentiful, however, the provision of molten cupola metal for refining in the open-hearth furnace offers several technical and economical advantages.

The Economics of the Various Processes

When deciding the particular steelmaking process to be adopted, the following factors must be considered.

(1) The raw materials available.
(2) The type of steel to be produced.
(3) The economics of the various processes.

The qualities of the raw materials most suitable for the different processes and the range of application for the products of the various methods of steel production have been discussed in the appropriate chapters. It remains, therefore, to consider the economic factors of the steelmaking processes available.

The cost of any process can be analysed under the following headings:

(1) Materials.
(2) Wages or labour costs.
(3) Manufacturing charges.
(4) Capital costs.

Naturally the *material costs* will depend very largely on local market conditions. In districts where large quantities of steel scrap are produced those processes which permit the use of this scrap will be recommended. In composite steel works high percentages of molten iron will be employed in the furnace charge. Although this high percentage of pig iron will increase the cost of the mixture, some credit will be obtained from the iron ore reduced in the process, which tends to increase the metallic yield.

The acid open-hearth process requires low sulphur and phosphorus materials and this type of pig iron and steel scrap is expensive. The material costs of the acid open-hearth process will, therefore, be higher than the material cost of the basic open-hearth process. The acid process, however, is usually employed in the manufacture of special high-grade and alloy steels for which a higher selling price can be demanded.

The Bessemer processes both require a high percentage of pig iron which renders them high in material cost. The amount of scrap used in these processes is low.

Since the electric-arc process employs chiefly scrap the material costs of this process are low. In districts where large tonnages of alloy steels are used and where, therefore, considerable quantities of alloy-steel scrap arise, this process is recommended for the production of such types of steel. The conservation of alloying elements in the basic arc furnace is high.

The control of the metallic yield has an important effect on the material cost. The theoretical yields of the various processes have been discussed in the appropriate chapters and the various factors affecting this yield should be carefully controlled.

The *wage cost* is generally divided into direct and indirect wages. Sometimes these are called productive and non-productive wages. The direct or productive wages include all monies paid that can be charged directly to the productive processes. The indirect wages include those items which cannot be directly charged to the productive processes, such as general labourers, painters engaged in periodical painting of the plant, and similar charges. It follows that this item will largely depend on the layout of the plant and will not be greatly affected by which particular process is adopted.

The direct or productive labour will be affected by the process employed. Where large quantities of cold materials have to be loaded and charged into the furnaces, the wages paid to the loaders, etc., will be much higher than in those plants where large quantities of molten metal are cheaply transported in bulk. To some extent this particular item of economy will be off-set by increased equipment necessary and the maintenance thereon.

The *manufacturing costs* can be conveniently sub-divided into:

(1) Fuel.
(2) Productive services.
(3) Ladles, ingot moulds and pitside equipment.
(4) Repairs to plant.
(5) Repairs to furnaces.

The fuel costs will be greatly influenced by the process employed. In the Bessemer plants the fuel cost will be low. The only external heat required is that used for firing the mixer and for the heating of the converters, ladles, etc.

In the open-hearth processes the fuel costs vary according to the practice adopted and the type of fuel employed. In America on oil-fired open-hearth furnaces of 200 to 220 tons capacity working hot metal, the fuel consumption averages 3,300,000 to 3,500,000 B.Th.U. per ton of ingots. On smaller furnaces this figure increases to about 4,000,000 B.Th.U. per ton. The fuel consumption on furnaces operating the cold pig and scrap process ranges from about 6,000,000 B.Th.U. per ton of ingots on furnaces of 95 tons capacity, to 5,700,000 B.Th.U. per ton on furnaces of 140 tons capacity. This oil consumption on a thermal basis is equivalent to about 5·7 cwt. of coal per ton at the producers, which appears to be fairly normal cold-metal practice.

At a British plant quoted by Kerr[11] the fuel consumption varied from 4,000,000 to 4,500,000 B.Th.U. per ton of ingots. This consumption referred to furnaces of 50 tons rated capacity operating with 50 per cent. hot metal. Another British plant mentioned by Bainbridge,[12] employing 75 to 80 per cent. hot metal in a 240-ton tilting furnace had a fuel consumption of 3·62 cwt. of coal per ton. Assuming that the calorific value of the coal was 12,500 B.Th.U. per lb. this figure would be equivalent to 5,068,000 B.Th.U. per ton of ingots at the producers. If the efficiency of the producer plant is taken at 75 per cent., the consumption at the furnace is 3,801,000 B.Th.U. per ton of ingots.

At the plant discussed by Davies,[13] 200-ton tilting furnaces were being operated on approximately 70 per cent. blast-furnace and 30 per cent. coke-oven gas with a fuel consumption of about 5,000,000 B.Th.U. per ton of ingots. This plant used between 45 to 70 per cent. of hot metal.

It will, therefore, be seen that the fuel is affected by the use of hot or cold metal. It is also affected by the type or composition of the metal used. Where impure materials have to be used and large slag volumes are produced, that is, where the metallurgical load is high, a higher fuel consumption will result.

At the Australian plant discussed by Knight,[14] where a very good pig iron was available, and where a low slag volume was produced, the fuel consumption was about 3,700,000 B.Th.U. per ton of ingots on the furnaces fired with coke-oven and producer gas. On the furnaces fired with coke-oven gas and tar, the consumption was slightly lower. This plant was operating 75 per cent. hot metal and the furnaces were 230-tons capacity.

The actual fuel costs will depend on the type of fuel employed and the market conditions prevailing. When surplus coke-oven and blast-furnace gas are available, a low-cost fuel can be used. The economics of the various methods of firing open-hearth furnaces has recently been discussed by the author,[15] but changing market conditions quickly alter comparisons and fuel costs must always be examined under current conditions.

The fuel cost of the electric processes will be dependent on the price of electricity. This price will largely vary with the locality.

The productive services include steam, hydraulic and electric power required to operate the plant, which will to some extent depend on the degree of mechanization. The locomotive services will vary with the general layout of the plant. At plants where extensive water-cooling is employed the cost of water may be considerable.

The cost of ladles, ingot moulds, etc., are fairly constant irrespective of the process employed. In the case of hot-metal shops the extra cost of hot-metal ladles and their maintenance must be considered.

The repairs to plant are largely a function of the equipment installed and the degree of mechanization. For purposes of analysis this item can be conveniently divided into two items—wages and materials.

The cost of repairing furnaces will be higher in open-hearth plants than in Bessemer shops, where the item is usually low. At plants employing large percentages of cold materials the refractory costs will be higher than at hot-metal plants.

The final division of the productive cost is the capital outlay involved in the erection of the plant. It is estimated that the capital cost of an open-hearth plant capable of producing 500,000 tons of ingots per year would vary between £2,250,000 and £2,500,000 according to the type of furnace installed. A tilting-furnace shop to produce this output would require about five 300-ton furnaces and an active mixer costing about £2,500,000. This is equivalent to about £5 per ton of annual output. A fixed-furnace shop of the same capacity would entail less capital expenditure, amounting to approximately £4·5 to £4·75 per ton of annual output. About eight fixed furnaces of 100 tons rated capacity would be required.

The capital cost of a basic Bessemer plant to provide this annual output would be cheaper and would probably represent an outlay of about £4 to £4·25 per ton of annual output. The capital cost of an acid Bessemer plant would be still lower.

Several years ago Carnegie[16] reviewed the cost of liquid steel, but since that time prices and costs have changed. His figures give some guide to the economics of the various processes. The constant fluctuations in the price of raw materials, rates of wages, and the cost of plant and equipment make comparisons extremely difficult.

It will, however, be noted that the Bessemer permits a high rate of conversion, but entails a high material cost. The open-hearth processes give a lower rate of conversion, but in general the material cost is lower. The more regular supply of hot ingots to the rolling mills, which is possible in the case of the Bessemer processes, is an economic advantage which will be reflected in the reheating costs at the rolling mills. It is obvious therefore that the economics of any process is a very involved matter, which demands very careful investigation on a wide basis, before any definite opinion can be satisfactorily expressed.

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CHAPTER 13

BLISTER, SHEAR AND CRUCIBLE STEEL

No work on the manufacture of iron and steel would be complete without some reference to the production of blister, shear and crucible steel. These three processes represent the earliest method of steel production, and the crucible process is still operated on a fairly extensive scale for the manufacture of certain high-grade steel.

Cementation—Blister Steel

The origin of this process is lost in antiquity. It has been employed in Sheffield for a great many years and is based on the principle that when iron is heated in contact with carbon, the iron absorbs carbon. The amount of carbon absorbed depends on the temperature and the length of time the two elements are in contact. When the iron bar is heated to a welding heat, the carbon slowly travels into the bar, and thus the iron becomes carburized.

The Process

The conversion takes place in vessels called converting pots. These pots are usually four feet deep, four feet wide and about twelve feet long. They are made of sandstone, which is found in the vicinity of Sheffield, and which has the property of withstanding the temperature changes connected with the process. Slabs of this sandstone are formed into pots and are held together with mortar made from fireclay.

Two of these pots are built up on either side of a fireplace and are supported by bricks so that the heat or flame can pass all around them. The general arrangement of the furnace is shown in Fig. 176. At each end of the pot there is a hole from which trial bars can be withdrawn, and the progress of the conversion can thereby be observed.

Charcoal is employed as the carbonizing agent and this material is ground to about one-quarter to half an inch. The charcoal is carefully riddled through a sieve to get rid of the dust. A layer of about one inch of charcoal is laid on the bottom of the pot and a layer of iron bars is put on the top of this material. The bars are usually about three inches wide and about half an inch thick. The spaces in between the bars are filled up with charcoal. Another layer of charcoal is placed on these bars, followed by a further layer of bars and so forth until the pot is filled. Each pot is capable of holding up to 30 tons of bars.

When the pots are filled they must be made airtight to prevent the carbon being burnt away and the iron oxidized. Sheffield practice is to employ "wheel swarf", which is finely divided steel mixed with siliceous sandstone—the mud from the grinding wheel troughs—for this sealing. This material is spread over the top of the pots and the fire started.

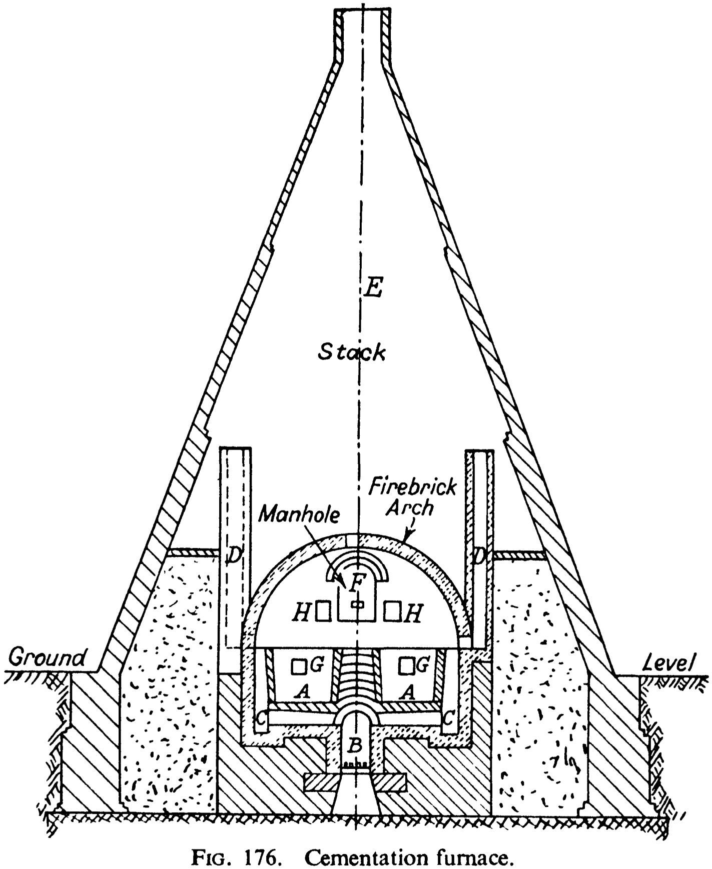

Fig. 176. Cementation furnace.

A. Converting post. *B*. Firegrate. *C*. Flue for distributing heat. *D*. Short chimneys communicating with stack. *E*. Conical hood or stack. *F*. Manhole for introducing or withdrawing charge. *G*. Holes for removal of trial bars. *H* Charging holes.

The temperature is brought to a full redness, that is, a welding heat of about 800 deg. C. to 900 deg. C, sometimes to a yellow heat about 1,150 deg. C. This will occupy about two days and the temperature is then maintained at this point for seven to ten days according to the quality of steel being made. The production of high-carbon grades requires a longer time for conversion. Trial bars are withdrawn from time to time to ascertain the progress of the carburization. When the desired carburization has been obtained the fires are withdrawn and the pots are allowed to cool down slowly. This cooling occupies about a week. When cool the pots are opened, the charge withdrawn and the fractures examined. The whole process takes about three weeks and therefore about fifteen to sixteen charges can be worked in each furnace per year. Each pair of pots can be used twenty to forty times before they require repairing.

The Products

A fair amount of the charcoal is consumed and a varying amount of fine powder is produced. This material is sifted and the larger material used again.

During conversion certain changes in the bars occur:

(1) The bars increase in weight to the extent of about one per cent.

(2) Although quite smooth when charged, the bars become rough and covered with blisters. This characteristic accounts for the term "blister steel".

(3) The appearance of the fracture changes. Before conversion the fracture of the bars exhibits the fibrous wrought-iron appearance, whilst after conversion they assume a coarsely crystalline fracture.

The finished bar shows degrees of carbon penetration, and the change travels from the outside towards the interior. The core of iron is called the sap.

The product is graded according to the extent of its conversion. The following are the normal grades:

	Per cent. carbon
No. 1. Spring heat	½
No. 2. Country heat	⅝
No. 3. Single shear	¾
No. 4. Double shear	1
No. 5. Steel-through heat	1¼
No. 6. Melting heat	1½
No. 7. Double converted or glazed bars	

In No 1 Grade the outer skin only is converted into steel, whereas in No. 2 Grade the crystals have become more distinct and the sap has been more "killed". As the grade increases so the amount of converted skin increases and the sap diminishes. In melting-heat grade the crystals will be very large and the facets will normally extend right through the bar.

In good converted bars the line of demarcation should not be too marked, but should rather pass slowly and gradually from the exterior to the centre. If the line of demarcation is too sharp the conversion has been too rapid and the bars are said to be *flushed.*

Sometimes the pots crack and the bars become oxidized in which case they are referred to as *Aired Bars.* If the temperature is raised too high the bars will melt and they become *Glazed Bars.*

Theory of the Process

Two changes have to be accounted for:

(1) The transmission of carbon into the metal.

(2) The formation of the blisters.

Transmission of carbon into the metal has been explained in several ways. This diffusion may be solid diffusion or it may result from the formation of carbon monoxide by the interaction of the charcoal with the oxygen of the air entrapped within its pores. At an elevated temperature in contact with iron this carbon monoxide is partially decomposed:

$$2CO \rightleftharpoons CO_2 + C$$

the solid carbon being absorbed by the iron.

It has been shown that the iron can be converted to the point of saturation (0·9 per cent. carbon) by heating in contact with carbon up to a temperature of 800 deg. C. but supersaturation, or the production of higher grades, necessitates high temperatures.

The *blisters,* which appear on the surface of the bar and which are usually hollow, are presumably formed by liberated gas. The wrought-iron bars contain a certain amount of slag. The iron oxide of this slag reacts with the carbon present, to form carbon monoxide. If high-grade, well-worked wrought iron is employed, the slag will be uniformly distributed as small particles throughout the bar. Bars of this quality will produce blister steel which is characterized by small, uniformly distributed blisters. Large or irregularly distributed blisters are indicative of low-quality steel resulting from the employment of a low grade of wrought iron.

Blister steel is chiefly used as the raw material for the manufacture of shear steel, or for melting in the production of crucible steel.

Shear Steel

At one time a large quantity of blister steel was converted into shear steel, which was a more uniform product. Bars of blister steel were broken into pieces about ten inches long and drawn out under a hammer until they were about half an inch thick and about one and a half inches in width. Several of these bars were piled, heated to a welding heat and welded under a hammer. In the process of hammering the bars were reduced to about two inches square. The product was known as "*Single Shear Steel*".

"*Double Shear Steel*" was produced by piling and welding together bars of single shear steel. Obviously double shear steel possessed a more uniform distribution of carbon and impurities.

Crucible Steel

In spite of the progress in the art of steelmaking, crucible steel is still being produced in very much the same manner as it was many years ago. Its products are usually high-grade alloy and special steels, such as high-speed tool steel, magnet steel, etc. Even in this field, however, there is evidence that the crucible process is slowly being replaced by the high-frequency electric furnace.

The crucible process appears to have been the first method of producing cast steel and it was introduced by a clockmaker called Huntsman. The only raw material for his graft was either Hindoo Wootz Steel at about five guineas per pound or selected converted bar. Owing to the fact that both these materials contained a large amount of impurities, Huntsman conceived the idea of melting these materials in a crucible, thereby producing a more homogeneous product.

As Lange[1] suggests it is difficult now to imagine the technical problems which Huntsman had to overcome. One of his chief difficulties was obtaining a crucible sufficiently refractory to withstand the high temperature and conditions of service. Other problems included the provision of a sufficiently high temperature and the production of a sound ingot. After many years of patient experiments in Doncaster, he established the first crucible steelworks in Sheffield in 1740. The process which he initiated was so successful that it held the field in steel production until Bessemer's invention nearly 150 years later.

Huntsman's original process has been little modified. Various improvements have been suggested from time to time, and some developments have been adopted, but the greater proportion of the crucible steel being made to-day is still made in the old-fashioned coke-fired crucible holes, such as employed by the inventor and his immediate successors.

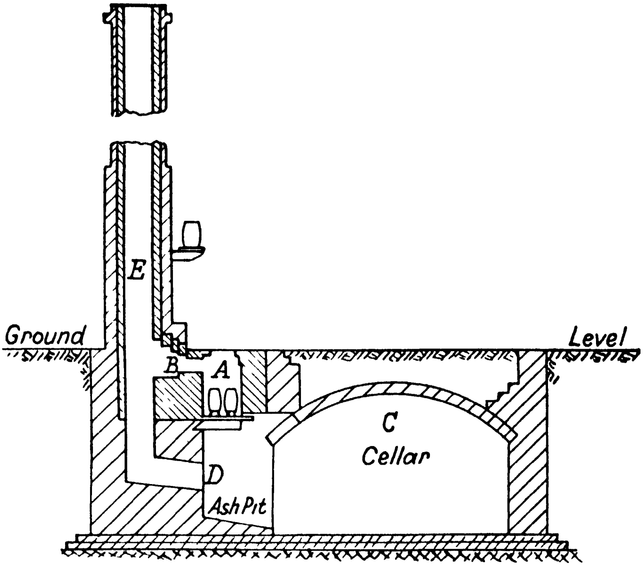

FIG. 177. Crucible furnace.

A. Melting hole or fire hole. *B*. Flue. *E*. Chimney. *D*. Opening communicating with chimney flue, which can be partially or entirely closed by a brick to control the draught.

The Furnace

The melting furnace consists of a row of holes in the ground as shown in Fig. 177. These holes, which are oval in cross-section, are usually about 2 ft. 6 in. by 2 ft. and are about 6 ft. deep. They are designed to receive two pots or crucibles. Ten or twelve holes are arranged in a row and their flues enter a long low stack.

The tops of the holes are level with the ground line and each hole is covered by a lid, which consists of a steel frame lined with firebricks. The furnace is a firebrick construction, the hotter zones of which are faced with ganister.

The fuel employed is usually hard burnt coke, the combustion of which is assisted by natural draught. The air supply, which passes through the firebars, is derived from a large cellar, which is so constructed as to assure an ample supply of air. The draught, and thereby the air supply and the temperature of the furnace, is regulated in a primitive manner, which consists of inserting or withdrawing a brick in the draught-hole or flue leading from the cellar to the stack flue.

The Crucibles

The crucible employed is made either of fireclay or graphite. In Sheffield clay crucibles are generally employed. These crucibles are composed of various mixtures of fireclay, which vary in the different works. They are actually made up in the particular works and essential points to be noted in their manufacture are the careful and thorough mixing of the various clays and adequate kneading of the mixture.

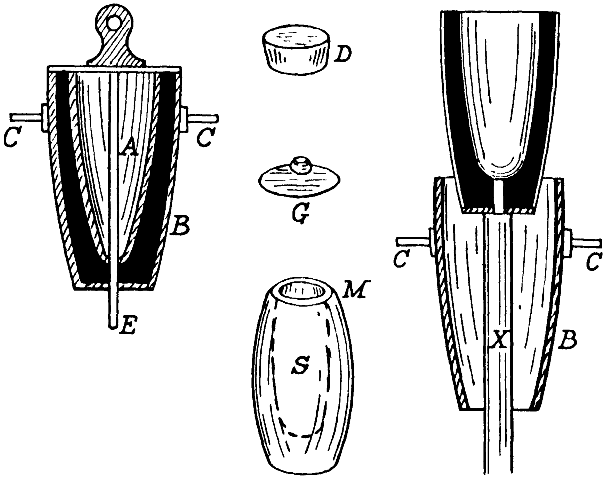

FIG. 178.

A. Wooden plug. *B*. Flask or mould. *C*. Trunnion arms on flask. *E*. Pin for centering plug, A. *X*. Stripping post. *S*. Crucible as it leaves mould. *M*. Crucible in final shape. *D*. Circular firebrick to stand pot on in furnace. *G*. Lid.

A ball of clay is transferred to a moulding flask, as shown in Fig. 178, and a wooden plug, provided with a guiding pin, is inserted. This plug, which is fitted with a distance piece to ensure the correct thickness of the crucible walls, is rotated several times before withdrawal. The top of the crucible is roughly shaped before being placed on the withdrawal table. The crucible lids and the circular discs, on which the crucibles stand in the furnace, are made of the same material.

After the crucible has been formed it is carefully dried, first on shelves in the pot house and subsequently on shelves situated along the stack flue in the melting house. This drying usually occupies about ten to fourteen days. The night before the crucibles are used they are placed in an annealing oven where they are brought up to a full red heat. Flather[2] strongly stresses the importance of thorough drying and annealing.

Graphite pots, which consist of equal parts of graphite and fireclay, are manufactured in a similar manner. This type of pot is more expensive, but gives a longer life. A fireclay crucible generally gives about three heats, whereas a graphite crucible may last for six or seven heats. With graphite pots the carbon pick-up, especially when producing low-carbon steels, must be considered. The carbon pick-up according to Hall[3] varies from 0·15 to 0·40 per cent. The carbon increase is greatest when the charge is high in manganese. New pots favour a higher carbon pick-up.

For the production of low-carbon steels fireclay crucibles are universally employed. These crucibles are usually about 9 in. external diameter and from 16 to 19 in. deep. When new these crucibles hold about 60 lb. of metal. During melting some erosion of the slag line occurs, which renders this part of the pot weak. In subsequent charges the position of the slag line is dropped, so that on the second and third charges the weight of the charge is dropped to 54 and 48 lb. respectively.

The Process

Most crucible plants operate on days only. In the morning the coke-hole is started with a small coal fire and a newly annealed crucible is placed in the hole, which is filled with coke level to the top of the crucible. The fire is then urged so that the pot and furnace is brought up gradually to a full red heat. A little sand is then thrown into the crucible, which fills the hole in the bottom formed by the guide in the wooden plug, and also welds the crucible to the fireclay disc on which it is placed.

The charge, which has previously been weighed up in a scoop or basket, is charged into the crucible by means of a charging funnel. The lid is then placed on the crucible and the fire made up. During the melting, which takes from three to four hours, the melter carefully maintains his fire and observes the progress of melting.

When the melting is complete and the boil on the metal has subsided, the *killing fire* is added. The killing process, which is still employed at many works, consists of making a fresh coke addition and bringing the metal up to a high temperature, during which carbon reacts with the fireclay crucible resulting in the reduction of silicon. The silicon reduced in this manner passes into the steel and kills the oxides formed or the gases occluded during melting.

At some plants the old-fashioned killing process has been replaced by the appropriate additions of ferro-manganese and ferro-silicon at the end of the melt, when the metal is sufficiently hot to be teemed.

Teeming

When the melter is satisfied that the steel has been killed, the crucible is lifted out of the furnace by means of a pair of tongs on to the ground level and placed alongside a trough in which the metal moulds are set. The crucible is then gripped around its middle by another pair of tongs, the lid is removed and the contents of the crucible are poured into the ingot moulds.

As each crucible is emptied it is replaced into the furnace as soon as possible and brought up to a full red heat in readiness for the next charge. Usually three charges per crucible are worked per day.

The ingot moulds are iron castings about 3 in. square internal cross-section and about 30 in. deep. They are made up of two halves split longitudinally and the two halves are normally held together by means of rings and wedges. Before use they are coated with soot, their inner surface being smooth and often machined. Frequently a small fireclay dozzle is inserted in the top of the mould to reduce the piping. This dozzle also assists the teemer in directing the stream and avoiding splashing the side of the mould.

The ingots are subsequently ended, that is, the ends are broken off. This *ending* serves a twofold purpose. It clears the piped portion of the ingot and allows the melter to judge the grade of metal from the fracture obtained. This ending is done when the ingot is cold. The ingot is then *wash-welded.* Wash-welding consists of reheating the ingot and hammering it down in cross-section. This hammering results in the welding up of any flaws or defects in the metal. The bars obtained from wash-welding may be subsequently rolled to the desired section, but generally they are regarded as the final product.

The Raw Materials

Since there is no refining in this process the raw materials employed must be pure. Formerly it was considered essential to use only such materials as Swedish Bar Iron, Blister Steel, high-grade wrought iron and similar materials. A more precise control of raw materials by analytical means has permitted the employment of a much wider range of raw materials, such as ingot iron scrap, open-hearth scrap, etc. Some charges are recarburized by charging charcoal into the crucible.

For calculating the composition of the charge the lb. per cent method is usually employed. For example, if it is desired to produce a steel containing 0·60/0·70 per cent. C, 0·40/0·50 per cent. Mn and 1·40/1·60 per cent. Cr in a 60-lb. crucible, with the following raw materials:

	C per cent.	Mn per cent.	Cr per cent.
Swedish pig iron	3·5	0·40	—
Ingot iron scrap	0·05	0·05	—
Ferro-manganese	6·00	80·00	—
Ferro-chrome	2·00	—	68·00

the method of determination would be:

The first item to calculate is the *chromium content* and the amount required can be determined:

$$\frac{\text{Per cent. of Cr required} \times \text{Wt. of charge in lb.}}{\text{Per cent. of Cr in alloy employed}}$$

thus:

$$\frac{1{\cdot}5 \times 60}{65} \text{ gives } 1{\cdot}385 \text{ lb. or approx. } 1 \textit{ lb. } 6 \textit{ oz.}$$

This ferro-chrome will also introduce:

$$\frac{1{\cdot}385 \times 2{\cdot}0}{60} \text{ approx. } 0{\cdot}045 \text{ per cent. C.}$$

Some carbon will also be introduced by any ferro-manganese added, therefore, we must aim at approximately 0·60 per cent. C. The proportions of Swedish pig iron and ingot iron scrap may be determined: Swedish pig iron will give 3·5 less 0·60, or a surplus of 2·90 per cent. C. Ingot iron scrap will give 0·60 less 0·05, or a deficiency of 0·55 per cent. C. Therefore, these constituents must be mixed in the proportions of 2·90 : 05·5 or 5·3 *ingot iron scrap to* 1 *Swedish pig iron.* Mixing in this manner would give the following *manganese*:

5·3 lb. × 0·05 per cent. Mn	0·265 lb./ per cent. Mn
1·0 lb. × 0·40 per cent. Mn	0·400 lb./per cent. Mn
divided by wt. – 6·3	0·665 lb./per cent. Mn
	0·105 per cent. Mn

Therefore the *ferro-manganese* additions would be:

	Per cent.
Manganese required	0·47
Manganese in mixture ..	0·10
Manganese to be added ..	0·37

By the lb./per cent. method:

$\frac{0{\cdot}37 \times 60}{80}$ gives 0·27 lb. of ferro-manganese or *approximately* 4½ *oz.*

The total charge is to weight 60 lb. of which 1 lb. 10½ oz. is made up of alloys, leaving approximately 58 lb. 6 oz. or 58·4 lb. of pig iron plus scrap, which will be employed in the proportions 5·3 : 1·0. Therefore the units are:

$\frac{58{\cdot}4}{6{\cdot}3}$ which is 9·27 *lb. of pig iron and*

9·27 × 5·3 or 49·13 *lb. of ingot iron scrap.*

The final charge is:

Swedish pig iron	9 lb.	4 oz.
Ingot iron scrap	49 lb.	2 oz.
Ferro-chrome	1 lb.	6 oz.
Ferro-manganese	—	4½oz.
Total	60 lb.	0½ oz.

The *analysis of the charge* should be:

	Carbon lb/per cent.		*Manganese* lb./per cent.	
Pig iron	9·25 × 3·5 =	32·38	9·25 × 0·40 =	3·70
Ingot iron scrap	49·125 × 0·05 =	2·46	49·125 × 0·05 =	2·46
Ferro-mang. ..	0·27 × 6·00 =	1·62	0·27 × 80·0 =	21·60
Ferro-chrome ..	1·375 × 2·00 =	2·75	— —	—
Total	60·02 lb.	39·21		27·76
that is		0·65 per cent. Carbon		0·46 per cent. Mn

The final check on the chromium addition gives:

$$\frac{1{\cdot}375 \times 65}{60{\cdot}02} \quad \text{or} \quad \underline{1{\cdot}489 \text{ Cr.}}$$

Chemistry and Control of the Process

As already stated the crucible process is merely a melting medium and little or no refining takes place. Nevertheless, certain reactions do occur in the course of the process, and the nature of these reactions largely depends on the raw materials employed. Any rust or oxide adhering to the raw materials forms an *oxidizing slag* at the early stages of the melt and this slag attacks the walls of the crucible forming a ferrous-silicate slag, which may contain some oxide of manganese.

This oxidizing slag is reduced as the melting proceeds by carbon present in the charge, and providing a sufficiently high temperature is carried and sufficient time is allowed, some of the silica in this slag will be reduced, the silicon passing into the metal thus:

$$SiO_2 + 2C \rightarrow Si + 2CO.$$

This reaction formed the basis of the "killing" practice formerly employed. The amount of silicon reduced in this manner is influenced by:

(1) Ferro-manganese additions tend to increase the amount of silicon reduced.

(2) High temperature and a long killing time increases the silicon reduced.

(3) The use of plumbago crucibles increases the amount of silicon reduced. According to Howe[4] when using plumbago crucibles the silicon increment may be as high as 0·60 per cent. With clay crucibles, without any prolongation of the killing time, the minimum silicon increase appears to be about 0·10 per cent.

(4) Oxide additions, however, would reduce the tendency for silicon to be reduced.

The increase of *carbon* when employing graphite or plumbago crucibles has already been mentioned. If producing low-carbon steels fireclay crucibles must be employed. When melting in fireclay crucibles some loss of carbon occurs. This loss varies from about 0·10 to 0·40 per cent. depending on local practice, the manganese content of the charge and the melting time. In certain works the carbon content of the charge may be made up with charcoal. This charcoal is charged with the scrap, etc., at the commencement of the process. In high carbon charges the charcoal can be considered to contain 85 per cent. of carbon, whilst on lower carbon heats the effective percentage of carbon from the charcoal varies between 65 and 70 per cent.

If *manganese* is added with the charge some loss occurs but when this ferro-manganese addition is made just before the crucible is drawn, the loss should not exceed 10 per cent. of the manganese in the alloy. The killing fire is frequently replaced by an addition of ferro-silicon immediately before the crucible is drawn and there should be no loss of silicon.

When alloys such as ferro-*chrome*, ferro-*tungsten*, ferro-*molybdenum* and metallic *nickel* are employed, they are generally added to the crucible with the charge and no loss occurs. Any ferro-*vanadium* additions should be made just before the crucible is drawn and a loss of about 10 per cent. is normal.

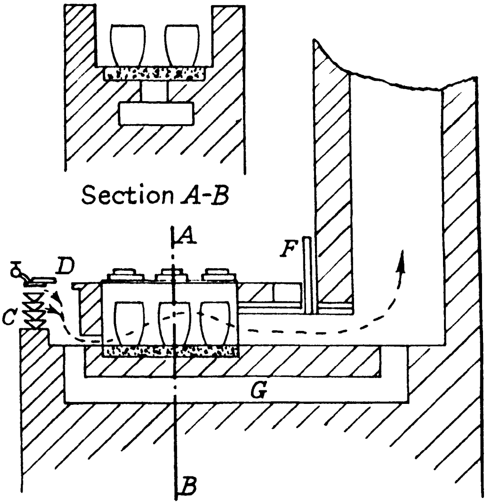

FIG. 179. Oil-fired crucible furnace.

Any *phosphorus* present in the raw materials passes into the steel and, therefore, this element must be kept low. Actually, due to a slight melting loss during the process, this element will increase slightly. *Sulphur* generally increases due to the penetration of the sulphurous fumes from furnace atmosphere through the walls of the crucible.

Modern Developments

In this country little change has taken place in the methods of producing crucible steel. Coke-fired crucible holes are still employed and

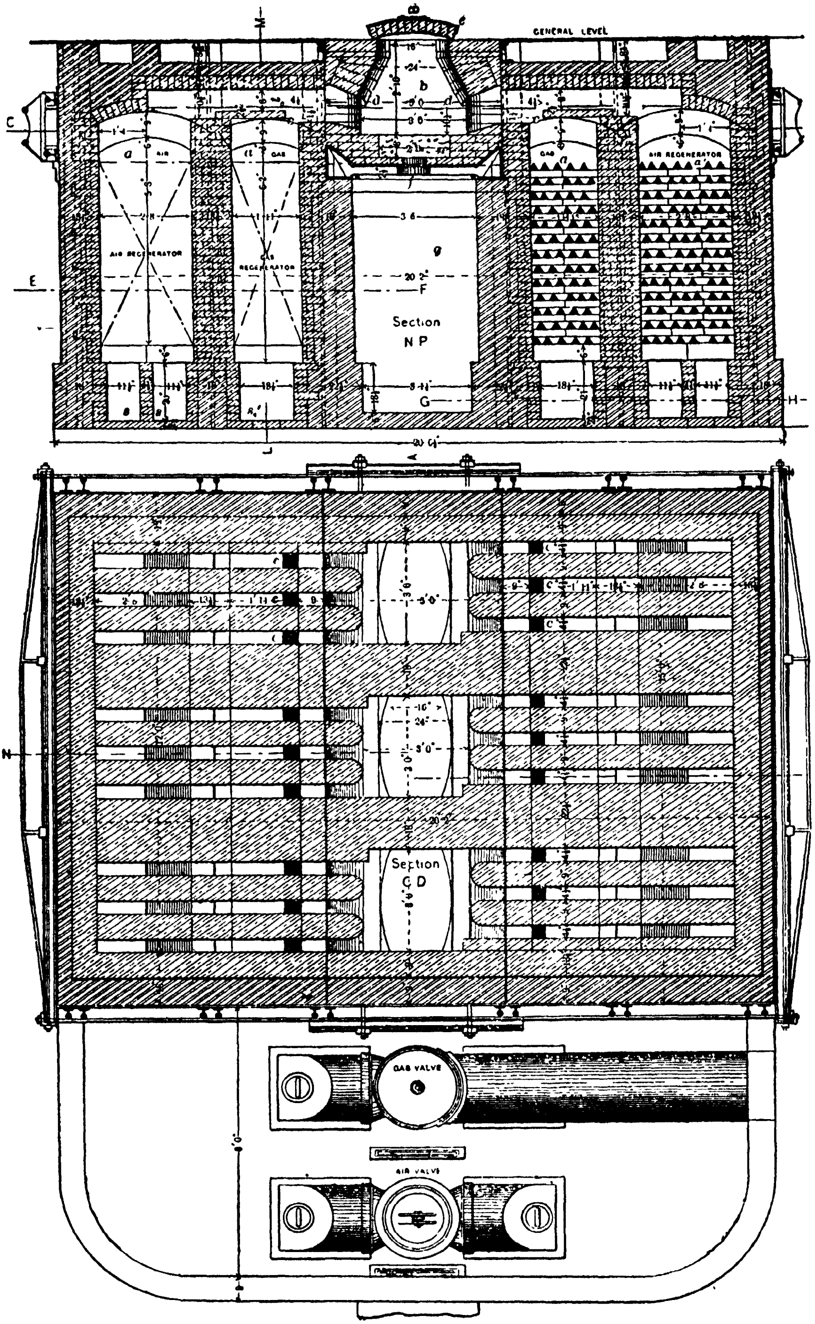

FIG. 180. Regenerative-type crucible furnace.

(By courtesy of Manc. Assoc. Engineers.)

these furnaces consume about three tons of high-class coke per ton of steel produced. In America oil-fired crucible furnaces have been adopted by some foundries, but no oil consumptions are available. The general design of oil-fired furnaces is shown in Fig. 179.

According to Hall[3] the furnace generally employed in America is the Siemens regenerative furnace, shown in Fig. 180, which is taken from a paper by Lange.[1] These furnaces are fired with producer-gas or natural gas. Harbord[5] gave the fuel consumption of producer-gas regenerative furnaces at 1½ tons of coal per ton of steel produced.

The chief disadvantage of these special furnaces is the higher capital cost as compared with the simple coke-fired. The coke-hole furnace is cheap to instal and very suitable for intermittent operation, which is the normal practice in crucible steel production.

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[2] FLATHER. *Jrn. Staffs. I. & S. Inst.*, 1901–02.
[3] HALL. *The Steel Foundry*, McGraw-Hill, New York, 1914.
[4] HOWE. *Metallurgy of Steel.*
[5] HARBORD. *Metallurgy of Steel*, Chas. Griffin and Co., London, 1903.

CHAPTER 14

STEELWORKS INSTRUMENTS

In recent years there has been considerable progress in the application of instruments to the control of metallurgical processes. This development has been made possible by the improvement in the type of instruments available and their increased accuracy and sensitivity. One feature in this progress has been the greatly accelerated speed of action, which is essential for instruments employed in process control. In some cases quicker action and increased sensitivity has rendered possible the automatic control of operations.

The application of the spectroscope and the photoelectric cell in the control of the Bessemer blow has been discussed in Chapter 2, page 18. The work of Jazwinski[1] in connection with the spectroscope and its application to the control of the Tropenas blow is mentioned on page 334. The other applications of instruments in steelworks include open-hearth furnace control and immersion pyrometer for liquid steel.

Open-Hearth Furnace Control

Instruments employed in connection with the open-hearth furnace are designed to control the following phases for furnace operation:

(1) Combustion control.
(2) Furnace pressure control.
(3) Reversal control or regenerative temperature control.
(4) Roof temperature control.

It would appear that the order in which they are stated above represents the order of their importance in the efficient operation of the furnace.

Combustion control consists of regulating the fuel/air ratio by means of flow-meters. This ratio may either be maintained manually, employing the instruments as guides, or the instruments can be adapted to operate the control valves. In this manner any adjustment of the fuel input to the furnace automatically regulates the air flow so that a constant fuel/air ratio is maintained. Typical layouts for this control are shown in Figs. 181 and 182.

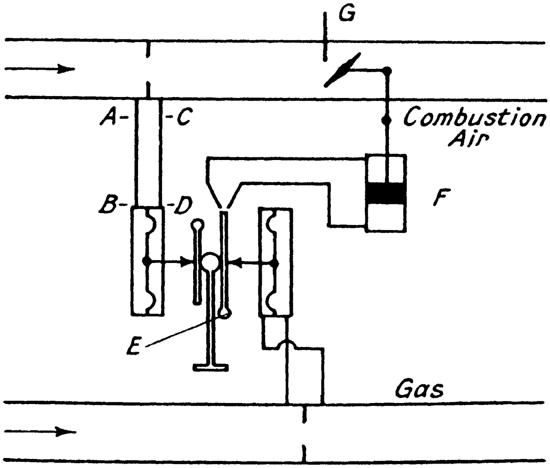


FIG. 181. Fuel/air ratio control.

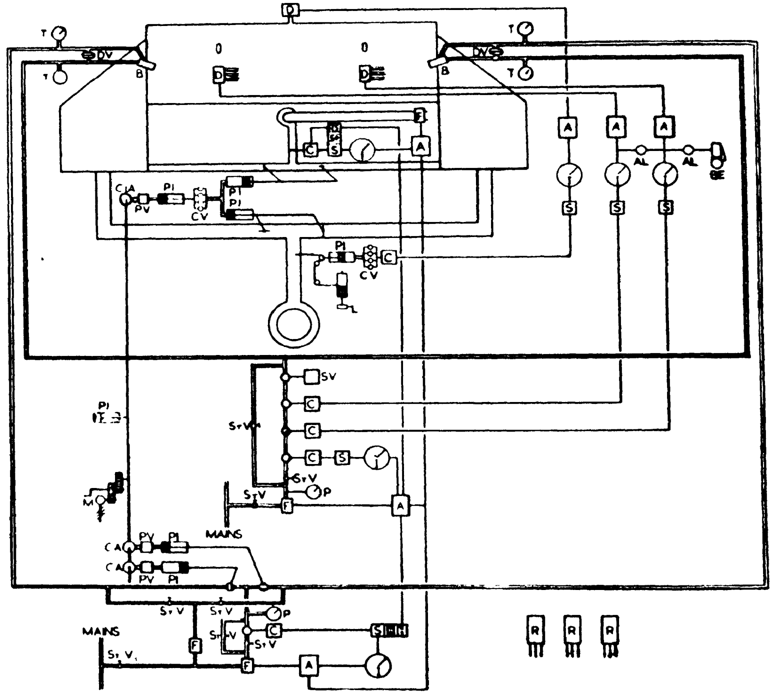


FIG. 182. Fuel/air ratio control.

(By courtesy of Messrs. George Kent.)

A. Amplifier. *AL*. Alarm. *B*. Burner. *BE*. Bell. *C*. Controller. *CA*. Cam. *CV*. Crowe valve. *D*. Detector. *DV*. Diaphragm valve. *F*. Flow meter. *I*. Indicator. *M*. Motor. *P*. Pressure gauge. *PI*. Piston. *PV*. Pilot valve. *R*. Recorder. *RS*. Ratio setting. *S*. Setting. *SO*. Switch over. *StV*. Stop valve. *SV*. Solenoid valve. *T*. Thermometer.

When employing mixtures of blast-furnace and coke-oven gas, the blast-furnace gas is usually set at a constant value and the heat input varied by adjustment of the coke-oven gas. The insertion of a regulator enables any adjustment of the coke-oven gas to automatically regulate the supply of combustion air, thereby maintaining a constant fuel/air ratio.

The efficiency of the combustion control should be checked by means of instruments indicating the analysis of the flue gases. For this purpose CO_2 and $CO + H$ recorders are frequently used. Even without flow-meters these flue-gas indicators can greatly improve the efficiency of combustion. With very little training the furnace operators will quickly use these instruments with intelligence, thereby assisting them to regulate the fuel and air flow to obtain or maintain set conditions of combustion.

The flow-meters normally employed depend on the differential pressure which occurs when the flow is passed through an orifice plate, venturi tube or similar device. If a suitable orifice is inserted in a pipe line a drop in pressure results when the flow has passed through this orifice. The pressure differential can be measured by a U-tube, one leg of which is inserted on either side of the orifice plate. From this differential pressure, that is, the difference of the levels of the two limbs, the rate of flow can be calculated from the expression quoted by Engel and Walton:[2]

$$Q = \frac{C}{1 - m^2} A_2\sqrt{\frac{2g(p_1 - p_2)}{p}}$$

in which

Q is the rate of flow,

C is the discharge coefficient,

m is the orifice ratio, or the area of the orifice opening divided by the area of the pipe,

$\frac{1}{1 - m^2}$ is the velocity of approach factor,

A_2 is the area of the orifice opening,

$p_1 - p_2$ is the pressure difference across the orifice,

p is the density,

g is the acceleration of gravity.

In practice two different arrangements of tapping are found. In some cases the differential manometer is connected to a tapping adjacent to both faces of the orifice plate. In other cases the upstream or high pressure tapping is about one diameter of the pipe upstream, whilst the low pressure or downstream tapping is about 0·5 diameters downstream. The pressure differences in the vicinity of the orifice plate are shown diagrammatically in Fig. 183 taken from the paper by Engel and Walton.[2]

A practical application of this principle of measuring flow is shown diagrammatically in Fig. 184, which shows the basic operation of a Bristol Recording Flow Meter. Any change in the rate of flow across the orifice results in a change in the differential pressure. If the rate of flow is increased the differential pressure increases. These changes can be expressed mathematically in the modified equation:

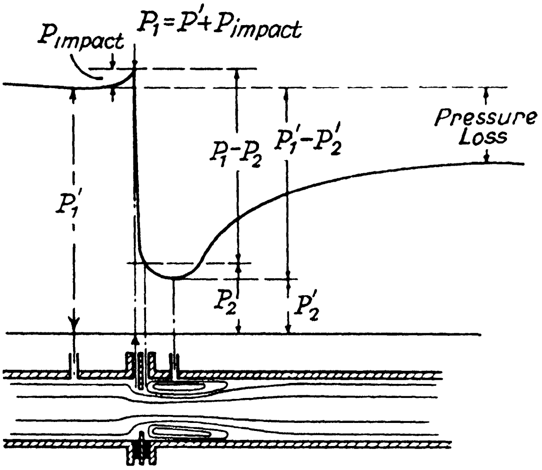


FIG. 183. Flow metering by differential pressure method.

$$Q = C\sqrt{hP}$$

where Q is the cu. ft. of gas per hr. referred to a base or contract pressure and temperature,

C is the hourly orifice coefficient,

h is the differential pressure in inches of water,

P is the pipe line pressure in lb. per sq. in. absolute.

Since liquids are practically incompressible the liquid flow can be calculated from the simplified expression:

$$Q = C\sqrt{h}$$

where Q is the quantity of liquid flowing in cu. ft.,
C is the hourly orifice coefficient,
h is the differential pressure in inches of water.

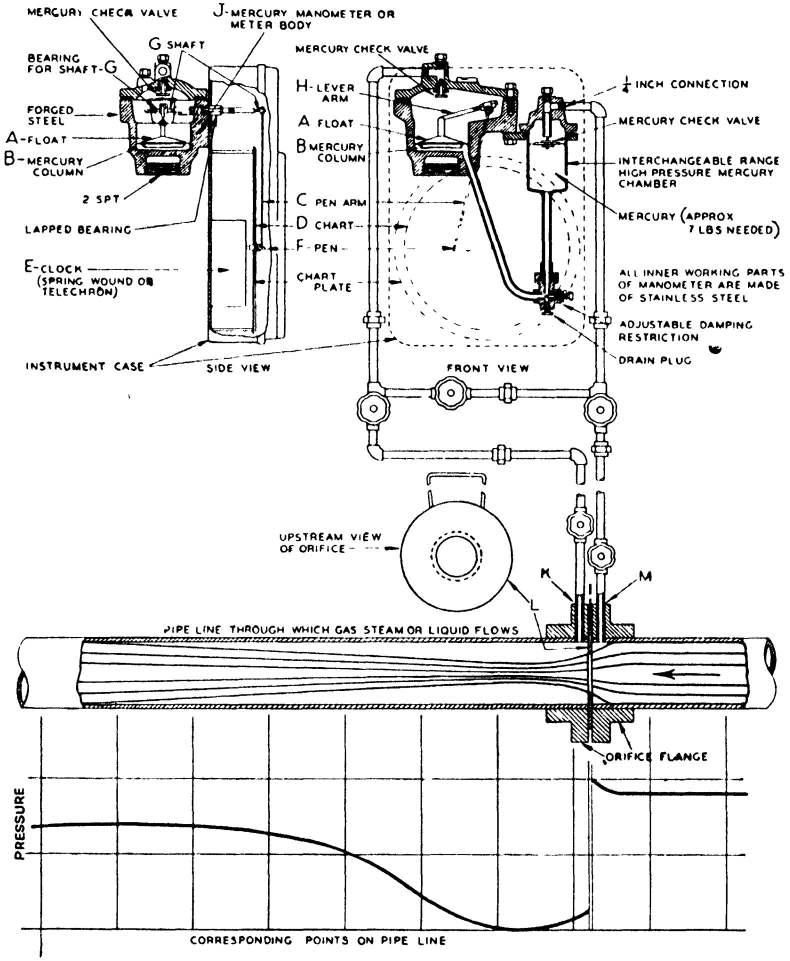


FIG. 184. Bristol recording flow meter.
(*By courtesy of the Bristol Instrument Co. Ltd.*)

These equations show that, other factors being constant, the rate of flow varies with the square root of the differential pressure across the orifice. It is, therefore, possible for an instrument measuring this differential pressure to be calibrated in terms of flow and employed for recording or measuring the flow of fluid through the pipe.

The registering instrument used in connection with these differential flow-meters may take several forms. The simplest is the manometer consisting of a glass tube bent into the shape of a U, as shown in Fig. 185. A ring-balance type of instrument, as shown in Fig. 186, may be employed. Another method adopted by Electroflo is shown in Fig. 187.

The maximum drop in pressure, or the "vena-contracta", occurs at a point downstream which depends on the area ratio of the orifice plate and the main, and for preference the low pressure limb should be inserted at this point. If the diameter of the pipe line or main is five times the diameter of the orifice plate, the contraction of the stream, and therefore its velocity, is greatest at a position downstream about 0·4 times the diameter of the pipe line. This factor is important when designing gas mains in which it is intended to instal flow-meters. There should be sufficient length of straight main on either side of the orifice plate to allow reliable readings to be obtained.

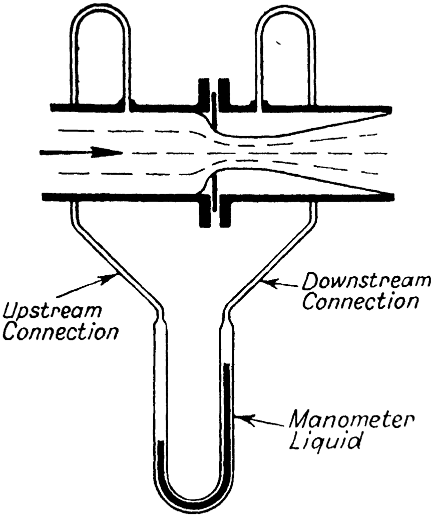


FIG. 185. U-tube manometer.

When metering hot raw producer-gas the orifice plate may be built of firebrick or may take the form of a venturi tube constructed of refractory material. In the metering of unclean gas care must be taken to provide facilities for removing any deposits of soot, etc., which may accumulate. Although cleaning devices are generally installed, at several plants with which the author is acquainted little trouble has been experienced, and these devices have seldom been used. Sometimes the venturi tube is placed in a vertical portion of the main, which reduces the trouble with deposition. Nevertheless, it is usual to provide compressed

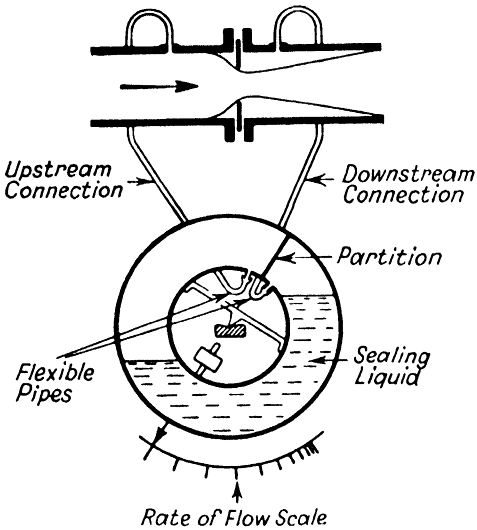


FIG. 186. Ring balance instrument.
(*From "Efficient Use of Fuel" by courtesy of H M Stationer Office*)

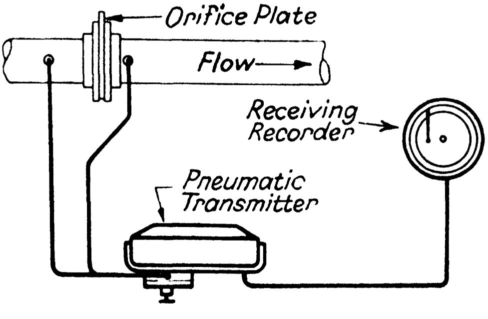


FIG. 187. Electroflo flow meter installation.
(*By courtesy of Electroflo Meters, Ltd.*)

air or steam jets to assist the removal of any deposits. Much useful information on this subject has recently been given in the *Instrument Manual*[3] 1949, whilst the general principles and application of metering raw producer-gas has been discussed in detail by Dall and Pluck.[4] The design of the main and the correct proportions of the venturi tube appears to be of supreme importance. The device used by Dall and Pluck for metering raw producer-gas is shown in Fig. 188.

When dealing with oil fuel it is usual to employ thick plates with a shaped inlet, which has the property of giving a constant coefficient down to considerably lower rates of flow than would normally be obtained with the standard orifice plate, venturi tube or other differential pressure generating device.³

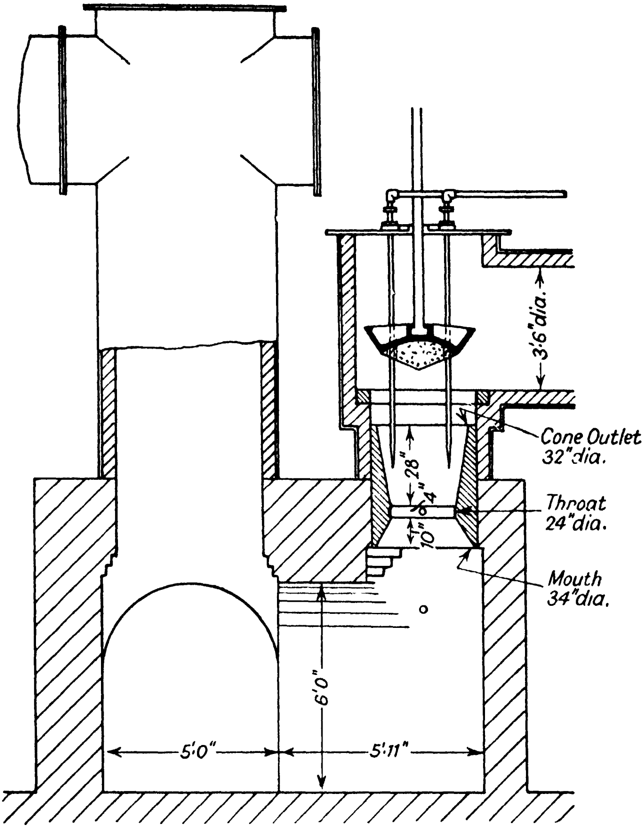


FIG. 188. Metering raw producer gas. (Pluck.)
(By courtesy of the Iron and Steel Institute).

Although many successful installations for metering oil fuel based on the differential pressure method have been installed, there is a general feeling in some quarters that a positive displacement meter gives a more reliable result. Even when the temperature and viscosity are maintained fairly constant, the differential pressure across the orifice of an oil line does not follow strictly the square function law. This may be due to the fact that oil flow usually shows "Reynolds numbers" in a range where the coefficient of flow is not constant, but is actually a function of flow. In America there appears to be a tendency to adopt positive displacement meters for the measurement of oil fuel in preference to the differential type of meters. Several meters depending on the orifice plate principle have been employed with satisfactory results on oil-fired open-hearth furnaces in this country. Their success, however, depends on the careful regulation of the oil temperatures and/or viscosity.

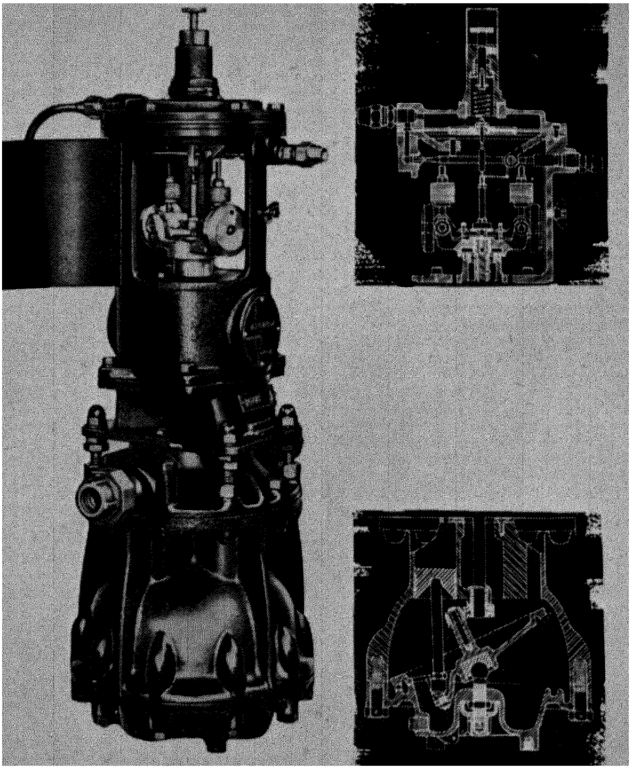


FIG. 189. Askania transometer.
(By courtesy of the Askania Regulator Co.

In America, where fuel oil has been extensively applied to open-hearth furnaces, several types of positive displacement meters have been developed. A typical positive displacement meter, the Askania

Transometer, is shown in Fig. 189. In this type of meter the cylinder may be regarded as a measuring cup and the speed of rotation is a function of the rate of oil flow. This speed of rotation can be employed to record or measure the volume of oil flow.

The instruments employed for checking the *composition of the flue gases* fall into two classes:

1. Those instruments based on chemical absorption.
2. Inferential methods, in which some variation in the physical properties is related to some change in the composition of the gases.

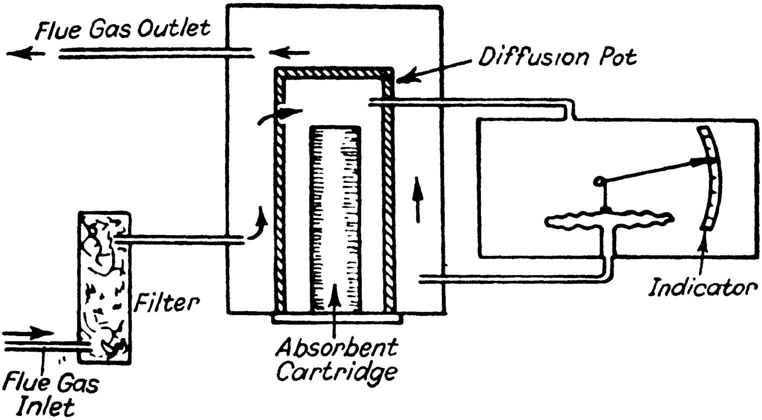


FIG. 190. Diffusion type CO_2 indicator.
(From "Efficient Use of Fuel" by courtesy of H. M. Stationery Office.)

The general principles of the chemical absorption and the electrical thermal conductivity meters have been discussed in Volume 1. Several meters based on these basic principles are available, although their construction varies slightly according to the make of the particular instrument.

A very simple instrument based on diffusion is shown in Fig. 190. In this meter a solid absorbent for CO_2 is placed in a porous pot, which is enclosed in a chamber. A steady stream of the gases is passed through a filter into the lower part of the chamber. Some of the flue gases within the chamber diffuse into the porous pot, where the CO_2 is absorbed causing a drop in pressure inside the porous pot. The pressure differential outside and inside the pot can be related to the amount of CO_2 absorbed. This reduction in pressure is proportional to the amount of CO_2 in the flue gases and can be recorded on a suitable gauge. This type of instrument requires little maintenance. The cartridges of absorbents, when in continuous use, require replacement about every twenty-four hours. Portable instruments of this type are convenient and compact, and are useful for spot readings and checking.

Whichever type of instrument is employed care must be taken to avoid the infiltration of air. Since the gases which are sucked into the instrument contain a certain amount of dust, soot and tarry matter, they must be filtered before entering the instrument. This filter frequently takes the form of a fireclay or carborundum pot near the entrance to the sampling tube. The filter should be sufficiently large that even when it is partly coated on the outside with dust or soot an ample supply of gas is allowed to pass into the measuring chamber. It is desirable to have another filter at a cooler part of the system so as to remove any tar which may condense. This second filter normally consists of glass or cotton-wool or asbestos.

The size of the sampling tube and the length of the leads are very important. These factors affect the time-lag. A large tube, or very long leads, results in a considerable volume of gases having to be removed before a new sample is analysed.

The introduction of *furnace pressure control* has resulted in a considerable improvement in furnace efficiency. A deficient stack draught, which results in a high furnace pressure, causes the flame to burn the arch plates and refractories. This condition results either in a lowering of the furnace output or an increased repair and maintenance cost. Buell[5] stresses the importance which is attached to this particular control in America, where it is regarded as the most important feature of open-hearth furnace operation.

Fisher[6] demonstrates the advantages of minimizing the air infiltration by maintaining the highest internal pressure in the furnace consistent with the least damage to the roof and refractory materials. The furnace he discusses was fired with Bunker C fuel oil, using steam for atomization. He shows that a 50 per cent. factor of infiltration, with air at 60 deg. F., the preheated air temperature is lowered from 2,100 deg. F. to 1,045 deg. F. which decreases the flame temperature from 4,400 to 4,100 deg. F.

Prevention of the infiltration of air can be attained by maintaining atmospheric pressure at the bottom of the charging doors. If the pressure at the bottom of the charging doors is to be atmospheric, a pressure of 0·04 to 0·05 in. water gauge may have to be maintained immediately underneath the roof, as shown by Smithson.[7] This figure, however, has to be varied with the height of the furnace, owing to the stack effect which causes the internal pressure to increase about 0·013 in. w.g. for each foot of height. On some of the larger furnaces, therefore, a pressure of 0·08 to 0·09 in. w.g. is maintained at the roof.

In order to obtain accuracy two impulse lines are employed. If only one impulse line is provided a false reading is given due to the fact that the external pressure immediately above the roof is affected by temperature conditions and is not the same as the atmospheric pressure at the position where the regulator is located. This error is overcome by the provision of a compensating line, which gives a true time pressure differential inside and outside the roof. These considerations are explained diagrammatically in Fig. 191.

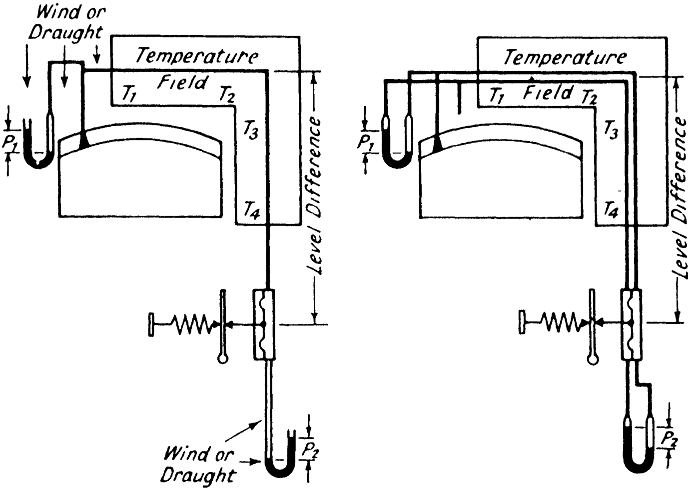


FIG. 191. Furnace pressure control diagram.

The regulator described by Smithson[7] consists of:

(1) A super-sensitive diaphragm of low inertia, which reacts to pressure changes of the order of 0·0005 in. w.g.

(2) An hydraulic power unit, which supplies high-pressure oil to a power cylinder capable of lifting dampers of considerable weight.

The general arrangement is shown in Fig. 192, whilst Fig. 193 gives the general principles of a pressure gauge.

When the furnace is provided with a waste-heat boiler the impulse from the instrument can be employed to regulate the speed of the waste-heat boiler fan, but owing to the frequent oscillations which occur within the furnace, many designers are not in favour of this system owing to the wear on the electric gearing. If the furnace pressure is to be regulated within narrow limits, which is essential, a large number of contacts are necessary. The frequent oscillations taking place necessitate constant movement and wear on the contactors. If this difficulty can be overcome, however, it would appear to be the best method of furnace pressure control. One important demerit is the fact that whenever the boiler is out of commission the pressure must be manually controlled, which is not impossible, if difficult.

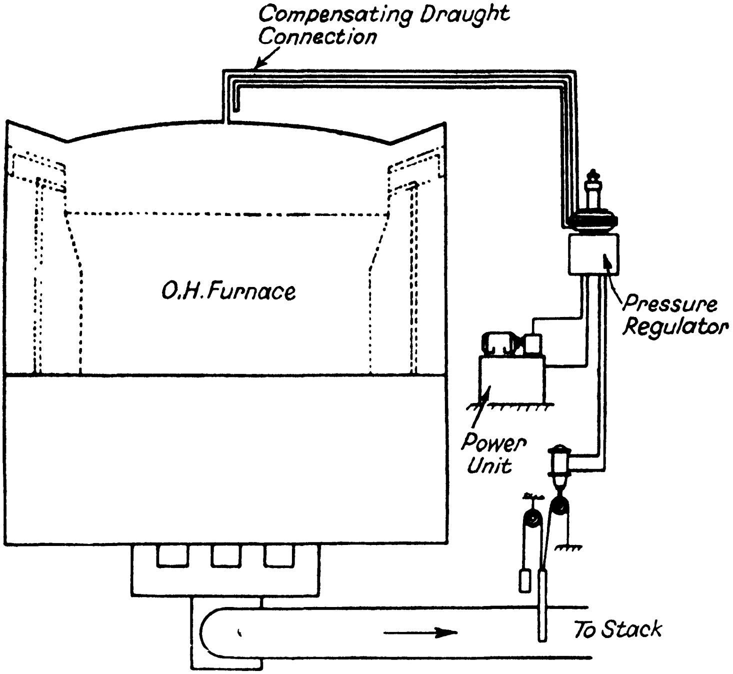


FIG. 192. Furnace pressure control—general arrangement.

Another method has been suggested which consists of regulating the fan speed by means of a relay from the damper position. In this system the furnace pressure is actually controlled by the damper, but the position of the damper, by means of a multi-position control device, regulates the fan speed. When the boiler is temporarily out of commission this method still allows the automatic control by the damper, but hunting may result. Any decrease in the furnace pressure automatically lowers the damper and at certain positions this action would reduce the speed of the fan. This reduction of the fan speed would increase the furnace pressure with the resultant movement of the damper. Since the damper is already at a critical position, this action would again affect the fan speed. It is possible that some compromise between stability and sensitivity and careful timing of the speed control may render this system a practical proposition.

The importance of maintaining a balance in the temperature of both sets of regenerators recommends the adoption of *reversal control*. Frequently owing to carelessness or pre-occupation the reversal periods are irregular, resulting in the furnace being thrown out of balance. Some method of reversing the flow of air and fuel on a *definite temperature differential* is therefore a decided advantage.

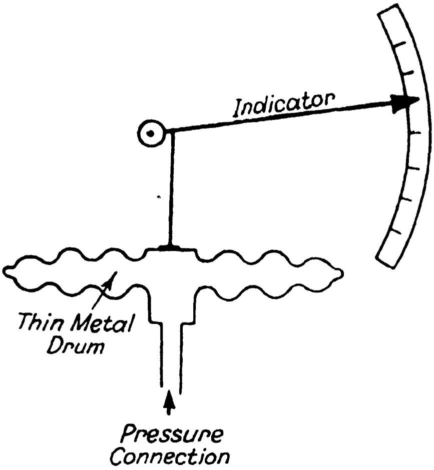


FIG. 193. General principles of furnace pressure control gauge.
(From "Efficient Use of Fuel" by courtesy of H. M. Stationery Office.)

Early attempts to achieve this aim were marred by the instruments available and the methods of application. These efforts to control regenerator temperatures employed optical pyrometers sighted on some part of the top wall of the regenerator chamber or the slag pocket wall. The readings obtained were frequently affected by dust and soot deposits which periodically collected on the lens of the pyrometer. Even when a stream of compressed air was caused to blow across the lens to remove the dust, the results obtained were not too satisfactory.

Modern practice favours the use of base metal thermocouples in the flues adjacent to the regenerators, or in the base of the regenerators themselves. This method has the following advantages:

(1) The temperature prevailing at this point seldom exceeds 700 to 800 deg. C., whereby robust base metal thermocouples can be employed.

(2) Base metal thermocouples develop a high e.m.f. which enables the temperature to be registered with great accuracy.

(3) The temperature–e.m.f. relationship of base metal thermocouples in linear, therefore, the temperature differential readings are a true temperature difference.

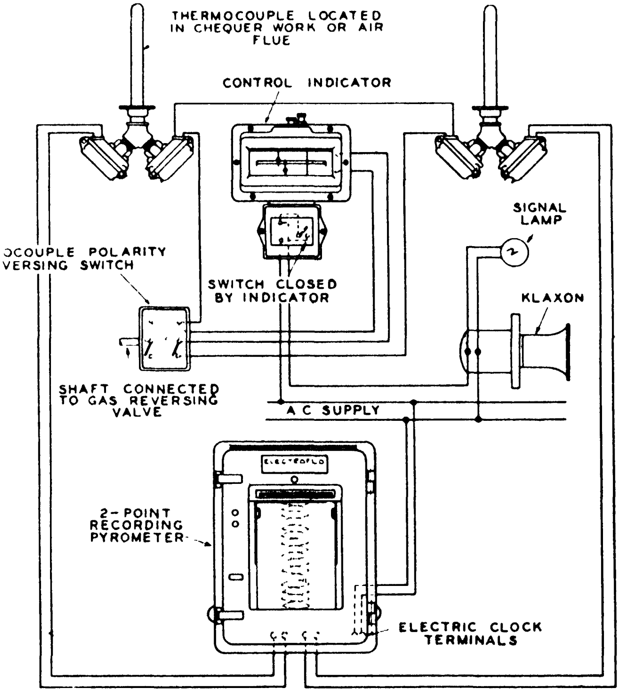


FIG. 194. Reversal control—Syren system.
(*By courtesy of Electroflo Meters Ltd*)

(4) At these points a true average temperature condition prevails owing to the fact that the heat from bath reactions, etc., has been dissipated.

(5) The temperature fluctuations are greater at these points than at any other part of the furnace system.

It is, therefore, possible to achieve furnace reversal on a true differential basis provided reliable instruments are selected. This reversal may be automatic or manually performed. In some plants automatic reversal is favoured, the chief disadvantage being the possible interruption of metallurgical processes. In any case the instruments can be switched from automatic control to manual control or vice versa as required.

Sometimes the instruments are connected to a system of warning lights or sirens, leaving the actual reversal to be performed manually as shown in Fig. 194.

Larsen and Shenk,[8] whilst pointing out the desirability of automatic control of reversal, indicate the difficulties involved and the differences of opinion which exist among steelmakers concerning this matter. This difference of opinion is largely due to the problems associated with all automatic control devices or methods in the open-hearth shop. The dirt, vibration and accidental shock render conditions of service severe and necessitate robust and reliable instruments which are as free as possible from maintenance troubles. Another difficulty is presented by the fact that metallurgical processes cannot be regulated with clockwise precision.

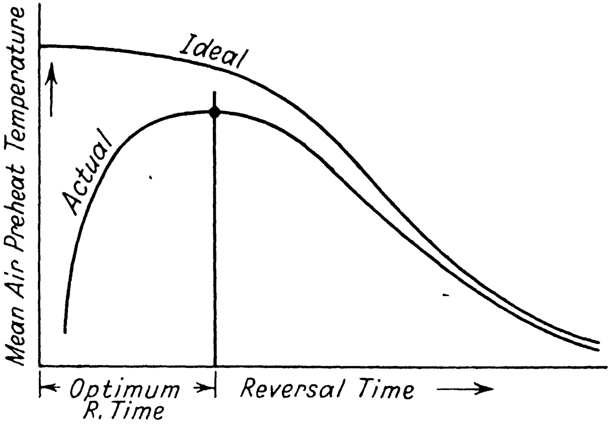


FIG. 195. Relationship between reversal interval and heat exchange by regeneration. (Larsen and Shenk.)

They[8] produce an interesting chart which is reproduced in Fig. 195, showing the relationship between the reversal interval and the efficiency of heat exchange by regeneration. This chart indicates that there is an optimum reversal period for each furnace, which must be determined and standardized. The author has found that this reversal period varies with the phase of the process. During charging and melting the period may be about 15 minutes, falling to 10 minutes or less during the refining stage. There do appear to be factors which recommend reversal on a time basis.

Rogers[9] also suggests that there is little justification for reversal on a temperature differential between the ingoing and outgoing checkers, especially when the air-checker "temperatures" are measured by total radiation pyrometers sighted directly on the checker brickwork. He points out that waste gases, high in CO and H_2O, would seriously affect the recorded temperature.

Larsen and Shenk, however, suggest that with a photo-cell unit, checked by optical pyrometers, only a small error from the true temperature of the brick surface is obtained under normal furnace conditions.

Even when not employed for reversal purposes, the use of pyrometers for determining regenerator temperatures is an advantage. A knowledge and record of any variation in these temperatures does indicate any excessive infiltration of air, collapse of brickwork or any other abnormal condition.

Many claims have been made during recent years for the increase output due to the introduction of *roof temperature control.* The temperature at which the roof of an open-hearth furnace begins to soften to a great extent controls the rate of heat input and thereby the output of the furnace. Increased output and reduced refractory costs will result from any reliable method of operating a furnace as near as possible to the safe maximum roof temperature without actually softening the roof. At one firm where roof temperature control has been introduced, they claim an increased output of 10 per cent. with a considerable economy in refractory materials.

There are two methods of roof temperature control:

(1) A carborundum block, in which a central hole is drilled to within one inch of the base, is built into the crown of the furnace. The interior face of this block is flush with the inside of the crown. A water-cooled radiation tube, with extension, is sighted on the bottom of this hole in the block and is connected with an indicating and recording pyrometer.

(2) The other method consists of sighting radiation pyrometers on to some point in the roof. These pyrometers pass through holes in the back wall of the furnace and the lenses are kept clean and cool by means of a current of compressed air blowing across them. Normally three pyrometers are employed, one being sighted on the centre of the roof and the other two at a point on each end of the furnace equidistant between the centre and the furnace ports.

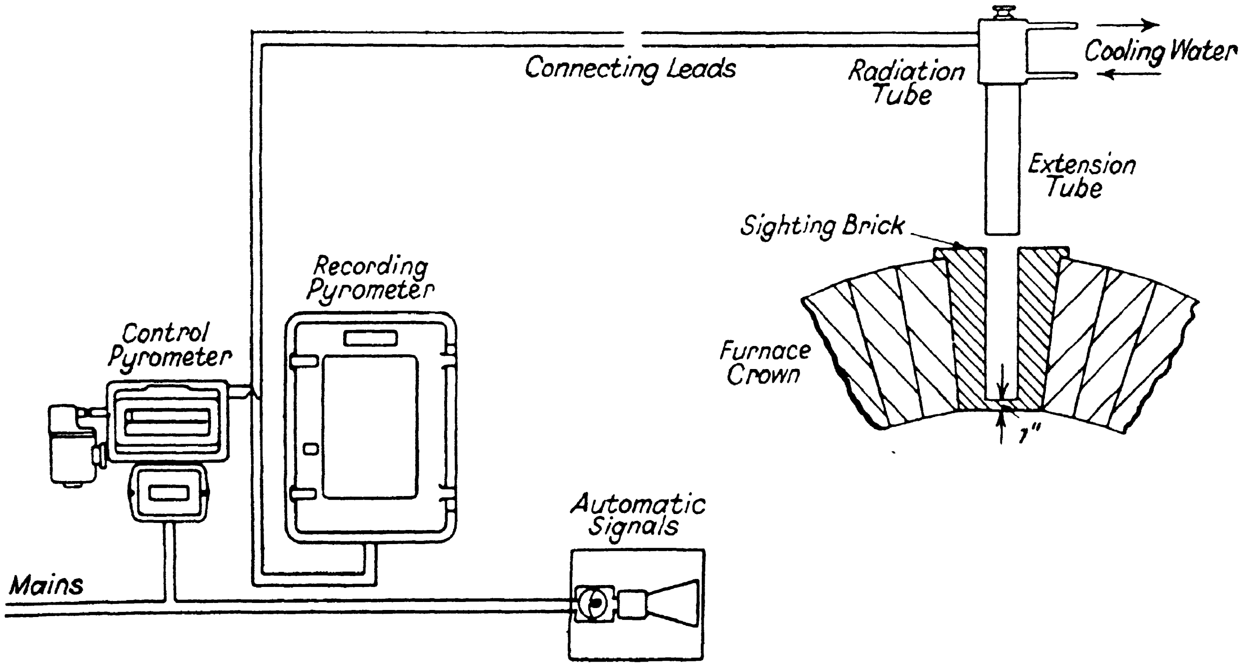


FIG. 196. Roof temperature control.

Whichever method is employed the actual control of the roof temperature may be achieved either automatically or by manual control of the fuel flow. If automatic control is adopted the impulse from the pyrometers actually controls the fuel flow, which in turn regulates the air flow to maintain the desired fuel/air ratio. Many successful applications of this type have been adopted on oil-fired furnaces in America, and it has also been applied to gas-fired furnaces with equal success.

Some operators prefer to connect the instruments to warning lights or sirens, which come into action when the maximum roof temperature permissible is being approached. This method allows manual adjustment of the fuel flow, and appears to possess advantages of simplicity.

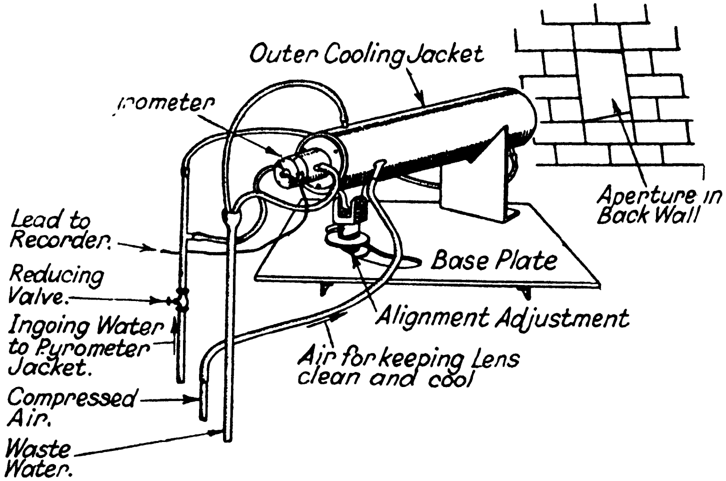


FIG. 197. Roof temperature control.

In both methods the roof temperature recorded is actually the temperature at one, two or three specific points. A roof is seldom worn evenly over its whole surface. The more general failure of the roof is due to some local damage at one or more points. It is, therefore, possible even when three pyrometers are employed for the roof to be above its maximum safe temperature at certain places, whilst the particular points on which pyrometers are sighted are perfectly safe. This condition appears to present a serious danger, which may nullify any advantages to be gained from this method of control.

The measurement of furnace temperatures generally depends on the use of optical or total radiation pyrometers. *Optical pyrometers* measure the radiation proceeding from a particular point in red filtered light.

There are three chief designs of this type of pyrometer:

(1) The hot point can be viewed through a telescope with a red filter interposed in the eye-piece. This beam is focused at a point inside the telescope where there is a small electric lamp filament. The external circuit of the lamp contains an ammeter or milliammeter and an adjustable resistance arranged in plain circuit or bridge circuit form. When this instrument is focused on a hot-spot the lamp filament will be seen against a red-hot background. The resistance is adjusted until the tip of the filament just disappears in the background. The temperature is recorded by the ammeter, or on a direct reading temperature scale replacing the ammeter. This type of instrument is frequently referred to as the "disappearing filament" pyrometer. It is standardized against a "black-body" furnace or standard instrument and a scale of corrections is determined. Any instrument of this type standardized against black-body conditions can be corrected by known information regarding emissivities for uses under non-black-body conditions.

(2) The current in the lamp filament may be kept constant and caused to illuminate a definite field of view, whilst the light from the furnace passes through natural-tinted absorbing wedges and illuminates an adjacent field of view. The absorbing wedges are adjustable so that the second field of view can be altered until it has the same intensity as the first field due to the filament lamp. The position of the wedges are related to a scale which gives a direct temperature reading. The method of standardizing this instrument is the same as that described in the first example. Frequently an amyl acetate lamp is employed to provide the standard flame.

(3) The light from a constant source, either an electric or amyl acetate lamp, and the light from the furnace can be passed through polarizing prisms. These prisms polarize the light from the two sources in planes at right angles to each other and these beams illuminate adjacent fields. Both these beams pass through a Nicol prism by the rotation of which the strength of one beam is increased and that of the other decreased. By rotating the Nicol prism a position may be attained where the intensity in the two fields is the same. The actual position of the Nicol prism can then be used to record the temperature. The standardization is performed as before on a black-body furnace or against standard instruments.

All three instruments are standardized for black-body conditions and require correction for varying emissivities. A scale correction is usually supplied with each instrument. It must also be remembered that none of the light proceeding from the source of the heat, or the hot spot, should be absorbed en route by smoke, fume or gases. When these limitations and conditions of service are appreciated this type of pyrometer can be successfully applied in industrial process control.

In the case of the *total radiation pyrometer* both the heat and light rays emitted from a particular source are focused on to a small, thin, blackened metal receiving disc. They may either be focused by a concave mirror at the rear of the instrument or by a convex lens at the front of the barrel. A thermo-junction of very fine wire is attached to the rear of the receiving disc and the cold junction is situated at a point which is screened from radiation. The angular dimensions of the cone of radiation are limited by passing the beam through a suitable aperture.

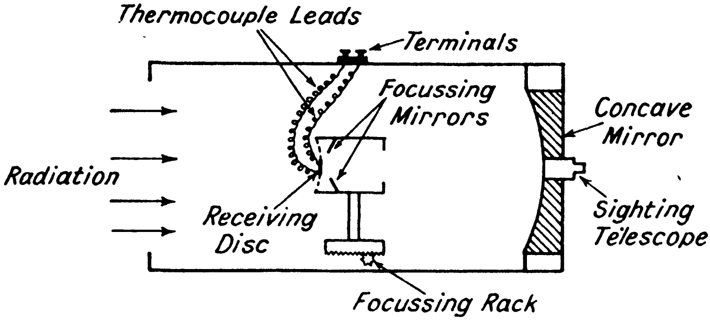


FIG. 198. Type of total radiation pyrometer.
(From "Efficient Use of Fuel" by courtesy of H. M. Stationery Office.

There are three main types of this instrument, which are shown in Figs. 198, 199 and 200. In Fig. 198 a concave mirror is employed to focus the beam on the receiving disc. The aperture which is between the mirror and the receiving disc, is at a fixed distance from the disc. The receiving disc and aperture assembly is movable and possesses two small mirrors, which reflect the image formed on the disc so that it can be seen through the sighting telescope in the centre of the concave mirror. The instrument must be focused to suit the distance from the furnace and this is done by moving the receiving disc and aperture assembly as required. In the process of focusing it is important to see that the image of the hot body completely overlaps the disc on all sides. The terminals of the instrument which are internally connected with the cold junction, are connected externally to a millivoltmeter which can be calibrated to give direct temperature readings.

The pyrometer shown in Fig. 199 operates on a similar principle, but the barrel is longer and the aperture and the mirror are placed at the opposite ends of this barrel. The mirror is smaller and more concave, whilst the aperture is larger than in the instrument shown in Fig. 198. The receiving disc is situated at the principal focus of the mirror. The instrument does not require to be focused and can be used without error over a fairly wide range of distances from the furnace. The area of the furnace viewed, however, increases with the distance from the object and the temperature recorded is the mean temperature of that area. Since the amount of radiation is proportional to the fourth power of the temperature variation, any wide temperature differences over that area may give a misleading mean temperature figure. The size of the area necessary is a function of the distance from the object, but that data is usually quoted by the makers of the instrument.

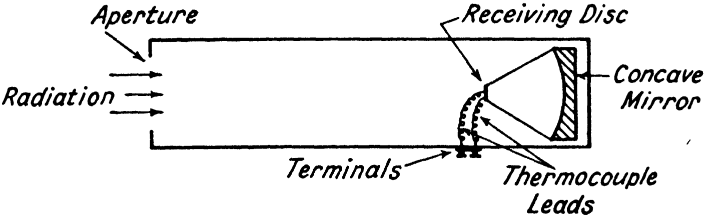


FIG. 199. Type of total radiation pyrometer.
(*From "Efficient Use of Fuel" by courtesy of H. M. Stationery Office.*)

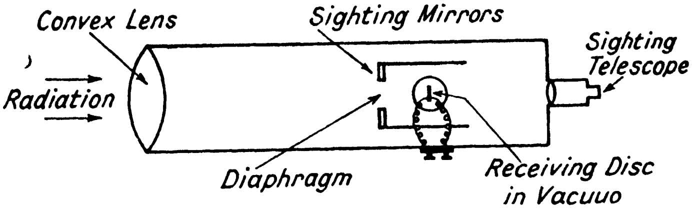


FIG. 200. Type of total radiation pyrometer.
(*From "Efficient Use of Fuel" by courtesy of H. M. Stationery Office.*)

Fig. 200 shows another type of total radiation pyrometer in which the mirror at the rear is replaced by a convex lens at the front of the instrument. The instrument is of the fixed focus type, but a sighting telescope is provided at the rear to ensure that the object is correctly focused on the receiving disc.

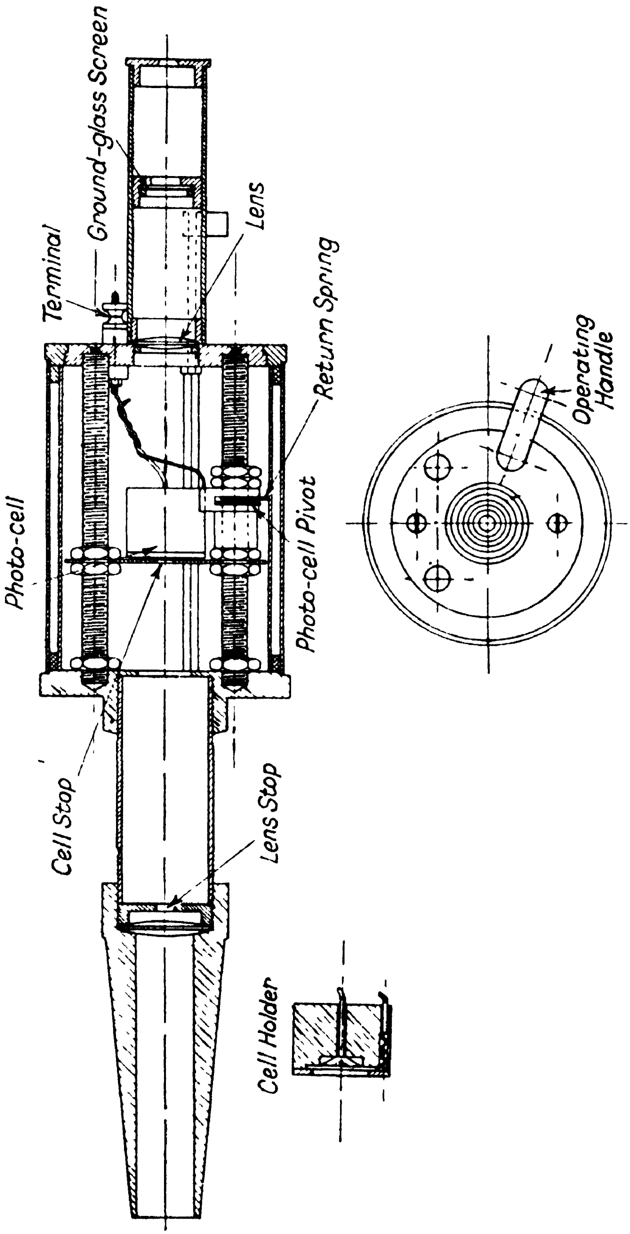


FIG. 201. Land's photoelectric pyrometer—construction.

By courtesy of the Iron and Steel Institute

These total radiation pyrometers are standardized on black-body furnaces or standard instruments, and the sources of error are similar to those which apply in the case of the optical pyrometers. Since infra-red rays convey much of the total energy, the presence of CO_2 and water vapour, which strongly absorb these rays, seriously affects the readings obtained. This type of instrument is not satisfactory for measuring flame temperatures.

Total radiation pyrometers can be used, however, to activate automatic recording and regulating mechanism, which is not possible with the disappearing-filament type of instrument.

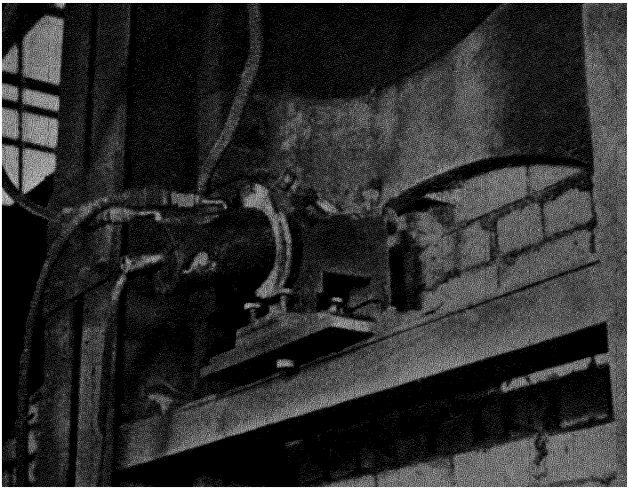


FIG. 202. General view of Land's pyrometer mounted on back wall above tap-hole.

(*By courtesy of the Iron and Steel Institute.*)

As stated by Rogers[9] the total radiation pyrometer is subject to some inherent errors which are quite serious from the scientist's point of view, nevertheless, no manager should be deterred from using one for this reason. He must, however, appreciate the nature of these errors and the precautions to be taken.

An interesting description of a photoelectric pyrometer, employing a selenium barrier-layer photoelectric cell, has been given by Land.[10] This pyrometer, which is shown in Figs. 201 and 202, has been developed for the measurement of open-hearth furnace roof temperatures. He

TABLE LV

DETAILS OF VARIOUS TYPES OF PYROMETERS (*Land*[10])

Type of Pyrometer	Wavelength Range	Effect of Gas Absorption by CO_2 and H_2O	Effect of Smoke and Dirt (Error at 1700° C. for 10% loss of radiation	Stability	Speed of Response	Comments
1. Total radiation mirror focusing.	All wavelengths.	Serious	50° C.	Can be Good.	Can be fairly good, not always so in commercial instruments.	Effect of reflected flame radiation is small and calculable, but instrument covers CO_2 and H_2O absorption bands, which may cause serious errors.
2. Partial radiation, mirror focusing, with glass absorption.	Visible light and near infra-red down to about 2·5μ.	Slight.	30° C.	Can be Good.	As 1.	A possible instrument; not too good in effect of smoke and dirt; speed of response may be poor in some types.
3. Partial radiation, lens focusing.	Visible light and near infra-red down to about 2·5μ.	Slight.	30° C.	Can be Good.	As 1.	Lens dispersion can cause serious errors in estimating size of object required; effect of smoke and dirt and comments on speed as 2.
4. Photoelectric, gas-filled cell.	Visible light and/or near infra-red, according to type of cell used.	None with suitable cell.	10–25° C.	Poor.	Good.	Not satisfactorily stable.
5. Photoelectric vacuum cell.	Visible light and/or near infra-red.	None with suitable cell.	10–25° C.	Good.	Good.	Requires amplifier. Could be quite a satisfactory pyrometer, but complicated.
6. Photoelectric barrier-layer cell.	Visible light.	None.	15° C.	Good with selected cells.	Good.	Simple, and accurate, and therefore selected for use.

discusses in detail the construction, installation and operation, together with the errors introduced in the measurement temperatures by the presence of the flame in the furnace. At the time of preparing his paper the pyrometer had been in operation for six months, during which time it had required little maintenance and no measurable change in calibration had occurred. The reasons for adopting this type of instrument are given, and he summarizes the characteristics of the various types of pyrometers, which are reproduced in Table LV.

The *automatic control* of any physical condition or quantity depends on the accurate measurement of that condition or quantity. It is, therefore, obvious that efficient automatic control of production processes is based on a reliable measuring instrument. The actual maintenance of a physical condition or quantity at a desired or predetermined value can be achieved by:

(1) The accurate measurement of that physical condition or quantity.

(2) The comparison of that measurement with the desired value, which may be varied by external means.

(3) The movement of a regulating device by the value obtained by the measuring device (item 1) in such a manner as to restore the condition of quantity to the desired or predetermined value (item 2).

The actual measurement of a condition or quantity can be accomplished with the minimum amount of power, but the control of that condition or quantity requires considerably increased power, therefore, the impulse from the measuring device must be amplified by some method so as to provide sufficient power to affect regulation or control. Any system of automatic control must, therefore, consist of the following basic elements:

(1) A primary measuring or detecting element, which measures the changes in the process which it is intended to control.

(2) A transmitting element, which conveys the impulse from the primary element to the controlling mechanism.

(3) The controlling mechanism, which transforms the impulse into motion, or magnifies impulse movement, ready for control action.

(4) The control mechanism, which finally activates the regulating or control valve, damper, rheostat, etc.

This arrangement is shown diagrammatically in Fig. 203. The actual control may be accomplished in four ways:

(1) On/Off or two-position control.
(2) Three-position control.
(3) Multi-position control.
(4) Gradual control.

The first three methods adjust conditions or quantities in steps, whilst the gradual control restores the balance gradually. It is the gradual control which is usually applied to open-hearth furnaces. Gradual control can be further sub-divided into:

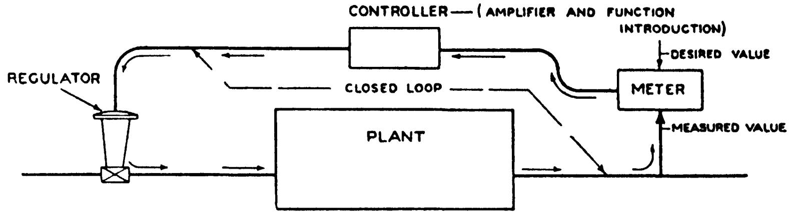


FIG. 203. Prmciples of automatic control.
(By courtesy of Messrs George Kent Ltd)

(1) Proportional control.
(2) Floating control.
(3) Proportional and floating.
(4) Proportional-floating plus re-set.
(5) Proportional re-set rate control.

The actual principles underlying these methods of control are beyond the scope of this work and the reader is referred to standard literature on the subject.[11, 15] The application of these various methods differs slightly in the instruments available, but the general principles of these applications are usually fully described in the pamphlets issued by the instrument makers.[16, 17]

The success of automatic control would appear to depend on the careful compromise between sensitivity and stability. The instrument must be able to detect slight changes from the set condition or quantity, and quickly restore the balance, with reasonable stability or freedom from hunting. Any increase in the sensitivity of the instrument will facilitate its ability to restore the balance quickly, but when the sensitivity is too great there will be lack of stability and a tendency for the apparatus to hunt. This subject is, however, beyond the scope of this work and again the reader must be referred to standard literature on the question of instruments.

As already indicated the type of instruments and the *degree of instrumentation and automatic control* on open-hearth furnaces is still a question of controversy. As Rogers[9] points out an instrument is part of a team and cannot perform its true function alone. Furnace pressure involves such frequent and varied oscillation that manual control cannot be completely satisfactory and automatic control is essential. When furnace pressure control is adopted and high furnace pressure is maintained, careful attention to port design, port maintenance and air/gas mixing is necessary to attain a compact flame, which is well down on the bath causing minimum sting over the doors.

He[9] suggests that no controlled furnace would be complete without the following instruments and control:

(1) Gas main pressure.
(2) Furnace pressure.
(3) Roof temperature.
(4) Air/gas ratio.
(5) Reversal control.

At some plants an orifice plate has been inserted in the stack flue in order to measure *the volume of waste gases*, which is a convenient method of assessing the infiltration of air. This paper, which discusses the variables of the open-hearth process with the aid of a diagram showing their interrelationship, gives a useful indication of the variables which should be measured. He does, however, urge caution in the adoption of rigid instructions until the application of the instruments has been tested and correct conditions determined over an appreciable period of time.

An interesting description of the installation and development of recording and control instruments at a small open-hearth plant is given by Baker.[18] The chief features of the system discussed are the recording of the roof temperature together with the gas and air flows on the same chart. The paper also contains a description of an automatic crown-temperature control.

In presenting the First Report of the Open-Hearth Instruments Sub-Committee, Robertson[19] quotes the instruments advocated and the method in which they should be used. The instruments recommended are:

(1) Automatic pressure control for the gas main.
(2) Gas meter.
(3) Air meter.
(4) Pressure recorder on the top of each gas chamber.
(5) Stack draught gauge.
(6) Furnace pressure recorder.

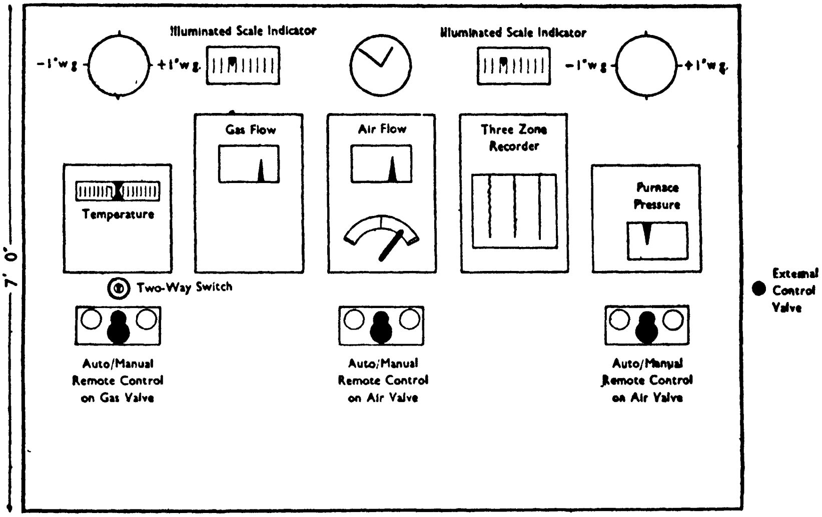


FIG. 204. Furnace control panel (diagrammatical).

(7) Automatic furnace pressure regulator.
(8) CO_2 recorder for fuel gas.
(9) Roof temperature recorder.
(10) Air-checker-top-temperature recorder.
(11) Timing device set to give signal for reversal.

For oil-fired furnaces, the committee recommend the following amendment in addition to items 5, 6, 7, 9, 10 and 11 specified for gas-fired furnaces:

Oil meter (indicating, recording and integrating).
Oil temperature gauges.

Oil pressure gauges.

Air meter.

Steam meter and steam pressure gauge for the steam supply to the oil atomizer.

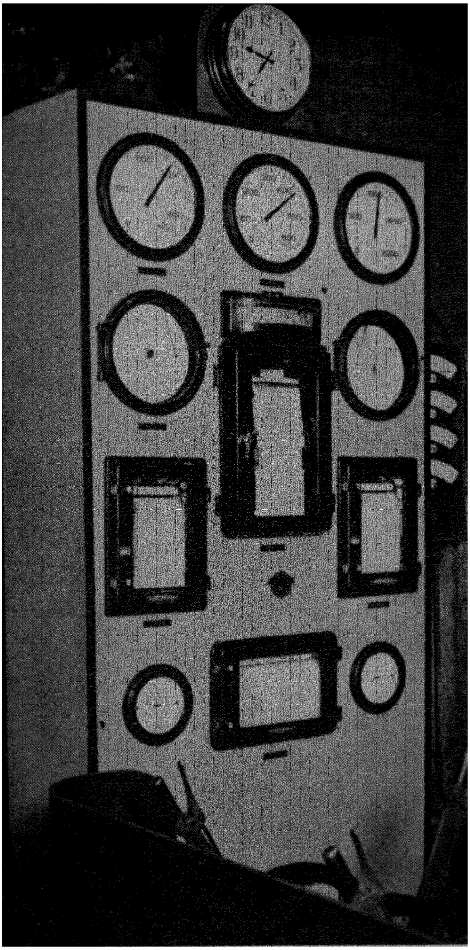


FIG. 205. Furnace control panel.

A more extensive set of instruments were employed by Chester and Thring[20] during their investigation into the influence of port design on open-hearth flames. This report contains much useful information on the subject of instrumentation.

Typical layouts and instrument panels are shown in Figs. 204 to 206, which are designed for gas- and oil-fired furnaces.

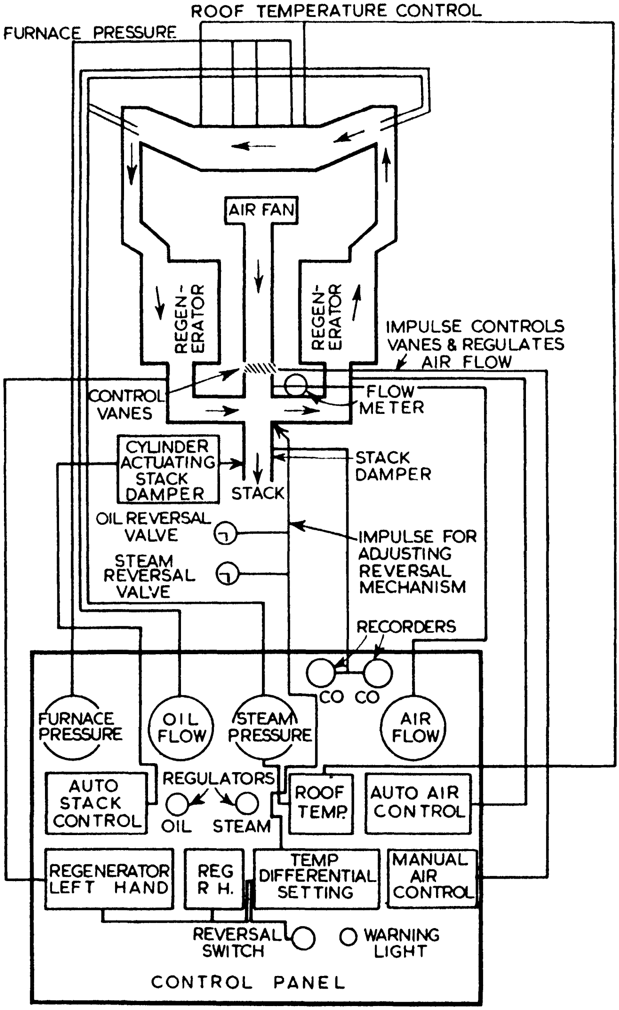


FIG. 206. Layout of oil-fired furnace control.

(*By courtesy of " Metallurgia "*)

Immersion Pyrometry

The ability to measure the temperature of liquid metal with a great degree of accuracy was a noteworthy advance in production control. The method adopted for measuring this temperature by means of the immersion pyrometer is due to the work of Schofield[21] and Schofield and Grace.[22] Their system depends on the use of a platinum-platinum 13 per cent. rhodium thermocouple, protected from the attack of the liquid metal by a thin tube of vitreous silica. The apparatus is designed to permit temperature readings to be obtained in 10 to 20 seconds. Owing to the short immersion period necessary, a lightly insulated tube may be employed for protecting the thermocouple when immersed in the metal.

The progress made in the design and application of this type of pyrometer has been reviewed in several able papers, including those by Land[23] and Manterfield.[24] This progress had justified many plants adopting immersion pyrometers as normal works routine and practice. Its adoption has contributed considerably to the control of quality and avoidance of many casting-pit troubles.

The "quick immersion" pyrometer consists of a platinum-platinum 13 per cent. rhodium thermocouple, lightly sheathed in a silica tube, capable of being immersed in liquid steel covered by slag. The thermocouple arm is formed by a steel tube through which the thermocouple wires pass to the hot junction end. These wires are insulated internally by fireclay insulators or braided glass sleeving. The hot junction end is usually insulated to within about one inch of the end by a twin-bore silica or fireclay insulator, whilst a thin-walled silica sheath protects the tip from the action of the fume, slag and liquid metal. The silica sheath requires replacement after each immersion, whilst the hot junction gives a life of about 15 immersions. Compensating leads proceed from the cold end to an indicator and/or recorder.

For high-frequency furnaces and small ladles, no external insulation of the thermocouple arm is necessary. In the case of small-arc furnaces a similar design is employed, but the end of the arm is protected against the action of the slag. This protection usually consists of a graphite sleeve about 1½ in. in diameter, which is employed on the last 6 in. of the arm.

A trolley type of instrument is employed on large-arc and on open-hearth furnaces. This type of instrument has a longer and heavier arm with a vertical member at the hot junction end. Until recently this vertical member was protected by graphite sleeves, but recently these graphite sleeves have to some extent been replaced by mild-steel cylinders. The main arm is protected for about half its length at the hot end by diatomite sleeves. Normally, about 9 in. immersion is allowed.

The earliest type of arm insulation consisted of blocks or sleeves of diatomaceous material, but the end block was rapidly eroded and was replaced by graphite. This graphite block gave a life of about 200 to 300 immersions, but some trouble was experienced with the embrittlement of the thermocouple wires partly due to the close proximity of the graphite. The employment of mild-steel sleeves instead of graphite removed the trouble due to this embrittlement. Some of these steel sleeves or cylinders have given lives of over 1,000 immersions.[24]

The construction of the pyrometers is ably discussed by Land,[23] who recommends a pyrometer for open-hearth work built up of about 12 ft. of $\frac{3}{8}$-in. gas pipe, bent at the end to allow the thermocouple to be dipped into the steel. The last 6 ft. of this tube is insulated with refractory cement, which should be reinforced by a spiral of 16-gauge iron wire wound loosely around the tube. This wire greatly increases the strength of the insulation and allows it to be patched if it is mechanically damaged at any point. The insulating cement is squeezed on the tube in a plastic state and the best results are obtained when it is applied as dry as possible and is allowed to dry at room temperature for forty-eight hours.

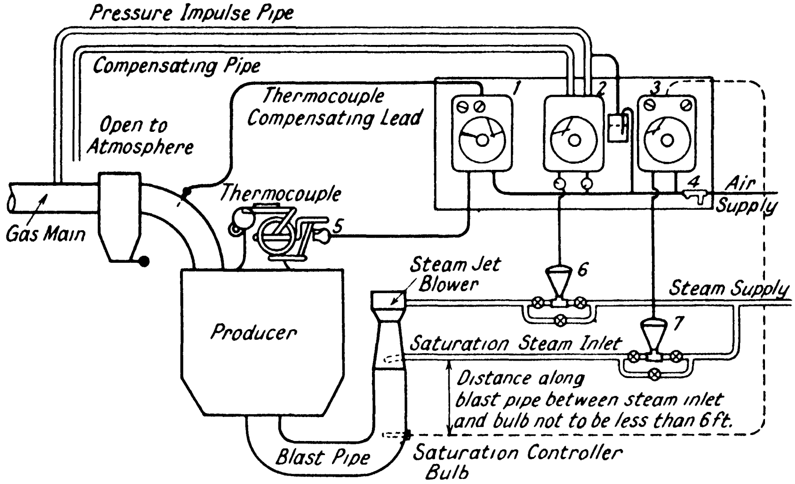
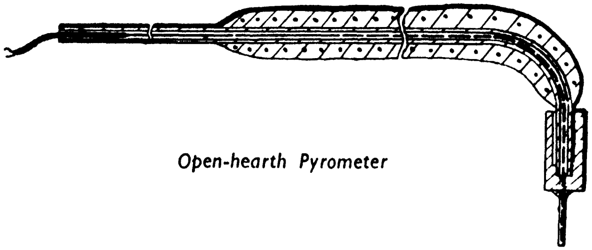
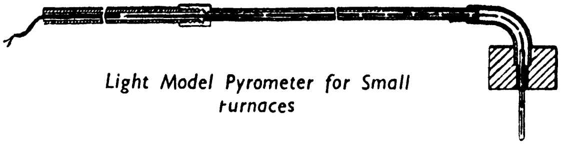
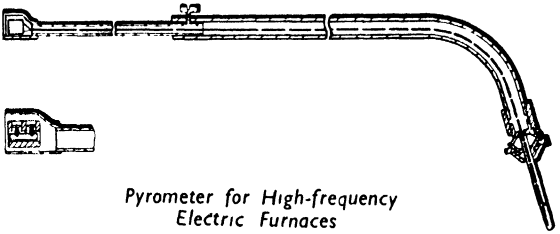


FIG. 207. General principles of gas pressure regulator.
(By courtesy of the Bristol Instrument Co. Ltd.)

The end of the pyrometer which is immersed into the liquid metal is protected by a block of electrode graphite $1\frac{1}{2}$ in. in diameter and about $4\frac{1}{2}$ in. long. This block is drilled and tapped so as to screw into the gas pipe. Care is taken to see that the steel tube lines the graphite block to the bottom, to avoid contamination of the platinum thermocouple. This graphite block is not wetted by either slag or metal, which is a very useful property. A silica sheath, which protects the hot junction, is held in this block with aluminous cement, and projects about 2½ in. from the block. This sheath is about 4¾ in. long and 7 mm. external diameter with a wall thickness of about 1 mm. This sheath must be changed after each immersion.


Open-hearth Pyrometer


Light Model Pyrometer for Small Furnaces


Pyrometer for High-frequency Electric Furnaces

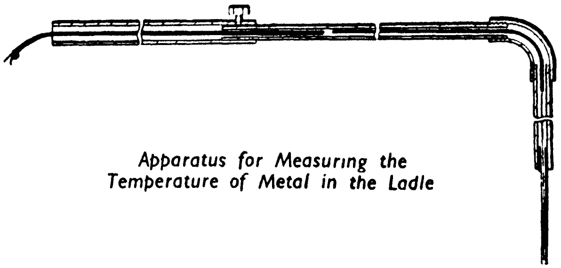

Apparatus for Measuring the Temperature of Metal in the Ladle

FIG. 208. Types of immersion pyrometers.

The platinum-platinum rhodium thermocouple, which is about 0·5 mm. dia., extends from the hot junction to a point well inside the cold portion of the pyrometer, where it is soldered to the compensating

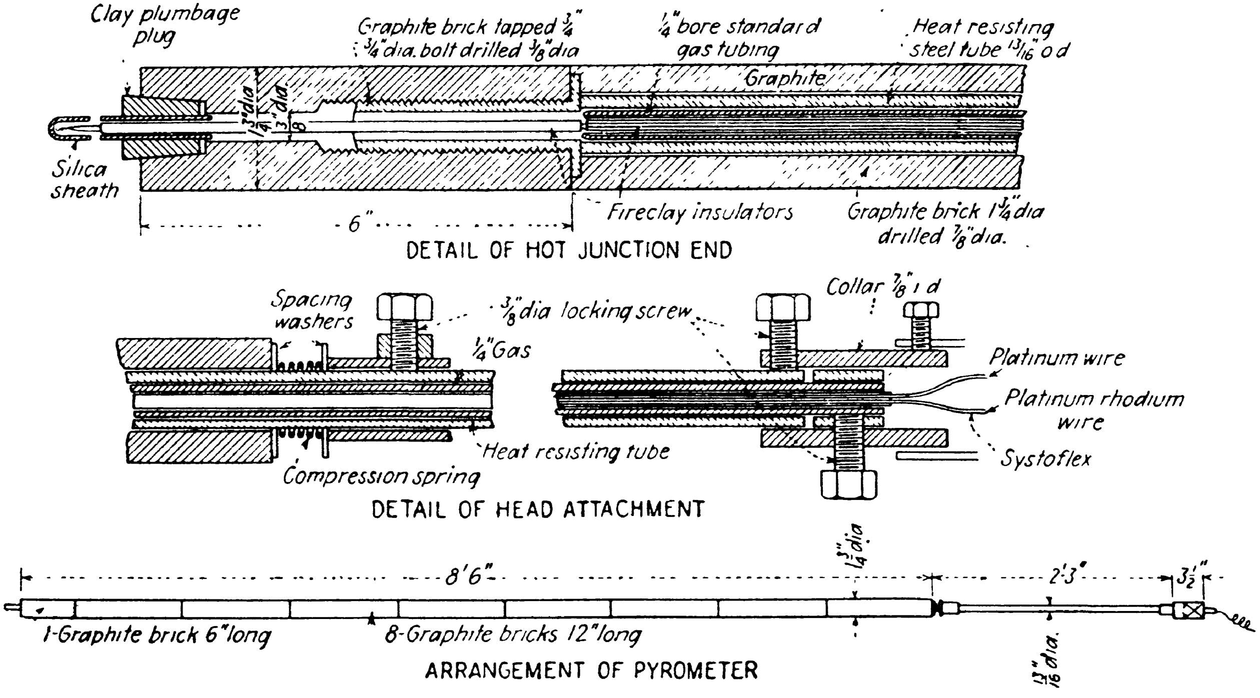


FIG. 209. Modifications of immersion pyrometers.

By courtesy of the Iron and Steel Institute and Tinsley Industrial Instruments) lead. This thermocouple is insulated internally as already described. Typical pyrometers are shown in Figs. 208 and 209 taken from the work of Land[23] and Manterfield.[24] When the wire at the hot junction has to be renewed, it can be pushed down from the cold end of the arm and a new junction can be made by twisting the wires together.

The weight of the pyrometers varies according to their use. An instrument designed for open-hearth work will weigh about 13 lb., and is fitted on a trolley. Smaller instruments for use on small furnaces may weigh only about 5 lb. They should have an accuracy of plus or minus 5 deg. C. and should be capable of quick action. The speed of temperature measurement requires special instruments and this aspect has been discussed by Land.[23]

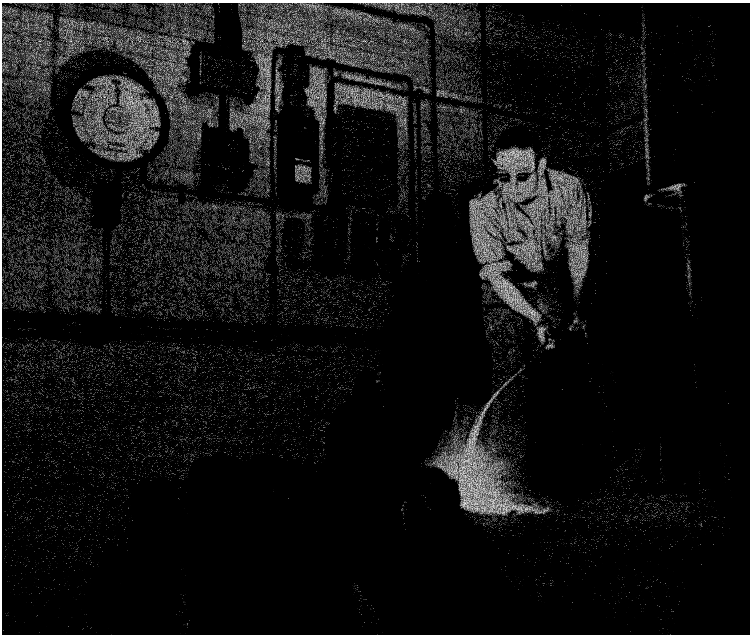


FIG. 210. " H.F." furnace immersion pyrometer.
(By courtesy of Tinsley Industrial Instruments, Ltd.)

The millivoltmeter is inexpensive, but is only accurate for certain external thermocouple resistance. As shown by Manterfield[24] this instrument has a rapid response and a high initial resistance, but can only be used where the external (thermocouple) resistance varies less than 1 ohm. Any pyrometer in which, due to the heating of the thermocouple wire or other causes, the resistance varies by more than one ohm should not be used with a millivoltmeter of this type.

A potentiometer gives accuracy independent of the thermocouple resistance. It can, therefore, be employed with pyrometers embodying long thermocouple wires exposed to wide temperature changes. A reflecting galvanometer is fitted into the instrument having a period of $\frac{1}{5}$ second. A slide wire, which is sufficiently robust to withstand severe works usage, is operated by a long wire and indicates the temperature on an open scale. The potentiometer is calibrated in millivolts covering a range of 0 to 48 mv. in twelve steps. A removal scale is also provided so that direct temperature readings over a range of 1,400 to 1,700 deg. can be obtained.

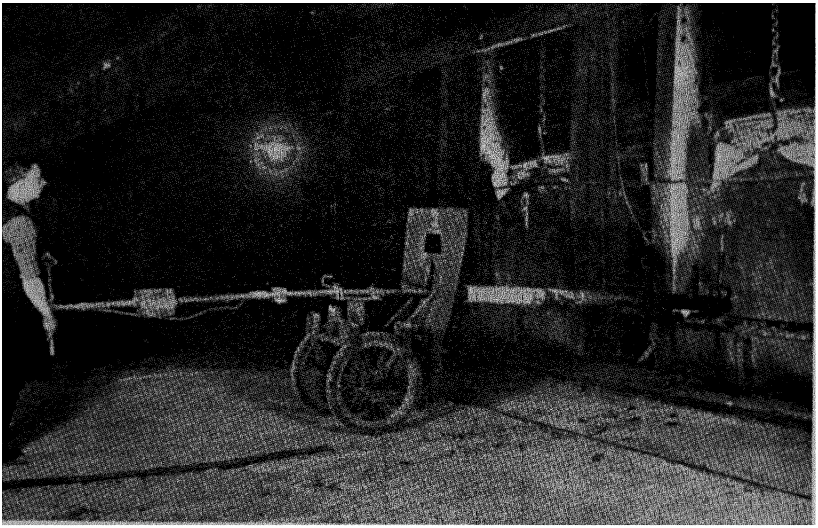


FIG. 211. Open-hearth furnace immersion pyrometer.
(By courtesy of Tinsley Industrial Instruments Ltd.)

A third method, introduced by the Tinsley Industrial Instrument Co., consists of a simple and robust amplifier, which converts the e.m.f. from the thermocouple into a strictly proportional current of sufficient magnitude to operate a direct reading or writing recorder in a period of about 2 seconds. Since the amplifier takes very little current from the thermocouple in comparison to the output current, it is not affected by any resistance in the thermocouple circuit. The accuracy of the amplifier and recorder is about 2 deg. C. and the calibration can be readily checked by means of a press-button switch on the amplifier. It is operated from A.C. supplies and is independent of fluctuations of the supply.

Fig. 212 shows a *multipoint system* developed by the Tinsley Industrial Instrument Co., in which each furnace is provided with thermocouple sockets, together with warning lights. When the immersion thermocouple is in operation on any furnace the warning lights are switched on at the other connections. The e.m.f. developed by the "quick immersion" thermocouple (platinum-platinum 13 per cent. rhodium) is conveyed to the amplifier, where it is converted into a proportional current of sufficient magnitude to deflect large indicators on the furnace stage and a recorder in the control office.

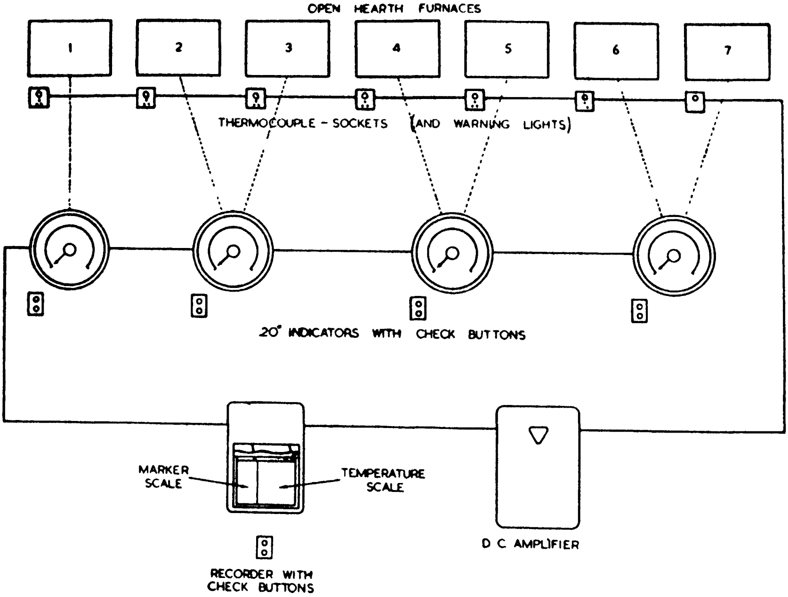


FIG. 212. Multi-point immersion pyrometer system.
(By courtesy of Tinsley Industrial Instruments Ltd.

Future Developments

The importance attached to industrial instrumentation and process control is emphasized by the extensive research which is at present taking place on this subject. The close co-operation between users and makers of instruments and between the various scientific bodies interested, is a proof of the advantages of this method of control and predicts further developments in the future. Some examples of new measuring techniques and their application in the iron and steel industry has been reviewed recently by Carlisle[25] of B.I.S.R.A.

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CHAPTER 15

THE MANUFACTURE OF SOME TYPES OF SPECIAL STEELS

The general procedure of producing the various classes of steels has been discussed in the chapters dealing with the different steelmaking processes. At one time there was a certain amount of prejudice against some of the methods of steel production. The greater degree of control, the more modern equipment available, and the more precise knowledge of the scientific principles underlying the different methods of production, has done much to change these views. A wide application is now permissible for the products of all the steelmaking processes.

The Bessemer process offers an ideal method of producing very low carbon steels, owing to the fact that much lower limits of carbon can be attained than in the other processes. The chief limitation of the Bessemer process is the high nitrogen content, which is detrimental in certain applications. This problem has been discussed on page 49, and it is obvious that the difficulties are receiving much attention, and considerable progress to a greater degree of control has already been made.

The Bessemer process and the hot-metal open-hearth process, where a high percentage of virgin pig iron is employed, produces a steel low in residual elements, such as nickel, copper, tin, etc. The residual metal content in the cold-pig and scrap process can, however, be controlled to a considerable extent by careful selection of the raw materials.

For the production of alloy steels, the acid open-hearth and the basic electric-arc processes are usually favoured. Large quantities of low alloy, together with many types of special and higher-alloy steels are produced by the basic open-hearth process. The fundamental factor in the production of high-grade steel would appear to be the degree of control exercised rather than the particular process employed.

The general principles underlying the production of these various types of steel have been discussed, but there are two classes of steel to which special reference must be made, namely rimming steel and grain-controlled steel.

Rimming Steel

As Mr. James Mitchell[1] stated, "the production of rimming steel is an art demanding accuracy of control, and the product is not, as is sometimes too loosely assumed, material melted and cast into moulds at random with a minimum of finishing addition". The production of rimming steel does require very accurate control and considerable skill on the part of the operative. This class of steel is generally produced by the acid or basic Bessemer or the basic open-hearth process. The Bessemer process is capable of producing very low carbon varieties better than the open-hearth process.

The structure of this type of steel has been discussed in Chapter 10, page 279, whilst typical examples of rimming ingots are shown in Figs. 146, 147 and 152. The theory explaining this blowhole formation has also been discussed and it would appear that the characteristics of this steel are due to the evolution of gas which results from the reaction:

$$Fe_3C + FeO = 4Fe + CO.$$

Some of the gas evolved may result from the liberation of gases occluded by the steel in the molten state, although recent work[2] indicates that the reaction theory is the most probable explanation. The liberation of gas due to this reaction and/or the release of the occluded gas causes a stirring action in the molten metal which results in differential freezing typical to rimming steel. This evolution of gas promotes a rimming or effervescent action on the top surface of the ingot, delaying its solidification.

Some years ago the author endeavoured to review the published work on the theory and practice of rimming steels.[3] A considerable amount of information on the subject has also been published by the Ingots Committee,[4, 5] whose work has been quoted in Chapter 10.

Fleming[6] suggests that there are four main types of rimming ingots:

(1) The ingot which neither rises nor sinks from the teeming level but which within about one minute of finishing teeming commences to freeze along the shore line and begins to rim. This rimming continues straight across until a flat shore line one to several inches wide has been established. The molten metal then begins to sink until solidification is completed, thus resulting in a slight depression in the centre of the ingot top. An ingot possessing these characteristics will be thick-skinned, the size and number of blowholes will be small and they will be deep-seated.

(2) The second type of ingot discussed is similar to the first ingot quoted, until the final settling, when a well some 3 to 5 in. wide is formed. At this stage the metal becomes mushy and builds up in the centre of the ingot forming a projection, which if not plated off will grow considerably in height. Ingots of this type are thick-skinned, but there is an increase in the number and size of the blowholes in the upper portion of the ingot, necessitating a greater discard.

(3) In the third type the metal starts to rise in the mould shortly after teeming, and continues to rise gradually whilst rimming proceeds. By the time solidification is completed, this rise may amount to several inches. Ingots showing this behaviour in the mould are thin-skinned with excessive blowholes situated at random throughout the ingot. The ingot will produce an excessive number of seamy and blistered bars.

(4) The fourth type of ingot begins to rim at the exact height to which it was teemed, leaving a flat top without any central rise. These ingots are of high quality, having a thick skin and a minimum number of blowholes. The bars rolled from ingots of this type usually inspect well, but yield slightly more seams and blisters than bars obtained from ingots similar to the first example quoted.

He suggests that the first type mentioned should be regarded as the ideal rimming ingot. Its production is favoured when the carbon is under 0·08 per cent. and the manganese about 0·30 per cent. He recommends that for the production of high-quality rimming steel the carbon should not exceed 0·16 per cent. and the manganese should be under 0·50 per cent.

The range of analysis is one of the limitations of this class of steel. In normal practice the carbon should not be over 0·25 per cent., whilst the manganese should not exceed 0·60 per cent. Another disadvantage is the typical segregation which occurs. This segregation has been discussed on page 320 and is shown in Fig. 165. Unless the process of manufacture is carefully controlled the blowholes may be too near the skin, in which case serious troubles may arise in the subsequent working of the ingot.

In spite of these disadvantages this class of steel possesses several merits and beneficial properties. The bars produced from rimming steels normally possess a very good surface. The very pure skin imparts a ductility which renders the steel admirable for deep-drawing work. The thickness of the skin is, however, of considerable importance as regards this property. A good effervescence in the moulds, upon which the thickness of the skin depends, is essential. Providing the piping and/or the position of the blowholes is controlled, the billet yields are very good.

The Ingots Committee[7] demonstrate the effect of the method of teeming on the thickness of this rim and their conclusions may be summarized as follows:

(1) With top-teeming, the solid rim or shell is considerably thinner in the lower than in the upper portions of the ingot.
(2) Tundish-teeming materially decreases this difference in thickness.
(3) With bottom-pouring, whilst there is a difference in the thickness of the solid rim, it is not so clearly marked as with top-teeming.

They also show that there is a tendency for the rim to be thicker in ingots obtained from the later periods of teeming from the same ladle. The solid rim invariably consists of material which is more pure than the average composition of the cast. They attribute the rimming or effervescent action chiefly to the interaction of iron oxide and carbon.

Reinertz[8] suggests that the successful production of this class of steel depends on an efficient, easily regulated furnace, because of the relatively high finishing temperature necessary; the careful control of the raw materials employed, and the intimate control of the refining and tapping practice. He recommends that the carbon melt should be 0·30 to 0·50 per cent. so as to assure a good boil in the furnace.

Referring to slag control he[8] suggests that spar should be used sparingly. Too much spar results in the formation of wild fluid slags at the end of the heat, and this type of slag causes an over-oxidized condition of the bath, loss of yield and a poor-quality product. When producing 0·10 to 0·15 per cent. carbon steels he recommends that the tapping slag should contain 18 to 23 per cent. FeO, but when the cast is to finish under 0·07 per cent. carbon, the tapping slag should contain 30 to 35 per cent. FeO. If the iron oxide exceeds these figures the slags will be thin and excessive de-oxidizers will be required. When the iron oxide content is less than these values, the slag will not contain sufficient oxide to produce the necessary rimming action. The residual manganese recommended is 0·08 to 0·12 per cent. and considerable importance is attached to the correct tapping temperature.

Washburn and Nead,[9] discussing the structure of rimming ingots and the phenomena underlying their production, attribute the boiling or rimming action to the evolution of CO produced by the interaction of FeO and C, and to the evolution of occluded gases, such as hydrogen and nitrogen, as their solubility in the steel decreases as a result of solidification. They point out some variables, such as the relation between the rimming time and the depth of the rim produced, which are fairly definite. They suggest that other factors, such as the relation of the FeO in the slag to the type of primary blowholes formed, are indirect and often offset by other variables.

They maintain that the chief variables affecting the thickness of the skin are the temperature of the mould, the temperature of the steel and the rate of pouring. The thickness of the skin decreases if the mould is too hot, that is, over 700 deg. C., or if it is too cold, especially if the surface is damp or rusty.

They[9] suggest that the size and position of the primary blowholes are affected by:

(1) *The FeO in the slag.* A low FeO is usually associated with a slow evolution of gas and an unsatisfactory rimming action, which results in an increase in the diameter and length of the blowholes. These blowholes usually begin to form closer to the surface thereby decreasing the thickness of the skin.

(2) *The type and amount of deoxidizers employed.* The type and amount of deoxidizers used in the ladle and in the mould affect the rate of gas evolution and thereby the type of skin formed. If the amount used is too great or if silicon alloys are employed, the effect is similar to having the FeO in the slag too low.

(3) *The temperature of the steel.* The temperature of the steel must be sufficiently high to allow free rimming action, but excessive temperatures may result in very wild effervescent action. This temperature would appear to depend on local practice and should be standardized accordingly.

(4) *The ingot dimensions.*

(5) *The height of the pour.*

(6) *The composition of the steel.*

The size and characteristics of the secondary blowholes are affected by:

(1) The FeO in the slag.

(2) The pouring temperature.

(3) The rate of solidification.

(4) The rimming time.

(5) The sulphur in the steel.

A high FeO content of the slag and high teeming temperatures are associated with the rate of solidification and the rimming time, and act to promote deeper-seated secondary blowholes with a higher concentration of impurities, which tend to prevent the blowholes from welding up when the ingot is rolled.

The relationship between the mould dimensions and the rimming action has been discussed by Mitchell,[10] who shows that moulds of different sizes require metal with varying degrees of effervescence to produce the same type of ingot. Varying sizes of ingots can be produced so long as the rimming characteristics of the steel are adjusted to suit the particular mould employed. In his investigations Mitchell[10] used four moulds and he found that the thickness of the rim increased and the ratio of the area of the rim to the area of the core decreased as the mould size increased. These results may be summarized:

Mould size (average)	24 in.	21 in.	19 in.	13 in.
Ingot weight	87 cwt.	62 cwt.	47 cwt.	26 cwt.
Average thickness of rim	4·4 in.	4·1 in.	3·8 in.	2·9 in.
Ratio area of rim to area of core ..	1·57	1·72	1·85	2·34

In these experiments the carbon content of the metal was kept constant.

In this work it is shown that larger ingots should be employed when extreme purity of the rim is desired, in preference to small ingots, but the effervescent action should be sufficiently vigorous to be finally controlled by aluminium additions. Other conditions being the same it is suggested that smaller-sized ingots give better mill yields.

The published data on rimming steels does appear to indicate that the type of ingot desired for the best results is the ingot which neither rises nor sinks appreciably after teeming, but which rims in level forming a flat top, with probably a slight hollow or well of small dimensions in the centre of the ingot. So far as the open-hearth process is concerned the factors favouring the production of this type of ingot appear to be:

(1) The selection of raw materials so as to avoid elements such as silicon, copper, nickel, etc., which may affect the solubility of gases. Although hydrogen is introduced by rusty scrap and moist materials in the charge, it is probable that this gas is evolved during an active carbon boil. Any hydrogen introduced by moist feed materials or alloys during the refining or finishing stages, however, is more serious and is not so easily eliminated.

(2) A suitable carbon melt to ensure a good vigorous boil which controls the state of oxidation and occluded gas content.

(3) Proper slag formation to ensure that the slag gives adequate protection to the metal against gas absorption and is still sufficiently fluid to allow the free escape of gas. The FeO content of the slag must be regulated to ensure the correct degree of effervescence.

(4) It would appear that the Brinell Density Quotient mentioned on page 286 is a useful empirical guide to the range of chemical analysis when adapted to local conditions. The limit of composition will fall between 0·04 to 0·25 per cent. carbon and not over about 0·60 per cent. manganese. The higher carbon and manganese contents will result in a weaker effervescence.

(5) The accurate control of temperature throughout the working of the charge and especially at the time of tapping.

(6) The standardization of casting-pit practice with special reference to:

i The methods of teeming.
ii The temperature of teeming.
iii The rate of teeming.
iv The ingot mould design and size.
v The cleanliness and temperature of the ingot moulds. Mitchell[1] suggests that the correct mould temperature appears to be between 100 and 150 deg. C.
vi The freedom from moisture in the launders, chute, ladle lining, runner bricks, etc.

Such conditions as a high FeO concentration, or a high casting temperature, or a rapid teeming speed, favour the early interaction of FeO and C, and skin blowholes, containing the product of this reaction, will result. Skin blowholes may also result from a very high hydrogen content. When the hydrogen concentration is normal, the evolution of this gas will be delayed resulting in the formation of deep-seated blowholes.

It is obvious that the production of rimming steels in the open-hearth furnace demands very careful control of the oxide additions and slag conditions. The duration and vigour of the carbon boil is an important factor. In the Bessemer process the carbon boil and the mode of slag formation is a natural sequence, and conditions favourable to the production of rimming steel occur automatically. It is true that in the basic Bessemer process the phosphorus is not eliminated until after the carbon has been removed, but it has been shown by Mitchell[1] that phosphorus is as effective as carbon in controlling the state of oxidation.

The same fundamental principles apply to the Bessemer processes as to the production of this type of steel in the basic open-hearth. In the *acid Bessemer* the same blowing procedure is adopted as when making other qualities of steel. The analysis of the mixer or molten metal must be carefully controlled in order to produce a " dry " slag. This control consists of regulating the Si and Mn contents of the metal so that a Si/Mn ratio of 2·0 to 2·5 is maintained. This acid Bessemer practice has been ably described by Swinden and Cawley.[11]

The production of rimming steel in the basic converter, which has been reviewed by Kerlie,[12] involves the normal basic blowing procedure. The metal is blown until the phosphorus is reduced to the desired limits and the metal is deoxidized by suitable additions to either the converter or the ladle. Normally cold spiegel is added to the converter, whilst small hot ferro-manganese is added to the stream as the metal is poured from the converter to the ladle. The behaviour of the metal in the first mould is usually observed and the degree of effervescence forms a guide to the amount of aluminium which must be added to the subsequent moulds to obtain satisfactory results.

Grain Size Control

As pointed out by Bain,[13] inasmuch as certain alloy steels, especially those containing vanadium, tungsten, molybdenum, and to some extent, the high carbon chromium steels, owe their valuable properties in some part to the natural retention of fine grain size at heat-treating temperatures, grain size control may be said to have been practised in their manufacture. The modern term "controlled grain size" refers to a particular predisposition imparted in preparing the molten metal for pouring into the ingot moulds. By grain size control reference is made to the "inherent grain size". The inherent grain size is determined under standard conditions, and is not the grain size influenced by such factors as ingot size, casting temperature and conditions, hot or cold work, or heat-treatment. It is a condition inherent in the steel as cast into the ingot moulds.

The standard method of classifying the inherent grain size is prescribed by the American Society for Testing Materials.[14] This method consists of removing all scale from the sample, either by filing or grinding. The sample is packed in a box containing a good carburizing compound and the box is adequately sealed to avoid the escape of any carburizing gases and to prevent the formation of a hyper-eutectoid zone. The box and its contents are heated in two hours and soaked for six hours at 927 deg. C. It is then allowed to cool slowly down to 400 deg. C. or less before being emptied. The temperature should be regulated to within ± 5 deg. C. A section is cut from the sample, polished on the cut face and etched with boiling alkaline sodium picrate solution. The carburized case has then a clearly defined grain size, which is examined on a projection microscope at a magnification of 100 diameters. The projected field is compared with those given on a standard chart which show typical fields for each grain size enabling the inherent grain size to be assessed.

The grain size is reported as a number, which is really the number of grains per square inch at a magnification of 100 diameters. The following figures illustrate the basis of the eight standard grain sizes normally adopted:

Grain Size Index Number	Number of Grains per sq. in. × 100		
	Mean	Maximum	Minimum
1	1	1·5	—
2	2	3	1·5
3	4	6	3
4	8	12	6
5	16	24	12
6	32	48	24
7	64	96	48
8	128	—	96

In general it may be stated that fine-grain as compared with coarse-grain steel slightly reduces the maximum strength and yield point, increases slightly the ductility as shown by the elongation and reduction of area, and increases very considerably the toughness as shown by the Izod Test. As stated by Swinden and Bolsover[15] grain-controlled steel facilitates heat treatment and renders standardization of machine-shop practice more certain and reliable.

The original theory advanced in America, and still common in much technical literature, is that the fine grain is achieved through the agency of minute particles of alumina distributed throughout the steel. Swinden and Bolsover[15] show that there is no essential difference between the aluminium content of fine- and coarse-grain steels, but this statement does not necessarily disprove the original theory. The fundamental factor appears to be the stage of deoxidation at which the aluminium addition is made.

It appears that the steel must be carefully deoxidized in the usual manner and an aluminium addition made to the ladle after the normal deoxidation has been accomplished. In this manner small particles of alumina are distributed evenly throughout the steel, which act as nuclei promoting crystallization. The number of nuclei present will affect the fineness of the grain.

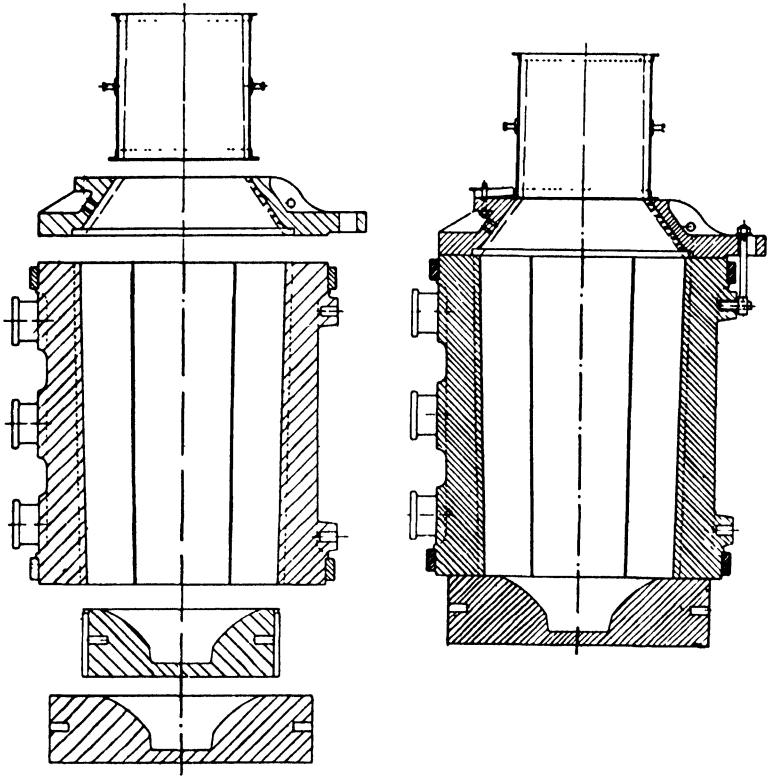

FIG. 213. Large ingot moulds—head and chill.
(*By courtesy of B.I.S.R.A.*)

The amount of aluminium to produce a precise effect on the grain size appears to depend on the carbon content of the steel, the temperature of the metal, the state of deoxidation and the manner in which the aluminium is added. These conditions must be determined for each particular plant and standardized.

Production of Large Ingots

It is sometimes necessary to produce very large ingots up to 150 to 200 tons in weight. These ingots are usually employed for heavy forgings or pressings. They are cast into ingot moulds, which consist of three component parts—the bottom, the chill and the feeder-head, as shown in Figure 213. These mould chills are usually octagonal on cross-section, having dimensions up to 110 inches, accordings to their capacity. Ingots up to about 45 to 50 tons can normally be teemed direct from a single ladle in the usual manner, but larger ingots demand the co-ordination of several furnaces. The size of ingot which can be teemed from a single ladle depends on the capacity of the furnace available. With very large ingots, however, two or more furnaces may have to be operated in conjunction to give the necessary tonnage, and this procedure requires very careful and accurate planning and co-ordination.

The moulds are carefully prepared and a tundish trough is frequently employed. Even if this tundish is not used for the teeming of the whole ingot, it is generally employed until there is 8 to 10 inches of metal in the bottom of the mould. This precaution reduces the likelihood of a defective ingot surface.

When two or more furnaces are employed, it is customary to employ a feeder or a primary ladle. The first ladle of metal is transferred from the furnace to the mould. When the second furnace is ready for tapping, teeming from the first ladle is commenced. When the second ladle is filled and the head of metal in the first ladle has dropped sufficiently the metal from the second ladle is poured into the primary ladle. For very large ingots a third or fourth ladle may be necessary according to the furnace capacity, in which case these additional ladles are poured into the primary ladle. Care must be taken to ensure that there is no stoppage in the actual pouring of the metal. A typical layout is shown in Fig. 214.

The size of the nozzles employed in the tundish and the ladle require accurate calculation to ensure the correct rate of flow. The nozzle in the secondary ladle must be of sufficient size to ensure the height of metal in the primary ladle being maintained at the correct level.

The feeder-head requires to be of adequate size to feed the ingot. Frequently the size of this head is also influenced by the fact that it is used for manipulative purposes during the forging operation. To ensure that the feeder-head adequately deals with the piping, the head is frequently 20 to 30 per cent. of the total weight of the ingot. Piping in ingots of this type and size is a serious defect, which must be avoided by the employment of a feeder-head of sufficient size and by the use of anti-piping compounds.

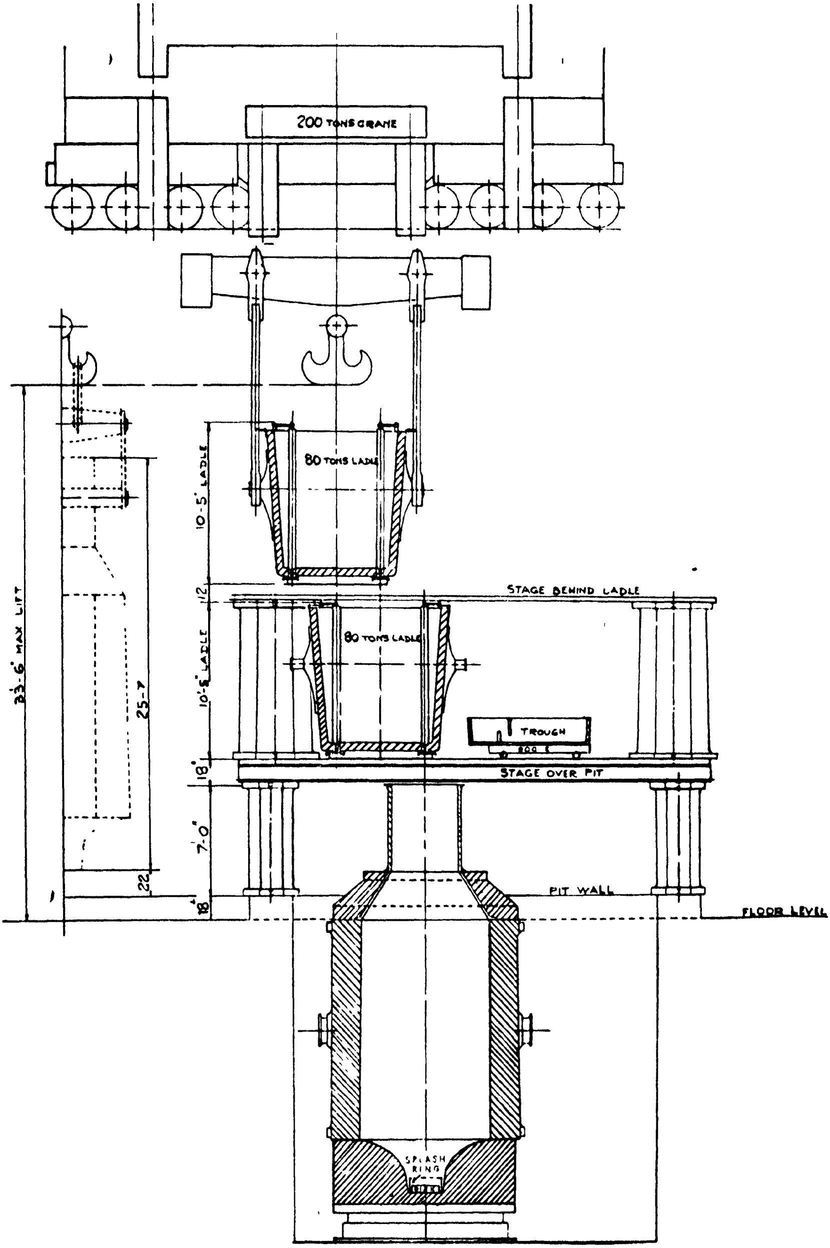

FIG. 214. Teeming of large ingots.

(*By courtesy of B.I.S.R.A.*)

These ingots are prone to the same transverse and corner cracks as were discussed in Chapter 10, pages 293, so that the rate of teeming must be regulated.

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CHAPTER 16

STEELWORKS REFRACTORY MATERIALS

The chief types of refractory materials employed in steelworks may be classified as follows:

Acid	Fireclays and firebricks.
	Semi-silica refractory materials.
	Silica materials.
Basic	Dolomite.
	Magnesite.
	Chrome-magnesite.
Neutral	Chrome ores.

In addition to these materials there are certain special refractories, such as sillimanite, etc., which find a limited application. Typical analyses and properties of these various refractories are given in Tables LVI and LVII.

It will be noted that the fireclays and firebricks in general contain under 75 per cent. silica with under 38 per cent. alumina, although some grades of aluminous firebricks range between 38 and 45 per cent. alumina. The normal range of semi-silica material is between 75 and 92 per cent. silica, whilst the best silica refractories consist of over 92 per cent. silica.

The fireclays are associated with the Coal Measure rocks and, although they vary considerably in composition, there are abundant deposits in this country. Ganister and quartzite rocks containing over 92 per cent. silica are also associated with the Carboniferous series and are found in the Sheffield and Durham districts, in North and South Wales and in Scotland.

Dolomite, which is prepared by the calcination of magnesian limestone, is a mixed oxide of calcium and magnesium. This material is in plentiful supply in several parts of the country. There are also supplies of chrome ore in the Shetlands, but this material is not too suitable for the manufacture of refractory materials, and, therefore, much, if not most, of our chrome ore requirements must be imported. Typical analyses of chrome ores are given in Table LVIII.

TABLE LVI

Typical Refractory Materials.

Type.	SiO_2	Al_2O_3	TiO_2	Fe_2O_3	Cr_2O_3	MgO	CaO	Alkalis
Firebrick	55/70	25/40	1·0/2·0	2·0/5·0	—	0·5/1·0	0·5/1·0	1·0/2·0
Aluminous	50/55	38/45	1·0/2·0	2·0/4·0	—	0·5/1·0	0·5/1·0	0·5/1·0
Semi-Silica	75/90	8/15	0·5/1·0	1·0/2·0	—	0·5/1·0	0·5/1·0	0·5/1·5
Silica	94/96	0·5/2·0	0·2/0·5	0·5/1·5	—	0·1/0·5	2·0/2·5	0·5/1·5
Sillimanite	25/35	55/65	0·5/1·0	0·5/1·5	—	0·5/1·0	0·5/1·0	0·5/1·0
Chrome	3/10	10/25	tr.	12/15	35/45	15/35	2·0	tr.
Magnesite	1·5/5·0	1·0/4·0	tr.	2·0/8·0	—	84/92	2·0/5·0	tr.
Chrome-magnesite	3·0/10·0	5·0/15·0	tr.	8·0/20·0	20/35	40/60	1·0/4·0	tr.
Dolomite	12/15	2·0/3·0	tr.	2·0/4·0	—	38/42	38/42	tr.

TABLE LVII

TYPICAL PROPERTIES OF REFRACTORY MATERIALS (VARIOUS SOURCES)

Type.	Refractoriness ° C.	Refractoriness under load 28 lb./sq. in. (Failure).	True Sp. Gr.	Bulk Density.	Thermal Expansion up to 1000 ° C.*	Thermal Conductivity B.Th.U./Ft²/hr/° F./in.
Firebrick (25/30% Al_2O_3) ..	1,610–1,670	1,380–1,460	2·6/2·7	1·9/2·1	Under 0·5%	6·10/9·50
Firebrick (30/35% Al_2O_3) ..	1,630–1,690	1,460–1,520				
Firebrick (35/38% Al_2O_3) ..	1,650–1,730	1,520–1,580				
Aluminous (38/45% Al_2O_3) ..	1,710–1,770	1,580–1,650				
Semi-silica	1,630–1,690	1,500–1,610	2·5/2·6	1·8/2·0	—	0·00/10·00
Silica	1,710–1,750	1,650–1,710	2·3/2·4	1·7/1·9	1·2/1·3	9·40/12·30
Sillimanite	about 1,770	over 1,650	3·0/3·2	2·0/2·2	0·5	11.00/13·00
Chrome	1,850	1,350–1,460	3·7/4·2	2·95/3·18	—	9·00/15·00
Magnesite	1,850	1,500–1,630	3·5/3·7	2·7/3·0	1·3/1·4	20·00/30·00
		50 lb.-sq. in.				
Dolomite	—	1,470–1,550	—	—	about 1·3	13·2/20·6
Chrome-Magnesite	1,850	1,500–1,710	3·5/3·9	2·7/3·0	1·2/1·3	—

* At temperatures higher than 1,000° C. most refractories develop a softening of the matrix, whereby some of the expansion is absorbed by the brick.

TABLE LVIII

TYPICAL IMPORTED CHROME ORES

Country	SiO_2	Al_2O_3	FeO	Cr_2O_3	CaO	MgO	Total
	per cent.	per cent.	per cent.	per cent.	per cent.	per cent.	per cent.
Turkey ..	1·50	15·00	12·10	56·80	0·70	14·00	100·1
U.S.A. ..	3·00	11·00	19·50	54·50	1·50	8·00	97·5
U.S.A. ..	7·50	16·80	15.00	42·50	0·90	16·50	99·2
Canada .	9·50	8·30	13·80	42·60	—	21·10	95·3
Rodesia ..	6·00	16·00	23·80	41·30	1·50	11·80	100·4

There are no magnesite deposits in this country and nearly all of our magnesite requirements must be imported. During World War II the recovery of magnesite from sea-water and from dolomite was developed and this source of magnesite is now proving of considerable value in the refractory industry. Typical analyses of magnesites, together with the countries of their origin, are given in Table XXVI on page 188. Owing to the fact that most of our magnesite is imported this material is replaced whenever possible by dolomite. In America, where there are abundant deposits of magnesite, this material is employed in preference to dolomite for the lining of the hearths of basic-arc and basic open-hearth furnaces, etc.

TABLE LIX

MINERALOGICAL CONSTITUTION
(Hyslop)

	Constituents	Matrix
Silica ..	Cristobalite (SiO_2) and/or Tridymite (SiO_2)	High silica glass containing some lime and alumina.
Fireclay ..	Mullite ($3Al_2O_3 . 2SiO_2$); Cristobalite (SiO_2) and/or Tridymite (SiO_2)	felspathic, alkali-aluminium silicate glass.
Magnesite .	Periclase (MgO)	(a) Magnesium ferrite ($MgO . Fe_2O_3$) crystals (b) Forsterite ($2MgO . SiO_2$) crystals
Chrome ..	Chromite [$(MgFe)O . (CrAlFe)_2O_3$]	(a) Enstatite ($MgO . SiO_2$) crystals (b) Forsterite ($MgO . SiO_2$) crystals
Chrome-Magnesite ..	Chromite [$(MgFe)O . (CrAlFe)_2O_3$] or Periclase (MgO)	Forsterite ($2MgO . SiO_2$) crystals

The physical and chemical properties of refractory materials are influenced to a greater extent by their *constitution* than by the chemical composition. Recently Hyslop[1] reviewed the constitution of the various refractories and his general conclusions are summarized in Table LIX, which is taken from his paper.

Fireclays

Fireclays consist of numerous hydrous alumino-silicate minerals, which on heating break down into several constituents, but which tend to form the only compound of silica and alumina that is stable at high temperatures, namely, mullite ($3Al_2O_3$, $2SiO_2$). It has been shown by X-ray analysis that mullite begins to form at about 1,000 deg. C., but the crystals are too small to be seen with the aid of the microscope. At increasing temperatures the formation of mullite is accelerated. A well-fired firebrick will contain mullite and quartz, with a certain amount of cristobalite. Some of the original quartz will have been transformed into cristobalite, but even in bricks which have been in service for lengthy periods quartz grains will still be visible. The equilibrium diagram for these silica-alumina compounds is shown in Fig. 215. The high temperature strength, resistance to slag action, and many other useful properties of firebricks depend on the development of mullite and a glassy bond.

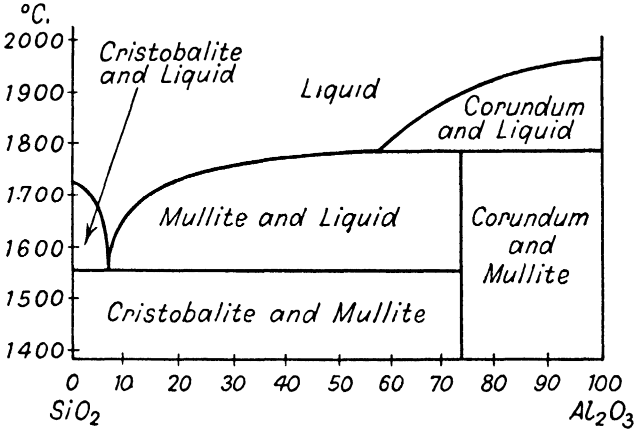

FIG. 215. Alumina-silica equilibria diagram.

Fireclays and firebricks have a limited application in steelworks, being employed chiefly as pitside refractories and in some cases in the regenerators of open-hearth furnaces. When they are employed in the regenerators it is the usual practice to employ high alumina firebricks for the upper courses where a higher temperature prevails and where there is more likelihood of slag action. High-duty firebricks are used as the bearer blocks of regenerators.

Singer[2] suggests that runner brick refractories should be made from fireclays with a low alumina content. These siliceous clays do not sinter and the bricks prepared from them retain their fine-grain structure and remain tight and strong. Recently Rait[3] demonstrated that the quality of runner bricks affected the non-metallic inclusion of high manganese steel. Manganous alumino-silicate slag was formed on the runner brick by reaction between the manganese in the steel and the free silica in the brick. Under certain conditions this slag was carried forward into the steel causing non-metallic inclusion. When the alumina content of the bricks was raised from 30 to 38 per cent. this trouble disappeared. The correct selection of quality in regard to casting-pit refractories has an important bearing on the exogenous inclusions in the finished steel.

The stopper sleeves are subject to chemical attack and must, therefore, possess a dense structure into which the slag cannot penetrate. They must also be capable of withstanding considerable temperature changes. Heindl and Cook[4] showed in a series of trials that the sleeve brick with the lowest porosity gave trouble due to cracking and spalling, whilst the best results were obtained from bricks which had a high refractoriness, a fairly high porosity, a fairly uniform texture, a high expansivity and a moderately high extensibility. The sleeves which appeared to give the best results in these trials had a refractoriness of cone 32-33 (approximately 1,725 deg. C.), a porosity of 21·8 per cent., a Young's modulus of 1,842,000 lb. per sq. in., an extensibility of 0·553 per cent. and a linear expansion of 0·570 per cent. over a range of 25 to 1,000 deg. C. These sleeves showed an analysis of:

	Per cent.
Silica	55·42
Alumina	38·40
Lime	0·14
TiO_2, Fe_2O_3, etc.	4·15

It was found that when in contact with acid slags a silica range of 55·42 to 65·99 per cent. gave satisfactory results.

Chesters[5] in his classical work on steelplant refractories stresses the importance of casting-pit refractories and comments on the cost of such materials in many modern steelworks and their effect on non-metallic inclusions. In spite of their effect on ingot costs it would appear that this branch of refractories has been somewhat neglected and a review of the materials employed would be justified. In connection with sleeve bricks he[5] shows that these cover the following range of analysis:

SiO_2	53 to 62 per cent.
FeO	3 to 5 per cent.
Al_2O_3	30 to 39 per cent.
TiO_2	1 to 1·5 per cent.
CaO	0·04 to 0·12 per cent.
MgO	0·72 to 1·60 per cent.
Loss on ignition ..	about 0·28 per cent.

The porosity ranges from 15·3 to 29·0 per cent., whilst the bulk density varies from 1·9 to 2·10 and the apparent specific gravity from 2·42 to 2·67. It is interesting to note that the cone melting point of the high silica sleeve was 1,665 deg. C. as compared with 1,720 deg. C. in the case of the lower silica (53·64 per cent.) and the higher alumina (38·96 per cent.) sleeve.

Stoppers and nozzles require considerable mechanical strength and resistance to erosion, which demand a homogeneous structure together with freedom from spalling. As a rule these refractories show a narrower range of analysis, containing from 53 to 55 per cent. silica and 28 to 40 per cent. alumina. The average iron oxide content is about 3·50 per cent., whilst the porosity varies from 16·5 to 17·0 per cent. and the cone melting point is usually over 1,700 deg. C. Leun[6] states that the best results are obtained when the porosity is as low as 12 per cent.

Regarding ladle bricks there are two schools of thought. Some works prefer a siliceous type of firebrick, which has a low refractoriness, but which vitrifies in service giving an increased resistance to corrosion and erosion, due largely to low permeability. At other plants a higher alumina brick is advocated. This higher alumina brick provides a greater slag resistance and a higher refractoriness. Chesters[5] discusses three interesting types of ladle bricks whose properties can be summarized as follows:

Code No.	34	32	33
Type	Low Alumina	High Alumina	High Alumina
App. porosity	18·7 per cent.	13·7 per cent.	17·0 per cent.
Permeability to air (c.g.s. units) ..	0·0036	0·014	0·046
After-expansion or contraction. ..	plus 3·6 per cent.	minus 1·4 per cent.	minus 0·3 per cent.
Cone melting point ..	deg. C. 1,580	deg. C. 1,680	deg. C. 1,710

He states that the best results were obtained from the brick quoted under code No. 33.

One of the chief causes of premature ladle-lining failures is due to bad joints between the bricks. The provision of good joints is a function of good bricklaying, the quality of the bricks and the truth of the bricks to shape and size. This truth to shape and size cannot be overstressed. The after-expansion of the siliceous type of ladle brick reduces the trouble due to bad joints.

The siliceous type of ladle brick frequently has a cone melting point of about 1,500 deg. C. with a porosity of under 15 per cent. The analysis of this type of brick may approximate:

Alumina	..	28 per cent.
Iron oxide	..	3 per cent.
Alkalis	..	3 per cent.

It will often have a permeability as low as about 0·002 c.g.s. units. High refractoriness is usually associated with high porosity, but low porosity and permeability can be obtained by fine grinding. In the author's experience the ladle bricks which have given the best results have generally been those with low porosity, reasonable refractoriness and truth to shape and size.

Silica Refractories

The mineralogical constitution of silica refractories is shown in Table LIX as consisting of cristobalite and/or tridymite, with a high silica glass containing some lime and alumina as a bond.

Silica has three principal allotropic modifications:

a-Quartz ..	having a specific gravity of 2·65.
a-Cristobalite ..	having a specific gravity of 2·33.
a-Tridymite ..	having a specific gravity of 2·27.

These various modifications are all stable at ordinary temperatures, but are capable of being transformed when heated through different

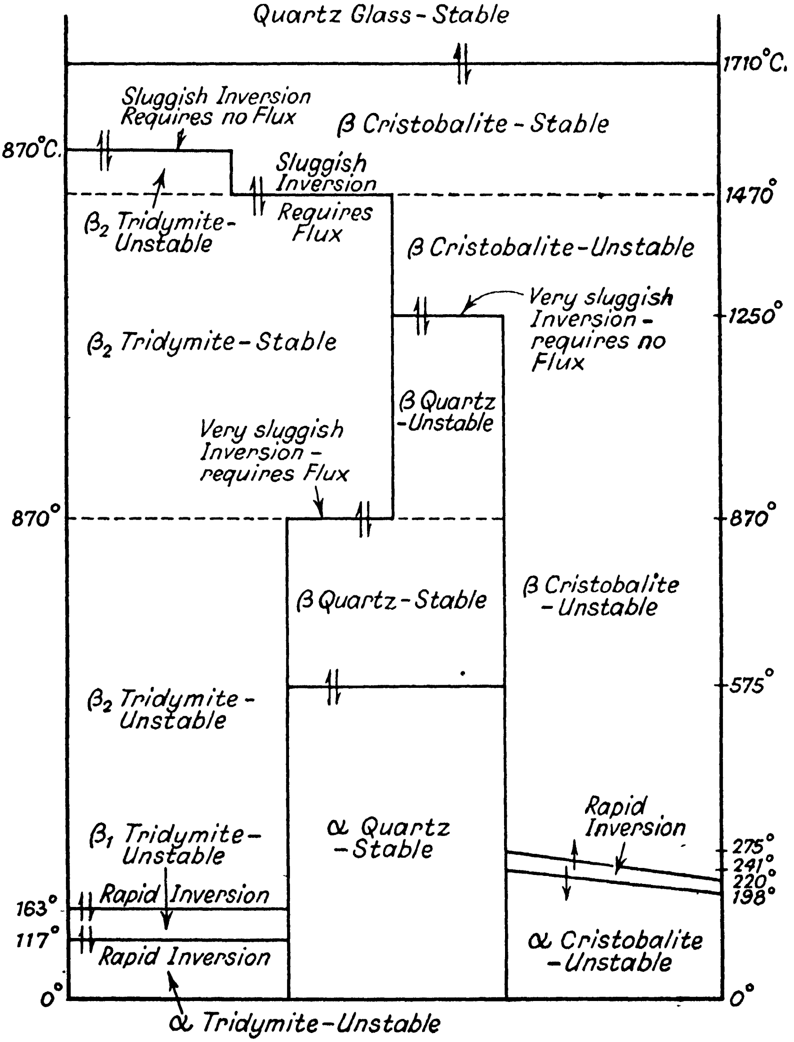

FIG. 216. Transformation of silica minerals.

temperature ranges. Quartz is transformed into tridymite on very prolonged heating at 870 deg. C., whilst shorter firing between 870 deg. and 1,470 deg. C. will result in this transformation. On heating between 1,470 and 1,710 deg. C. quartz and/or tridymite is transformed into cristobalite. These transformations or inversions are accompanied by volume changes. According to Craddock and Myers,[7] when quartz is converted into cristobalite a volume increase of 13 per cent. occurs, whilst a further increase of 3 per cent. takes place when cristobalite is inverted to tridymite. These transformations and the structural changes occurring are fully discussed by Chesters.[5]

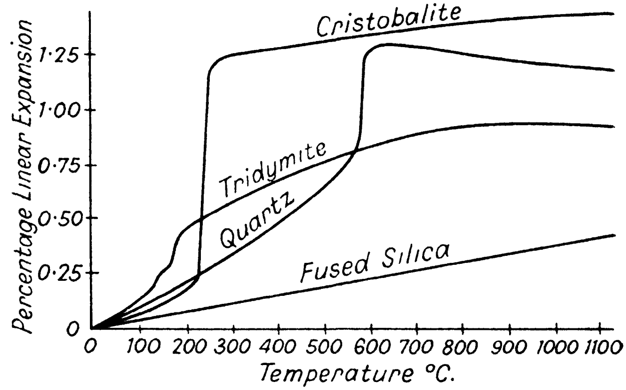

FIG. 217. Thermal expansion of silica modifications.

In an unfired brick the silica exists as quartz, but on heating in the kiln this quartz is transformed into various proportions of cristobalite and tridymite, according to the degree of burning. The specific gravity of the brick material will be an indication of the degree of firing. If the specific gravity is as high as 2·45 a considerable amount of unconverted quartz will be present in the brick, whilst a specific gravity of 2·32 to 2·37 indicates adequate firing and transformation to cristobalite and/or tridymite. These inversion are shown in Fig. 216.

Whenever these silica minerals are heated or cooled through certain ranges additional inversions take place as shown by Jones,[8] whose figures are given in Table LX, whilst the expansion which occurs on heating these various forms of silica is shown in Fig. 217. It will be noted that the greatest expansion occurs between 200 and 275 deg. C., whilst any unconverted quartz will be responsible for a further volume change at 573 deg. C.

TABLE LX

(Jones[8])

Temperature deg. C.	Change in Form	Volume increase Per cent.
117	a-tridymite to B_1-tridymite	0·14
163	B_2-tridymite to B_2-tridymite	0·20
200–275	a-cristobalite to B-cristobalite	2·80
573	a-quartz to B-quartz	0·86

These volume changes set up internal stresses within the brick and if not carefully controlled will result in the brick being fractured. This fracturing is known as spalling and occurs when the bricks are heated too rapidly through these temperature ranges. In order to prevent this spalling Jones[8] recommends a carefully planned warming-up schedule, which in the case of a 70-ton open-hearth furnace should approximate:

Temperature range deg. C.	Time in hours	Rate of temperature rise
15 to 200	10	18·5 deg. C. per hour
200 to 300	8	12·5 deg. C. per hour
300 to 550	12	22·0 deg. C. per hour

and after the furnace is brought up to 550 deg. C., it should be maintained between 500 deg. and 550 deg. C. until it is gassed. Accurate pyrometrical control of this heating, by such methods as roof pyrometers, is strongly recommended.

These figures demonstrate that silica bricks in which tridymite predominates should be least susceptible to spalling due to the low volume changes which occur with this modification of silica. When tridymite (sp. gr. 2·27) is converted into cristobalite (sp. gr. 2·33) a slight contraction in volume occurs. In highly converted bricks, those containing a high proportion of tridymite become fractured due to mechanical stresses during the early stages of heating, there is s tendency for these bricks to drop out of a furnace roof or suspended structure. This action is less likely to occur in bricks containing a fair proportion of quartz. It is for this reason that many operators specify bricks with a specific gravity of 2·35 to 2·37 for furnace roofs.

For repairs to door arches and similar structures this after-contraction may cause the arches to drop seriously and a brick with a higher specific gravity, or one containing more unconverted quartz, is recommended. The expansion of this quartz results in the maintenance of tight joints.

Some years ago Rees[9] pointed out the increased resistance to spalling which was imparted to silica bricks by small amounts of TiO_2. He showed that in certain South African bricks the TiO_2 was as high as 2·5 per cent. and the presence of this constituent greatly increased resistance to spalling. He reported the existence of one quartzite deposit on the north-east coast which had a TiO_2 content 1·5 to 2·0 per cent., which showed a similar resistance to spalling.

It should be noted that in the absence of fluxes quartz may change straight to cristobalite, but experience has shown that the presence of 1·0 to 1·5 per cent. iron oxide in the matrix greatly increases the cristobalite-tridymite inversion.

The refractoriness of silica bricks is a function of the impurities present in the brick. Kraner[10] discusses the SiO_2-CaO-Al_2O_3 equilibrium diagram and demonstrates the effect of alumina on the under-load strength at elevated temperatures. This strength is also reduced by alkalis and to a lesser degree by titania. This decrease in strength is undoubtedly due to the formation of " melt". It is, therefore, important that the alumina and alkalis should be kept low, whilst the lime content should be controlled so as to provide ample bond, without unduly lowering the refractoriness-under-load.

The porosity of silica bricks varies from about 20 to 35 per cent., and it was suggested that this property affected the slag penetration. Although the argument appears logical, as shown by Craddock and Myers[7] bricks with widely varying porosities have been found to give equal service in the roofs of open-hearth furnaces.

Formerly coarse-texture bricks, with particles ranging up to ¼ in. in size, were advocated for furnace roofs, but Chesters[5] suggests that better results are obtained from finer-texture bricks, which have usually a better shape, glaze more readily in service and wear more evenly. The strongest bricks will, of course, be those in which the bond is evenly distributed.

Chesters makes an interesting reference to a German specification for furnace roof bricks (D.I.N. 1088) which calls for:

Chemical analysis—	Silica	..	94·5 per cent. minimum.
	Alumina	..	2·0 per cent. maximum.
	Lime	..	3·5 per cent. maximum.

Cone melting point, at least Seger cone 32 (1,710 deg. C.).

Refractoriness under load of 2 kg/sq. cm., commencement of failure at least 1,630 deg. C.

Total porosity, under 25 per cent.

Specific gravity, 2·43 maximum.
Cold crushing strength over 100 kg./sq. cm.

Better results have been obtained when the bricks have been 94·5 per cent. silica minimum, under 2·0 per cent. alumina and under 2·5 per cent. lime. It is suggested that such bricks should possess a refractoriness under a load of 25 lb./sq. in. for one hour at 1,600 deg. C. without failure.

Semi-Silica Refractories

In recent years semi-silica refractories have been finding increasing application in steelworks. Their properties make them suitable for use in reheating furnace roofs, regenerator chambers, furnace doors and in any position where a high refractoriness linked with resistance to thermal shock and slag action is required.

This type of brick is:

1. Volume-constant in use.
2. Highly resistant to thermal shock.
3. Highly resistant to slag action.

The general range of analysis is:

SiO_2	.. 88 to 92 per cent.
Al_2O_3	.. 5 to 9 per cent.
Fe_2O_3	.. 1 to 2·25 per cent.
CaO	.. 0·1 to 1·0 per cent.
MgO	.. trace to 0·3 per cent.
Alkalis	.. 0·2 to 0·8 per cent.

These bricks usually have a porosity of 20 to 25 per cent. and a cone melting point of over 1,600 deg. C.

These bricks can be made from either naturally occurring sand-clays or artificially prepared batches. They require careful drying and even more careful firing, if a satisfactory product is to be obtained. The actual milling and grading appears to be of supreme importance, as the grain size of the brick has considerable effect on its behaviour in service.

Dolomite Refractories

Reference has already been made to the use of dolomite for basic converters, basic open-hearth furnaces and for basic-arc furnaces in the appropriate chapters. The dolomite brick has been cited as a substitute for the more expensive magnesite brick for many steelworks applications. The chief difficulty with this class of refractory is the tendency to hydrate when exposed to moisture, and efforts have been made to produce stabilized bricks.

Dolomite is usually stabilized by firing with some siliceous material, such as serpentine, whereby the CaO combines with the silica to form a calcium silicate. It has been suggested that tricalcium silicate is the main constituent of this reaction. Therefore, a stabilized dolomite brick will contain mainly periclase and tricalcium silicate, together with some calcium orthosilicate, calcium ferrite and brown millerite ($4CaO . Al_2O_3 . Fe_2O_3$).

The question of the stabilization of dolomite bricks has been reviewed by a special committee on refractories[11] in which it is stated that calcium metasilicate had been regarded as the stabilizing agent, but that this fact was by no means certain. In reviewing the properties of dolomite bricks this committee gave the following summary:

Porosity	19·5 to 31·0 per cent.
After contraction (1 hr. at 1,600 deg. C.)	1·8 to 2·7 per cent.
Refractoriness under load (50 lb./sq. in.).	
Comm. of subsidence	1,310 to 1,400 deg. C.
Failure by shear	1,470 to 1,555 deg. C.
Chemical analysis—Silica	16·0 to 18·1 per cent.
Alumina	3·1 to 3·4 per cent.
Ferric oxide	2·8 to 3·4 per cent.
Lime	36·9 to 41·2 per cent.
Magnesia	36·0 to 38·2 per cent.

It is possible to expose this type of brick to three months' immersion in water without hydration.

Certain types of dolomite bricks show a tendency to "dust" after heating and it has been suggested that this dusting is probably associated with the beta dicalcium silicate to gamma dicalcium silicate change. Bates and Klein[12] observed this change and its possible effect on dusting and suggested that it could be overcome by the addition of a small quantity of boric or chromic oxide to the batch. In general the stabilization is achieved as mentioned by Chesters[13] by adding some siliceous material, such as magnesium silicate, and a stabilizer such as recommended by Bates and Klein.

Chesters[13] gave an excellent summary of the properties of this type of brick in which he suggested the porosity should be between 15 and 25 per cent., largely depending on the grading and moulding pressure. The cold strength should be about 4,000 lb./sq. in., whilst the refractoriness under load should be 1,600 deg. C. for the fail point in the 50 lb./sq. in. rising temperature test. The spalling resistance index is generally somewhat lower than that for magnesite bricks, but special dolomite bricks will give 30 plus reversals. They do not tend to burst in the presence of iron oxide, but have a lower slag resistance than magnesite. The thermal expansion up to 1,000 deg. C. is 1·3 per cent. and the thermal conductivity is about two-thirds that of magnesite bricks.

The application of this type of brick is usually limited to the lining of basic open-hearth and arc furnaces, where they give performances equal to magnesite bricks at about two-thirds the cost. If employed in a furnace where a high temperature is maintained they give good service in exposed positions, but their application in exposed positions in periodic furnaces is limited owing to their low spalling resistance. The considerable progress which is being made in the manufacture of this type of refractory, however, indicates more extensive applications when the product is further improved. A short reference to the use of basic (dolomite) bricks in ladle lining has been made recently in a special report.[14]

Magnesite Refractories

The application and properties of magnesite refractories has been mentioned in thosechapters dealing with the basic steelmaking processes. These bricks consist of periclase (magnesia) with a matrix of forsterite ($2MgO . SiO_2$) and in some cases monticellite ($CaO . MgO . SiO_2$). It has been shown, however, that iron oxide can enter the periclase grains in solid solution or can form a magnesio-ferrite ($MgO . Fe_2O_3$).

Chesters[13] shows that the application of these bricks in exposed positions is limited by their relatively low spalling resistance, low refractoriness under load, high thermal conductivity and high cost. The cause of ultimate failure is "structural spalling". This spalling may be associated with crystal growth, but other explanations have been advanced. Chesters[13] suggests that shrinkage of the hot face due to a drawing together of the grains in the presence of slag may be a more likely cause of spalling. A brick which had an initial porosity of 25 per cent. may show a porosity as low as 8 per cent. after use. Some of this decrease is due to slag penetration, but some is due to shrinkage, which results in considerable strain. Frequently the junction between the slagged and unslagged portion of the brick is very sharp and this fact is suggested as a reason for spalling. This weakness may be overcome, however, by the employment of a brick with high initial bulk density and thermal shock resistance.

To some extent the specific gravity of magnesite bricks may be taken as an indication of the degree of burning. In the past a specific gravity of 3·65 was usually taken as a maximum value, but the present view is that this figure is somewhat high and should be modified according to the chemical analysis. A high iron oxide content would tend to increase this figure, whilst a high silica content would decrease the actual value.

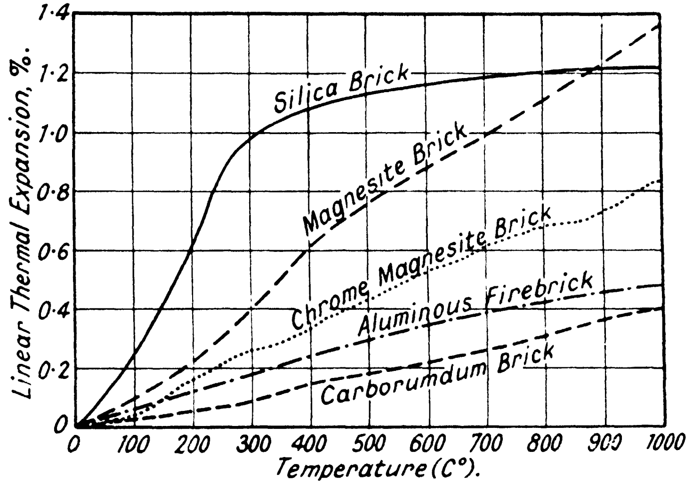

FIG. 218. Linear expansion of refractories.

The thermal expansion of magnesite refractories is shown in Fig. 218.

Chrome-Magnesite Refractories

The development of chrome-magnesite refractories has been mentioned in Chapter 6 page 140 in connection with the open-hearth furnace. The constitution of this type of refractory consists of a complex spinel (Fe . Mg) O, $(Cr . Al . Fe)_2O_3$ together with gangue material which is usually siliceous.

It was found that chrome-magnesite bricks prepared from certain ores were mechanically weak and oversize when drawn from the kiln. Experiments demonstrated that chrome ores are readily oxidized even at temperatures as low as 300 deg. C., the ferrous spinel chromite

$FeO . Cr_2O_3$ and the hercynite $FeO . Al_2O_3$ being altered to solid solutions of sesquioxides Fe_2O_3, Al_2O_3, Cr_2O_3. Under reducing conditions at temperatures over 650 deg. C. these solid solutions are reduced to chromite with a large volume change, which is often as much as 30 per cent.[15]

The ores which were liable to this friability all possessed a ferrous oxide content of over 18 per cent. and the tendency to friability can be reduced by selecting an ore with an iron oxide content of less than 15 per cent. It was also shown that the friability and growth could be reduced by increasing the amount of forsterite ($2MgO . SiO_2$) in the matrix by additions of serpentine or olivine.

One great difficulty with this type of refractory is the attainment of high thermal shock resistance coupled with a low bursting tendency in the presence of iron oxide. The best bricks from the aspect of spalling tend to possess an open texture and hence are more likely to absorb iron oxide and burst. Chesters[13] suggests that this bursting in the presence of iron oxide is probably due to the solid solution of iron oxide in the chromite:

$$\begin{pmatrix} MgO . Al_2O_3 \\ FeO . Cr_2O_3 \end{pmatrix} \text{ plus } FeO . Fe_2O_3 \text{ giving } \begin{pmatrix} MgO . Al_2O_3 \\ 2FeO . Cr_2O_3 \\ \quad Fe_2O_3 \end{pmatrix}$$

(Solid) (Liquid) (Solid)

Micrographs show that the chromite crystals grow as the mixed spinels form on the surface and diffuse inwards. This growth may result in a linear expansion of 20 per cent. and since the furnace structure cannot accommodate such expansion the face of the brick bursts.

This bursting in the presence of iron oxide, which is considerable even at 1,400 deg. C. and occurs as soon as iron oxide is absorbed, can be controlled by several means. It can be reduced by the addition of various materials and by regulation of the chrome/magnesite ratio. A considerable amount of research has been done on the properties of chrome-magnesite refractories by Hugill and Green,[16] Lynam and Chesters[17] and Chesters and Scott.[18] This work appears to confirm that the swelling is due to the solid solution of magnetite in the isomorphous chromite and that it can be controlled by the chrome/magnesite ratio. Rees,[3] discussing possible improvements in the properties of chrome-magnesite bricks, suggests a better chrome-magnesite balance and better grading to secure a lower permeability, which would result in less penetration and thereby increased durability.

Chesters[19] shows that high thermal shock resistance can be obtained by controlling the grading, the chrome fractions should be coarse, whilst the magnesite fractions should be fine in order to obtain the best results. The magnesite usually forms a forsterite bond ($2MgO . SiO_2$) with a high refractoriness, whilst the gangue associated with the chromite has generally a lower refractoriness. Most chrome-magnesite bricks show a high refractoriness under load compared with other basic bricks, but this quality is not as good as silica bricks shown in the maintained-temperature test.

The B.R.R.A. report[15] quotes the following range of analysis and properties for the pre-war Radex E brick, which gave very satisfactory results in certain parts of the open-hearth furnace construction:

Analysis—SiO_2	3·0 to 9·0	per cent.
Al_2O_3	2·8 to 14·6	,,
Fe_2O_3	10·1 to 17·3	,,
Cr_2O_3	30·2 to 36·0	,,
CaO	0·1 to 2·5	,,
MgO	34·6 to 44·5	,,

Refractoriness under load (28 lb./sq. in.) to fail point, 1,675 to 1,740 deg. C.

Apparent porosity 19 to 28 per cent., usually about 25 per cent.

Cold crushing strength, 2,120 to 3,550 lb./squ. in.

They recommend that the matrix of these bricks should be as near as possible to forsterite composition ($2MgO . SiO_2$), and that lime, which lowers the melting point of the matrix, should be kept low. The chrome ores employed should approximate 3 to 6 per cent. silica, under 15 per cent. ferrous oxide and under 1 per cent. lime. It is suggested for basic roofs that the alumina content should be fairly high so as to reduce the bursting expansion.

Basic bricks have proved very successful in the back and front walls of open-hearth furnaces and have had a limited application in the side walls of electric-arc furnaces as mentioned on page 235.

At many plants this type of refractory is employed for the construction of the ports of open-hearth furnaces. The "all-basic" furnace, mentioned on page 140, is still in the experimental stage both in this country and in America, but several interesting progress reports have appeared in the technical press,[15, 20, 21] to which the reader is referred.

It would appear that this type of furnace offers increased rates of output with a lower refractory consumption, but the overall economics have still to be assessed.

Chrome Refractories

Chrome bricks are frequently employed as the neutral course between acid and basic refractories. Ground chrome ore is sometimes used as a jointing material for chrome-magnesite bricks in open-hearth furnaces. It is also employed for fettling those parts of the banks where acid materials may come into contact with the dolomite lining. Typical analyses of chrome ores are given in Table LVIII.

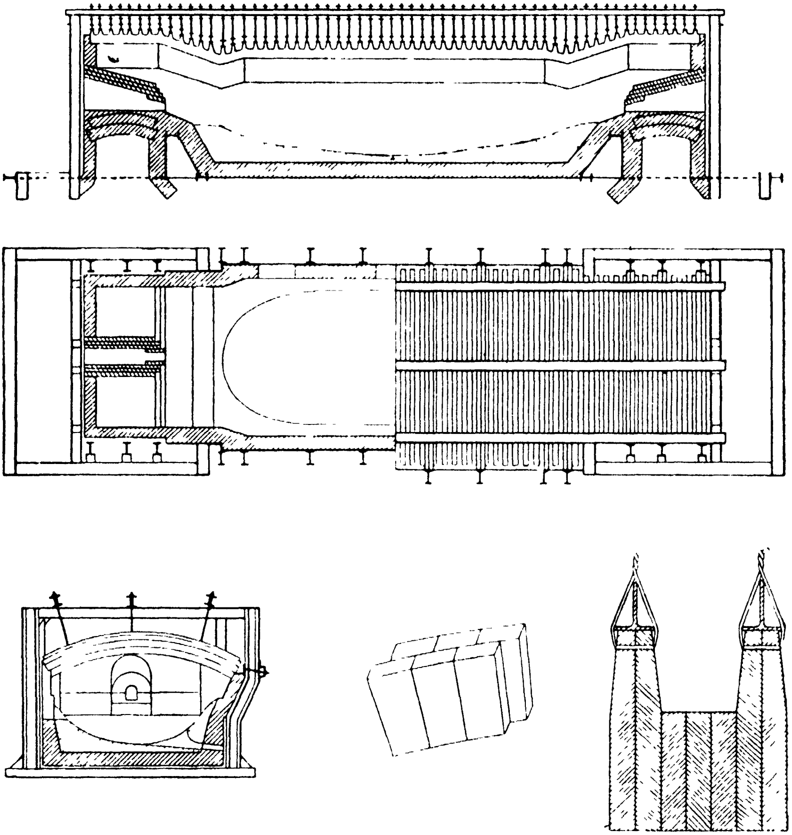

FIG. 219. The " all-basic " roof construction.

(By courtesy of the British Refractories Research Association.)

The chrome ores employed usually contain about 40 to 45 per cent. Cr_2O_3 with appreciable quantities of magnesia and alumina. They consist of a complex spinel of the type (Fe . Mg) O, $(Cr . Al)_2O_3$, which according to Dodd[22] comprises 75 to 90 per cent. of the ore. The remaining 10 to 25 per cent. consists of a silicate bond varying in composition from $3MgO . 2SiO_2$ to $2MgO . 3SiO_2$. The proportion of silicate present may seriously affect the softening point and should, therefore, be kept low.

Forsterite Bricks

This brick as the name implies consists essentially of the compound forsterite ($2MgO . SiO_2$), although some of the magnesia is usually replaced by ferrous oxide. These bricks have been sucessfully employed in the ports and end walls of open-hearth furnaces, but so far have not been successfully applied to furnace roofs.

They have a moderate spalling resistance and their resistance to iron-rich slag is about intermediate between silica and magnesite. They have a relatively low thermal conductivity and their thermal expansion between 20 and 1,000 deg. C. is about 1·0 per cent.

Sillimanite

Sillimanite has a limited application in steelworks, its chief employment being in the roofs of electric-arc furnaces where their performance was reviewed by Dufty.[23] The constitution of this type of refractory is shown in Table LIX. These bricks are prepared from either sillimanite, andalusite or kyanite, which approximate the composition $Al_2O_3 . SiO_2$. The final product on heating is mullite and crystobalite, the mullite needles can be seen in a siliceous ground-mass.

A reliable sillimanite brick will have the composition of 37 to 39 per cent. silica and 55 to 60 per cent. alumina. The iron oxide is usually about 1·0 per cent., whilst the titania and alkalis approximate about 1·5 per cent. An important feature of this type of refractory is its resistance to spalling, which recommends it for use in arc furnace roofs. It also possesses a low after-expansion at elevated temperatures. This after-expansion is frequently less than 0·04 per cent. per 100 deg. C. of temperature rise.

The porosity of hand-made sillimanite bricks is usually about 25 to 26 per cent., but these values can be lowered to 18 to 20 per cent. by suitable moulding and brickmaking. Although of little value, except for determining the porosity, the specific gravity varies from 2·8 to 3·0.

Cements

A frequent cause of failure in brick structures is the employment of unsuitable jointing material. Great care should be taken in this direction, as the ultimate strength of a structure depends on the refractoriness of the weakest member. The cements employed should be similar in nature and refractoriness to the bricks with which they are employed. For further information on this subject, however, the reader must be referred to the various works quoted or to standard theses on refractory materials.

Searle[24] suggests that dolomite bricks should be laid in a mixture of crushed dolomite or magnesite bricks and tar, or in a mixture of light magnesia with sufficient water to form a paste. It is probable that the main function of this jointing material is to afford an adequate bedding rather than bonding the bricks together. This bedding enables the bricks to adjust themselves to any pressure without fracture. The author has found that doloset cement, quoted on page 182, forms an excellent bedding mixture for dolomite bricks.

An excellent review of the various types of refractory materials employed at several steel plants prior to the war was given by Swinden and Chesters.[25] The correlation of this data shows that there was a general similarity at the different plants. In spite of the extensive research and development that has taken place in the manufacture of refractories and their application, there appears to have been little fundamental change in practice, except in the case of dolomite and chrome-magnesite qualities already mentioned in the text. This fact is largely due to the economics involved and the supply of the raw materials, much of which has to be imported.

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SUNDRY THERMAL DATA

Heat of Formation: (Clements: *Trans. Iron & Steel Inst.*, 1920, Vol. I).

SiO_2 per lb. of Si	12,600 B.T.U.
Fe_2O_3 ,, ,, Fe	3,143 B.T.U.
FeO ,, ,, Fe	2,112 B.T.U.
MnO ,, ,, Mn	2,976 B.T.U.
P_2O_5 ,, ,, P	10,600 B.T.U.
$FeSiO_3$,, ,, SiO	270 B.T.U
$FeSiO_3$,, ,, Fe	247 B.T.U.
CO_2 ,, ,, C	14,580 B.T.U
CO ,, ,, C	4,374 B.T.U.
CaS ,, ,, S	5,301 B.T.U.
CaO ,, ,, Ca	5,916 B.T U.
Fe_3C ,, ,, C	1,270 B.T.U.
$CaCO_3$,, ,, $CaCO_3$	812 B.T U.

Heat of Formation (Dichmann. *Basic Open-Hearth Process*, Constable).

1 kg. of Si on combustion to SiO_2	yields 7,830 cals.
1 ,, P ,, ,, P_2O_5	,, 5,965 cals.
1 ,, Mn ,, ,, MnO	,, 1,784 cals
1 ,, C ,, ,, CO	,, 2,387 cals.
1 ,, Fe ,, ,, FeO	,, 1,350 cals.
1 ,, SiO_2 forming $CaO . SiO_2$	,, 254 cals
1 ,, SiO_2 ,, $2CaO . SiO_2$	,, 384 cals
1 ,, SiO_2 ,, $(MnO)_3 (SiO_2)_2$	,, 134 cals.
1 ,, P_2O_5 ,, $2CaO . P_2O_3$	,, 1,131 cals

$2CaO + SiO_2$ giving $2CaO . SiO_2$ yields 550 B T.U per lb. of CaO.
$3CaO + P_2O_5$,, $3CaO . P_2O_5$,, 1,755 B.T.U. per lb of CaO.

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1 Kg. cal.	equals	3·968 B.T.U.
1 C.H.U.	,,	1·800 B T U
1 B T.U.	,,	0·252 Kg. cals.
1 Therm	,,	100,000 B.T.U.
1 kWhr.	,,	3,413 B.T.U.
1 H.P.H.	,,	2,546 B.T.U.

Heat required to melt various steelmaking components: (Leckie: *Trans. Iron & Steel Inst.*, 1944).

Cold pig iron and/or cast iron	10·0 Therms per ton
Steel Scrap	12·1 Therms per ton
Lime and Oxide	18·0 Therms per ton
Limestone	27·0 Therms per ton

For further thermochemical data and calculations reference is recommended to:

RICHARDS: *Metallurgical Calculations.* Two volumes. McGraw, 1907.

For Thermal Changes and Balance Sheet of the Basic Open-Hearth Process reference is recommended to:

" Basic Open-Hearth Steelmaking." Mudd Series. *A.I.M.E.*, 1944.

INDEX TO VOLUME TWO

Note. The figures in bold type indicate the main references.