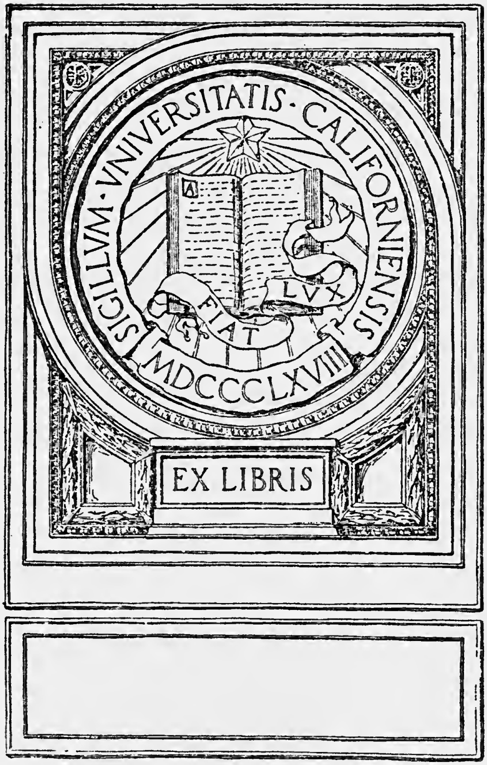

A TREATISE OF ELECTRO-CHEMISTRY
EDITED BY BERTRAM BLOUNT, F.I.C., ETC.

THE ELECTRO-METALLURGY OF STEEL

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THE ELECTRO-METALLURGY OF STEEL

BY

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WITH A PREFACE BY DONALD F. CAMPBELL

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PREFACE.

ALTHOUGH electro-metallurgy is still in the early stages of development, no apology is needed for the devotion of a volume to the application of electricity to the melting and treatment of steel.

The electric furnace gives us a new atmosphere, in which steel-making can be accomplished. Purification of miscellaneous qualities of steel containing noxious substances can be controlled with great exactitude. Scientific synthesis of complex steels by the addition of many valuable elements to pure metallic iron under accurate chemical control has replaced the old methods which are sometimes suggestive of the mysterious practices of the alchemists.

The melting of steel in a truly reducing atmosphere is only possible with electric heat, and, consequently, new phenomena are observed and often striking results obtained. Some of these we understand, or suppose we do, but others must be the subject of further research.

The electric furnace has given us at reasonable prices new materials of special value to the steel-maker, such as low carbon ferro-chrome and high grade ferro-silicon, which, in turn, have been economically made into such products as stainless steel and transformer iron. The former product is being used by engineers for remarkable new applications, and the latter has increased the efficiency of electric transformers to an extent which represents an annual saving of hundreds of thousands of tons of coal per annum.

The electric furnace came as a corollary to the construction of the first dynamos, and although only twenty years old, it is already absorbing millions of electrical horse-power for various purposes, and has produced more than a million tons of steel.

The widest application of this process in the immediate future will probably be for the treatment of the phosphoric ores of Alsace-Lorraine, where the process of Thomas and Gilchrist will be supplemented by electric refining. The stringent requirements of modern engineering will then be met by electrically refined steel of the highest quality, and the valuable phosphates will be rendered available for the great industry of agriculture. Other ores containing insufficient phosphorus for the basic bessemer process may soon be smelted with phosphoric limestone, and similarly treated, so that the phosphorus now wasted may be recovered and returned to the soil.

To the electrician, the problem of electric furnace design and operation is of absorbing interest. With currents of 20,000 ampères or more, phenomena of reactance, eddy currents, and skin effect have involved new problems only recently recognised, and as yet not fully appreciated.

The author has written a book which should attract both the student and the practical steel-maker, as his scientific attainments and wide experience of steelworks in several European countries enable him to appreciate the difficulties of those who have to deal with the electro-metallurgy of steel both in its theoretical and in its practical aspects.

The volume gives the early history of a new branch of metallurgical science, which is surrounded by many fascinating problems and great promise of new achievement, while a clear statement is made of our present knowledge of the use of electric furnaces for the metallurgy of steel.

DONALD F. CAMPBELL.

INTRODUCTION.

The electro-metallurgy of steel as an industrial science owes its present status to the vast development of the past few years. Prior to the outbreak of the Great War, the electric furnace had only a very limited application in Great Britain and other countries. The shortage of high grade raw materials, the enormous demand for alloy steels, and the vast accumulation of heavy steel turnings presented exceptional opportunities for the electric furnace, and it was only then that its economic advantages in certain branches of steel-making began to be generally recognised. Since that time various modifications have been made in the manner of utilising electric energy in arc furnaces, and a far clearer and wider understanding of the chemical reactions peculiar to the processes of steel-making in these furnaces has also resulted.

The induction furnace at one time certainly received greater prominence than the arc type, although it is doubtful whether it ever had a corresponding advantage in actual output of steel. Many furnace designers, in their belief that the principle of induction heating had superior advantages, concentrated their efforts on the production of a furnace which could operate on any standard electric supply system, and, at the same time, meet all the requirements of the steel-maker. As a result, numerous types were introduced, and considerable publicity given to their construction and operation.

In this book, the electro-thermic processes of steel-making have been studied more especially in their relation to arc furnaces, from which almost the entire output of electric steel is now being made. Little space has, therefore, been devoted to the construction and operation of induction furnaces, although sufficient has been written to indicate their electrical, constructional, and metallurgical features and the principles involved.

In the earliest days of crucible steel-making, the importance of the character of the fuel used, the method of heat application and temperature control were fully recognised and studied by the steel-maker; precisely the same consideration is also given to these factors now by those responsible for the manufacture of steel in open-hearth gas furnaces. It is obvious that a full knowledge of both the chemical and physical conditions, which constitute a thermo-chemical process for the production of a metal, should be within the province of the metallurgical engineer, and there is no reason why any such process conducted by electro-thermic means should be an exception. Naturally, it may be thought that the application of electric energy to either induction or arc heating is hardly comparable to that of solid or gaseous fuel, and demands the study of a special branch of electrical engineering. This is certainly true from the point of view of electric furnace design, but does not properly apply to the metallurgist, who will only require a general knowledge of the electric fuel provided. The metallurgist best knows the thermal and chemical conditions which conduce to maximum economy and technical efficiency, and should be in a position to judge whether the electric energy is being utilised in the most suitable manner.

The chemical and electrical conditions are always mutually dependent upon one another, and, without a knowledge of the general principles underlying the use of alternating currents for arc heating, it is impossible to know whether a slight adjustment of the electrical or the chemical conditions is required to remedy an unsatisfactory state of either. The problem is, therefore, more complex than for those processes in which the thermal conditions are quite uninfluenced by the chemical operations.

The general principles and application of alternating currents have been very briefly described in Chapters II.-IV., and every endeavour has been made to present this introductory study in a manner comprehensible to those without any mathematical or other special knowledge of the subject. This attempt probably falls far short of its object, and may receive criticism from those well acquainted with this special branch of electrical engineering. However, a little knowledge is not always dangerous, particularly if it should enable the cause of certain phenomena to be correctly interpreted and discussed with those who are responsible for effecting the remedy. Factors bearing upon the cost and economic use of electric energy have been dealt with in Chapter VI., and it is hoped that the issues raised will indicate how the maximum economy of power cost may be attained.

It has been impossible to give any definite figures bearing upon the cost of steel production as governed by factors other than the actual cost of power, raw materials, and labour. It has been emphasised throughout this book that such all-important factors as power-consumption, life of linings, and electrode consumption are not only all mutually interdependent for any given type of furnace, but are absolutely controlled by the furnace load factor and the precise nature of the process used. For these reasons, even comparative costs are dangerous and misleading without stating the exact conditions of operation in each case, and it is, therefore, far better to ascertain such figures under the exact working conditions provided for from any of the numerous sources available.

The author has to express his great indebtedness to Mr. R. P. Abel for the unstinted assistance he has given in connection with the electrical chapters, while his thanks are also due to Messrs. Thwaites Bros. for affording every facility to investigate their plant, and for providing several of the illustrations; to Messrs. B. Drinkwater and J. Holden for their assistance in compiling the Appendix; to those constructors of electric furnaces who have kindly placed at his disposal information and drawings relating to their operation and design; and to numerous friends who have given advice and information.

The majority of the illustrations are original, and permission to reproduce many of the remainder was kindly granted by The Iron and Steel Institute, The Faraday Society, The American Electro-chemical Society, and the Canadian Government Commission.

Lastly, the author desires to express his grateful acknowledgment to Mr. D. F. Campbell for contributing the Preface.

C. C. GOW.

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CHAPTER I.

HISTORICAL DEVELOPMENT OF ELECTRIC FURNACES.

THE Electro-metallurgy of steel is now an applied science of considerable industrial importance, and has only reached its present stage of development during comparatively recent years. The application of electric energy for melting steel had been demonstrated by Sir William Siemens almost twenty years before its commercial possibilities were recognised by later investigators, to whose work the present status of the electric steel industry is primarily due.

The generation of electric currents by means of the dynamo or similar device was only discovered in 1867, so that the slow advancement made subsequent to the work of Siemens was no doubt mainly due to the lack of thoroughly reliable electrical equipment, without which any electro-chemical process, though conducted with the best technical and organising skill, is doomed to commercial failure.

The use of electrical heating for melting steel on a commercial scale was first demonstrated by the induction furnace, but no real impetus was given to the industry until the introduction of the arc furnace a few years later. For several succeeding years the electro-thermic processes were limited to localities peculiarly favourable to economy of power production and to a few large undertakings, which, at the instigation of the inventors, constructed larger units for the further study of the products and cost of production.

Before investigating further the development of the electric steel furnace at this period, the work of Sir William Siemens in 1882 must be more closely considered. The melting of steel was accomplished in a small crucible furnace in which the principles of both direct and indirect arc heating were utilised and first established; Figs. 1 and 2 illustrate the two arrangements used. In the case of the former (Fig. 1), the furnace charge serves as one electrode, and is connected to the source of electrical supply through a carbon or metallic pole penetrating the furnace bottom or hearth. The electric current enters the furnace through an upper hanging electrode, and strikes an arc at the point of the conducting charge where contact is made. The principle involved is similar to that embodied in several modern types of furnaces. The furnace represented in Fig. 2 is dependent on indirect arc heating, and is furnished with horizontal electrodes which pass through the walls. The arc is struck between the ends of the electrodes, and the furnace charge receives heat by radiation from the arc and reflection from the roof and walls. Here, again, the principle is identical with that employed in another class of modern electric furnaces of which the Stassano furnace was the forerunner.

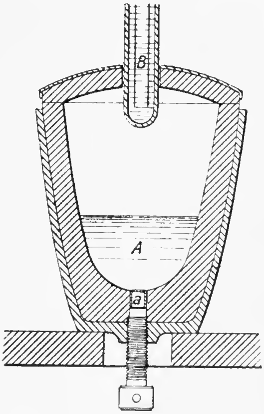

Fig. 1.—Siemens' direct arc furnace.

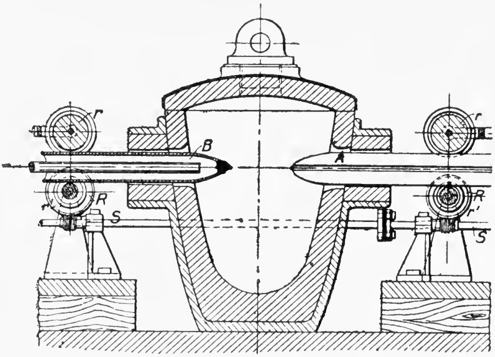

Fig. 2.—Siemens' indirect arc furnace.

To Sir William Siemens, therefore, belongs the credit of having first demonstrated two systems of electric arc heating as applied to steel melting, and in this respect he may be considered the pioneer of the industry. His researches were certainly confined to the laboratory scale, but it is more than probable that, had the requisite electrical plant been available at that time, his efforts would have been further extended to the commercial application of his discovery.

A period of inactivity in the development of arc heating followed until 1898, but during that time other discoveries were made which demonstrated the possibility of melting metals by the heating effect of low voltage currents induced in the metal itself. The electrical principles involved in the operation of induction furnaces are dealt with in Chapter II.

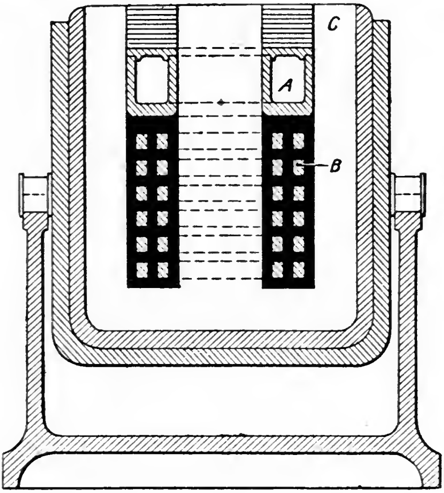

Fig. 3.—Ferranti induction furnace.

Ferranti[1] in 1887 constructed a furnace on this principle in which he was able to melt metals, although no special claim was made for melting steel. The principle of induction heating was later adopted in various furnaces which at one time promised to bear an important part in electric steel melting, but which have now been almost entirely abandoned in favour of arc furnaces.

The melting chamber of Ferranti's furnace (Figs. 3 and 4) consists of an annular channel A, provided with a suitable cover-

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[1] British Patent Specification, No. 700, 1887.

ing, surrounding the centre limb of a double closed magnetic circuit C. Primary windings B are also wound round the same limb immediately below the melting chamber. The apparatus consists essentially of a simple transformer, whose secondary winding of one short-circuited coil is composed of a ring of the metal to be melted. The voltage of the induced current is exceedingly low, owing to the low resistance of the secondary circuit, and the intensity of the current correspondingly high. The transformer core, consisting of laminated sheets of iron, is mounted in a framework which tilts on trunnions, so that the molten metal can be easily poured. Three years later Ferranti was followed by Edward Colby,[1] who took out several patents in America for melting metals in induction furnaces. In his original type the primary winding surrounded the secondary or annular melting chamber, but this arrangement was later given up and their respective positions reversed. Means were also provided for tilting the furnace and pouring the metal direct. At least eight years elapsed before the first steel was made (1898-1899) in an induction furnace by Colby and Dr. Waldo in America, and

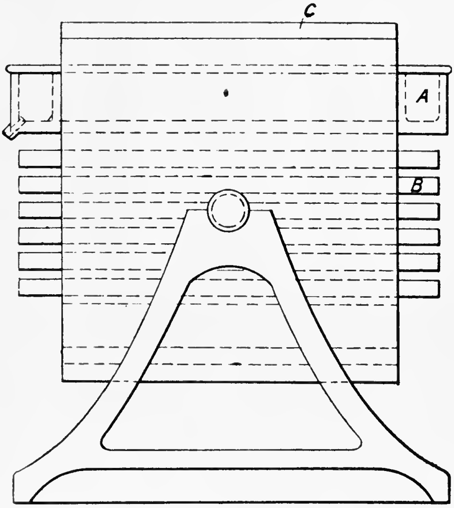

Fig. 4.—Ferranti induction furnace.

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[1] "El. Chem. Industry," Vol. V, p. 55; U.S.A. Patent Specification, No. 428,378, 428,379, and 428,552.

although several of these furnaces were later installed for the production of electric steel, the introduction of modified types soon gained prominence. One of the distinctive features of the later Colby furnace[1] (Fig. 5) is the double magnetic circuit comprised of three vertical legs connected together by horizontal members. Both the primary and secondary circuits surrounded the centre leg and were themselves enclosed between the two outer legs. The primary windings, it will be seen, are as close as possible to the centre core, and, by means of suitable water cooling arrangements, the secondary circuit or melting chamber could be constructed in close proximity to it, thus reducing magnetic leakage to a minimum and increasing the power factor. This was undoubtedly an excellent feature, but the difficulty of embodying this construction in larger furnaces (i.e. above 3 cwts.) was considerable on account of the high-voltage current necessary for the supply of increased power to the primary windings, which would then require elaborate insulation. The power factor of the furnace is given by Colby as about ·9, which compares favourably with modern arc furnaces.

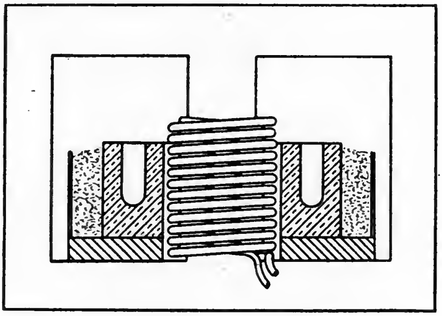

Fig. 5.—Colby induction furnace.

Colby's induction furnace, as actually applied to steel-making, was patented in America in 1900[2] or at least one year after his first production of steel. The same year was marked by further improvements in induction furnaces, and, what is far more important, by the re-introduction of arc heating, both direct and indirect, for steel melting. Thus the year 1900 may be regarded

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[1] U.S.A. Patent Specification, No. 859,641 (1907).

[2] "El. Chem. Industry," Vol. III, 1905, pp. 80, 134, 299, and 341, also Vol. V, 1907, p. 232.

as the most important in the history of the electric steel industry. To avoid confusion the further development of the induction furnace will be first considered, returning later to the history of arc furnaces, which in their commercial form date from this year.

The Kjellin induction furnace, first patented in 1900 and operating in 1902, embodied the same electrical principles as used by Ferranti and Colby, the salient difference lying in the use of high-voltage current in the primary windings for the purpose of increasing the furnace power capacity. One of the magnetic circuits was also eliminated, leaving of the three vertical legs the centre core and one outer leg, which were set in the plane of tilting. Figures 6 and 7 show a diagrammatic plan and section of the fixed furnace in operation at Gysinge in Sweden,

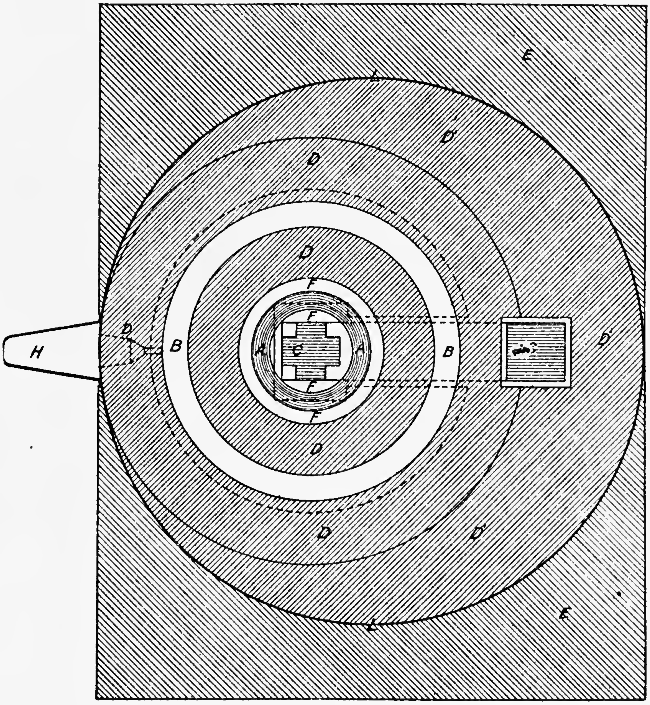

Fig. 6.—Section AB.

which was reported upon by the Canadian Government Commission in February, 1903. The melting chamber B, when charged with metal, constitutes a short-circuited secondary winding of one turn, and surrounds the primary windings A and one limb of the iron core C. The primary windings being carefully insulated were supplied with a single-phase current of 90 amperes at 3000 volts and 15 cycles, representing about 168 kw. at a power factor of ·62. They were further protected from the heat radiated from the melting chamber by means of water circulation supplementing natural air cooling. The melting chamber was closed in by a number of small cover bricks KK, which could be easily removed for purposes of charging, repairing and inspection, and was provided with a lining composed of silica or magnesite bricks filling the portion marked D set on a foundation of firebricks D^1. Sufficient metal was always left in the furnace after tapping to provide an unbroken secondary circuit for the induced current when starting a subsequent heat.

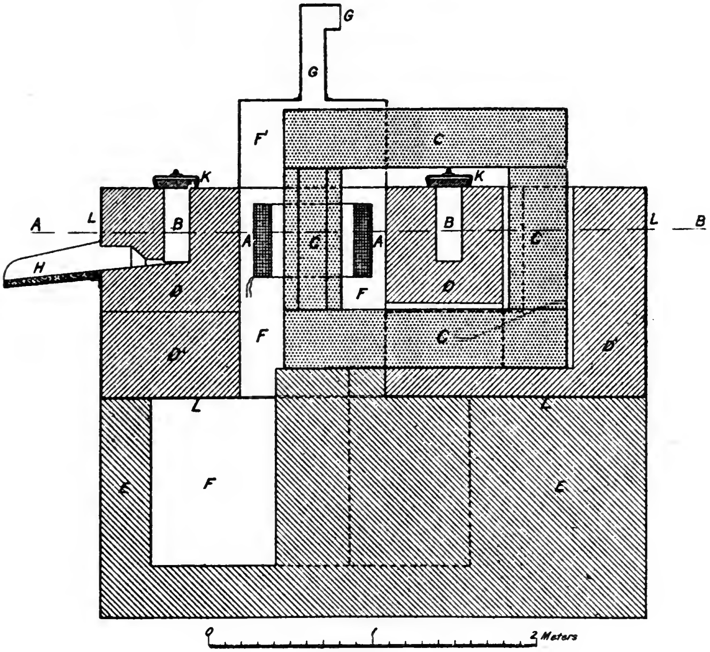

FIG. 7.

The radiation loss of the furnace mentioned was equivalent to 80 kw., leaving only 88 kw. for melting, and with such a ratio the power consumption could not possibly have been anything but high. The furnace was chiefly used commercially for melting Swedish white iron and steel scrap requiring no refining treatment. In this respect any advantage of the induction furnace over crucible melting lay solely in economy of heat utilisation.

The later type of Kjellin furnace was adapted for tilting and is illustrated in Fig. 8. The low frequency required necessitated the use of special generating plant, and consequently increased the cost of production by the heavier capital outlay involved. The power factor of the furnace was unavoidably low, being about ·6 to ·65, and fell still further on increasing the furnace capacity. For this reason a lower frequency than 15 was necessary for the larger units. In 1909 there were in operation only ten Kjellin and one Colby furnace, so that ten years after the introduction of the induction furnace no considerable progress had been made. The first Kjellin furnace to operate in England was erected by Vickers Sons & Maxim and began operation in 1908, but this did not lead to further development.

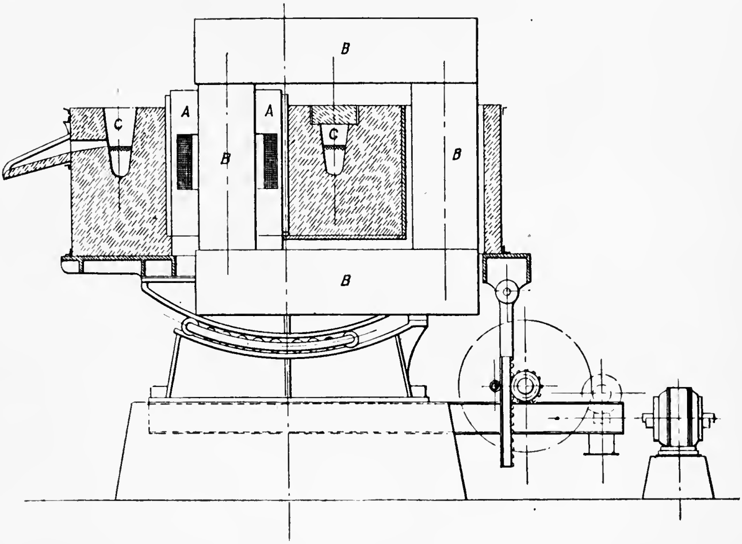

Fig. 8.—Kjellin tilting furnace.

Although at that time considerable advances had been made in other countries with induction and more especially arc furnaces, this Kjellin furnace was the first electric furnace plant to be installed in the United Kingdom. A fixed furnace[1] of 1 ton capacity, operating at Gysinge, produced 950 tons of tool steel and special steel ingots during the year 1906. The bulk of the output was produced from charges composed of about 80 per cent. Swedish white iron and 20 per cent. of steel scrap; briquettes of ore were added to regulate the percentage of carbon in the steel cast. The energy consumption[2] was normally 1128 kw. hours per ton, but when white iron and steel scrap were melted without briquettes the power consumption fell to 886 units per ton.

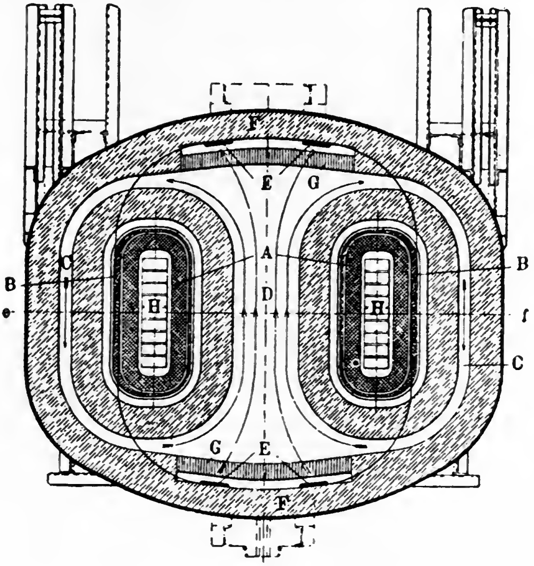

FIG. 9.

A distinct departure from the design of Kjellin was made in the Röchling-Rodenhauser furnace, which was patented in May, 1906, and designed to operate with either single or three-phase current. In each case the iron cores and primary windings were enclosed by a secondary circuit. Auxiliary secondary windings,

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[1] "Iron and Steel Institute Journal," Vol. III, 1906, p. 397.

[2] *Ibid.*, Vol. I, 1909, p. 296.

consisting of a few turns of heavy copper bar, were connected to terminal plates embedded in the furnace lining, and provided a further source of heat to the furnace charge, besides improving the electrical efficiency. Figs. 9 and 10[1] show a sectional plan and elevation of the single-phase furnace, and Figs. 11 and 12 corresponding views of the three-phase design.

Two furnaces, one of which was an 8-ton single-phase furnace and the other a 2-ton three-phase furnace, were being operated at the Röchling Iron and Steelworks at Volklingen in 1909. The 8-ton furnace of 600 kw. capacity was supplied with power from a 4000 to 5000 volt 5 cycle per second generator, while the smaller three-phase furnace of 250 kw. capacity operated on a supply of 400 volts at 50 cycles. The higher periodicity permissible with the three-phase furnace results from (*a*) its smaller capacity, (*b*) the higher power factor obtainable by the closer and better arrangement of the transformer cores relative to the primary and secondary windings, (*c*) the auxiliary secondary windings which reduce magnetic leakage.

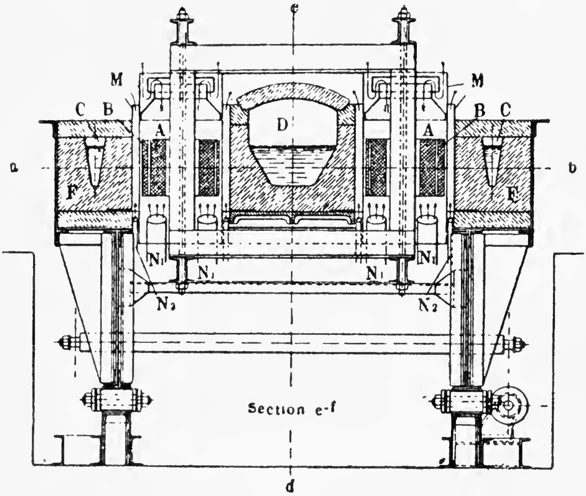

Fig. 10.

In both furnaces the annular melting chambers meet in a common hearth of wider proportions, situated between the vertical legs of the iron core, and it is here that the heat is

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[1] "Iron and Steel Institute Journal," Vol. I, 1909, p. 270.

intended to be absorbed by the current generated in the auxiliary

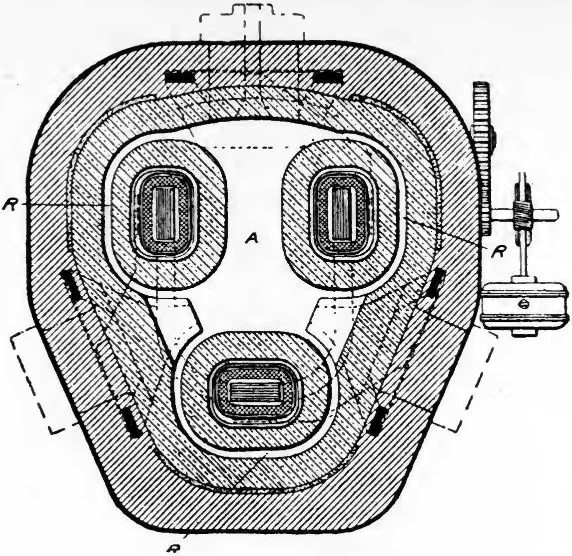

Fig. 11.

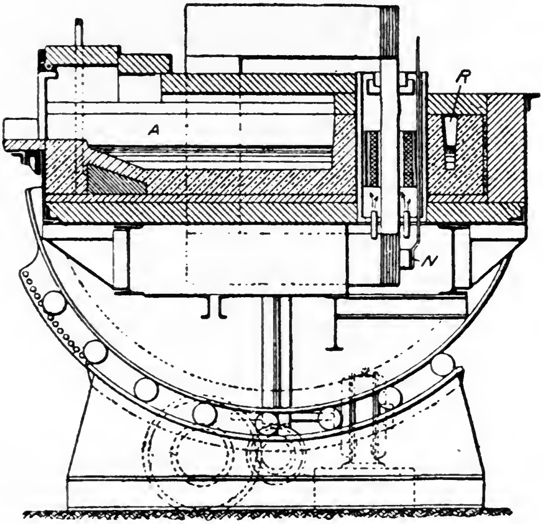

Fig. 12.

secondary windings. The increased dimensions of this hearth enabled certain metallurgical operations to be performed which had hitherto been impossible in the older forms of induction furnaces. The central chamber is arched over (Figs. 10 and 12), and provided with a charging door and spout. Fluxing materials can thus be easily charged and the bulk of slag removed if desired. The greater part of the heat is generated in the narrowed section of the secondary channels, and transmitted to the metal in the centre portion by circulation. The auxiliary heating already mentioned is of doubtful advantage, and it is not clear how the bulk of this heat can be generated anywhere but in the refractory lining separating the terminal plates from the metal. The circulation of metal along the narrow portion of the secondary circuit is promoted by electromagnetic effects in a fixed direction, and maintains a sufficiently high temperature in the centre portion of the bath to promote

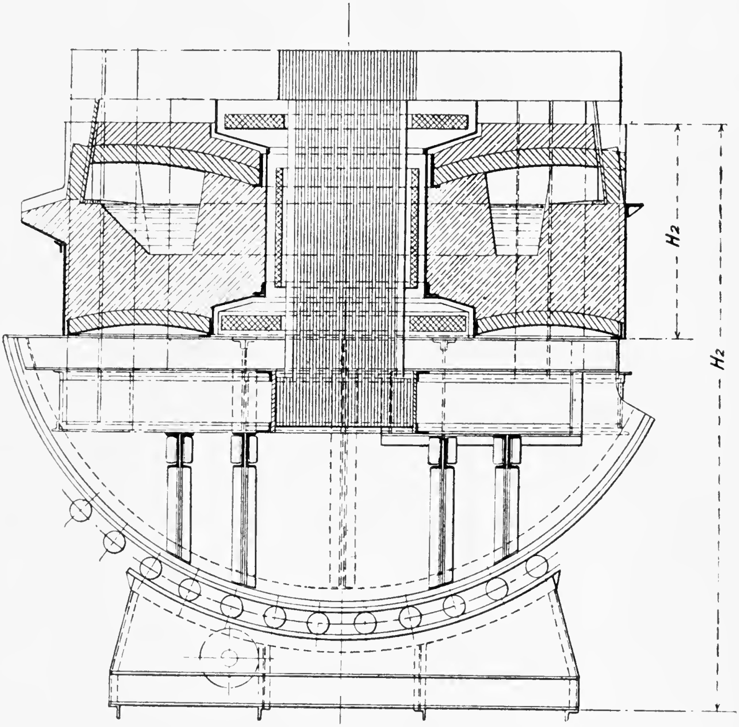

FIG. 13.

slag reaction and for casting. According to Rodenhauser the single-phase furnace, when under three tons capacity, is superior as regards efficiency to the polyphase, while for larger sizes the polyphase type is preferred. This is mainly due to the power factor of the single-phase furnace becoming increasingly lower as the distance of the secondary melting chamber from the iron core becomes greater.

From a metallurgical standpoint, the "Combination" furnace, as it was called by Rodenhauser, was a material advance on the simple induction furnace; not only could pure materials be melted and poured, but the construction of the hearth, as previously explained, permitted the removal of phosphorus and sulphur from less pure classes of scrap. In the year 1913[1] there were thirteen Röchling-Rodenhauser furnaces actually in operation, ten of which were employed for refining molten steel or pig-iron, and the remaining three for melting and refining cold charges.

Two years after the introduction of the Röchling-Rodenhauser furnace another induction type furnace, designed by Frick,[1] was operating at Krupp's Essen works with a power input of 750 kw. The single-phase furnace, as illustrated in Figs. 13, 14, and 15, has certain features in which it resembles both the Colby and Kjellin furnaces. A three-legged, laminated iron core is adopted, and the primary windings are designed for high-voltage currents. The peculiar construction of the primary windings was intended to improve the power factor by reducing magnetic leakage, and to effect this object the coils were split up into three portions, the top and bottom coils being flattened out to enclose the secondary circuit. This was commonly known as the "Umbrella" design. Side inspection doors were also provided in place of the removable cover bricks of the earliest Kjellin pattern. Frick, in his paper read before the Iron and Steel Institute in 1913, describes at considerable length the electrical and metallurgical operation of his furnace, but does not indicate any considerable improvement in power factor resulting from the special arrangement of the primary windings. The annular melting chamber,

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[1] "Iron and Steel Institute Journal," Vol. II, 1913, pp. 297 *et seq.*

necessarily steep-sided, was composed of a very pure magnesite treated in such a way as to resist contraction and expansion at high temperatures. This lining enabled charges to be melted down under basic slags for purposes of phosphorus removal. Further, a rotatable cover was provided for the melting chamber to facilitate uniform charging and repairing.

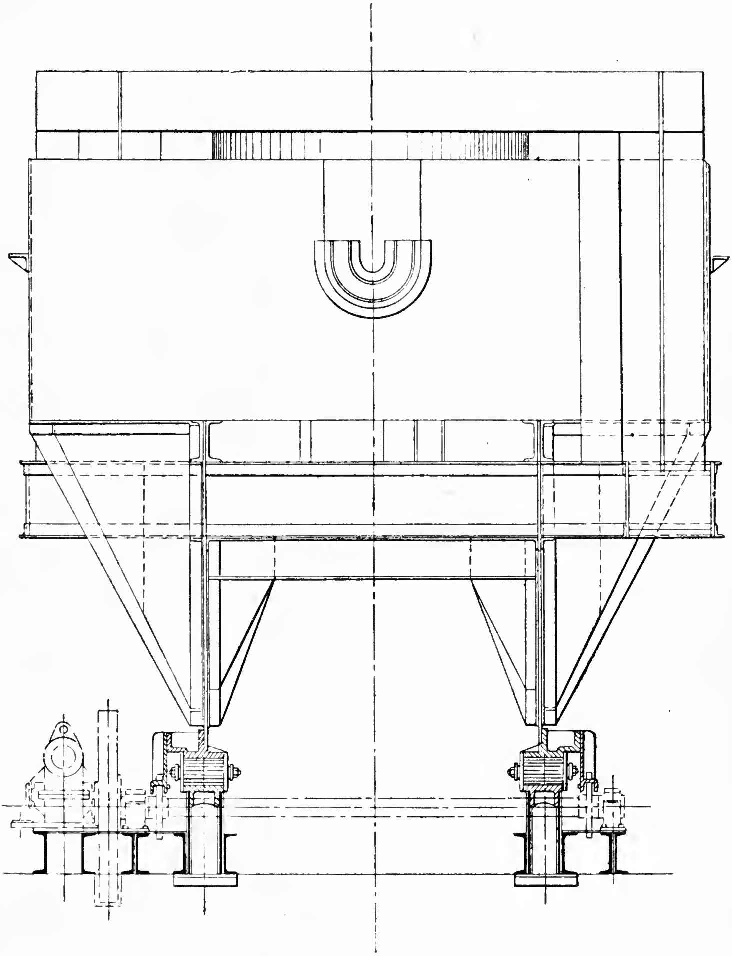

Fig. 14.—Frick induction furnace.

There now only remains to be mentioned the Hiorth[1] furnace which was patented in Norway in May, 1909, and put into operation early in 1910. J. W. Richards, in a paper read before the American Electro-chemical Society in 1910, describes a 5-ton capacity unit supplied with 400-500 kw. at 250 volts, 12½ cycles. The furnace (Fig. 16) resembles the Rodenhauser arrangement of a double secondary circuit joining to form a central hearth, and embodies also the "Umbrella" type of primary winding used by Frick. In this case, however, the primary voltage is low, and to enable the heavier current to be carried at greater current density the primary coils are constructed of copper tubing for water circulation. During the heat made on the occasion of J. W. Richard's visit to the Jossingford works, the power factor varied from ·57 to ·8. A

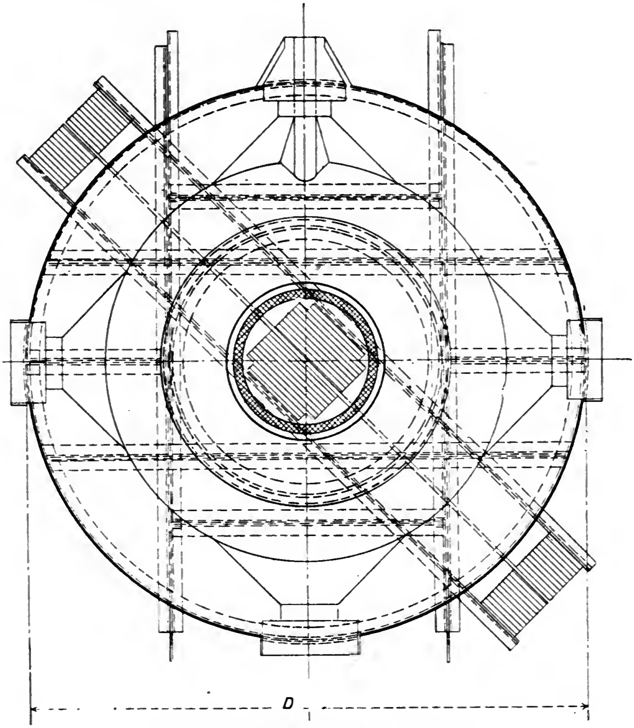

Fig. 15.—Frick induction furnace.

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[1] "Am. Electro-Chem. Soc.," Vol. XVIII, 1910, p. 191.

novel feature of the furnace construction enables the furnace body to be tilted for pouring without movement of the iron core and upper set of primary coils; the lower set, like that in the Frick furnace, is attached to the furnace bottom and moves with it.

The development of the induction furnace shows that considerable inventive skill has been concentrated on attempts to improve its electrical features, and to render it a useful appliance in the hands of the steel-maker. Unfortunately the fundamental elements of an induction furnace are adverse to the flexibility and economy of operation which are essential to the production of electric steel as a commercial commodity. It is therefore not surprising that arc furnaces by their simultaneous development have practically supplanted the induction furnace for the melting and refining of steel. The induction furnaces mentioned in this survey are only the better-known types that have been used commercially.

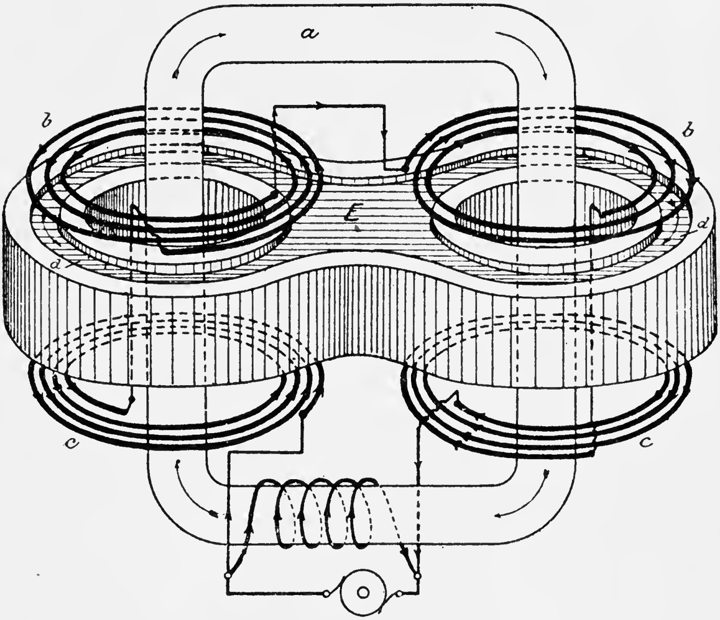

Fig. 16.

On returning to the history of arc furnace development a brief description of each type will be sufficient to indicate the material points of difference, a detailed description of those types now in extensive use being given later.

Following the publication of Sir William Siemens' work with arc furnaces, sixteen years elapsed before the problem of melting steel by arc heating was again attacked. In February, 1899, Stassano made known the results of experiments on the reduction of iron ores by electro-thermic means conducted during the preceding year. His experiments were confined to the reduction of ore to metal, which he proposed to further refine in the same furnace for removal of carbon when necessary.

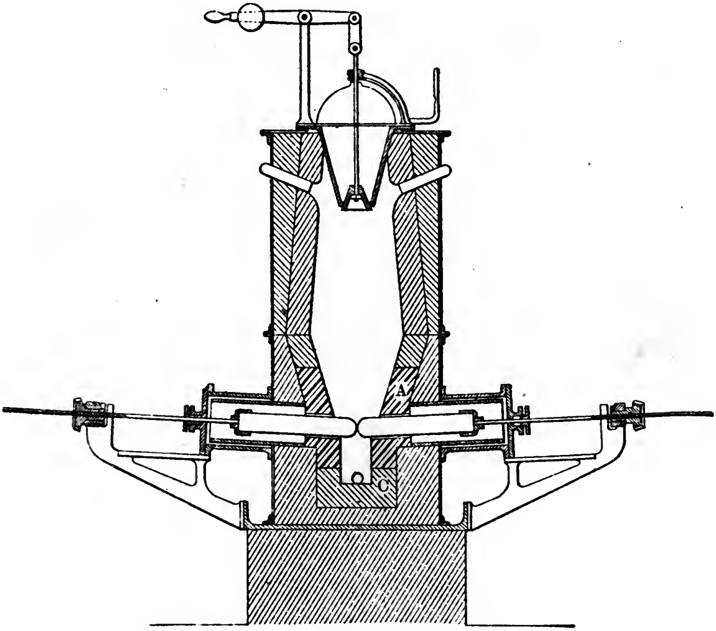

FIG. 17.

His first furnace, built in 1898 (Fig. 17), clearly resembles that type of Siemens' electric crucible furnace which embodied the principle of indirect arc heating. The furnace was built by Stassano in Rome and used solely for experiments on the reduction of iron ore. A mixed charge consisting of briquettes was supported in the basic lined shaft A by an iron grating fixed 8 inches above the arc zone. The reduced metal trickled through the unreduced charge into the crucible C, from whence it was tapped at intervals. The slag, however, was not sufficiently fusible, and remained as a solid arch above the arc zone, preventing further reduction of the charge above.

This serious difficulty led to a modified form (Fig. 18) in which the mixed charge was introduced below the arc zone instead of above it, and in this way the difficulties of maintaining a continuous reaction were overcome. It was found, however, that during the first stages of reduction the power required was much greater than during the final stages when the bulk of the metal and slag had been fused and melted. Owing to the impossibility of reducing the load without forming a long and unstable arc, Stassano was later forced to modify this furnace by using two or three pairs of electrodes in place of one pair (Fig. 19); the furnace load could then be varied within certain limits by extinguishing one or more arcs. The reduction from full to low load was at that time objectionable to the generating station, and it was then suggested that, by the use of two furnaces always operating at different stages of the reduction process, the mean load taken might approach a more constant figure. The plurality of electrodes and the excessive difficulty of operating the two furnaces so as to require at all times a constant generating station load soon showed the impracticability of this arrangement.

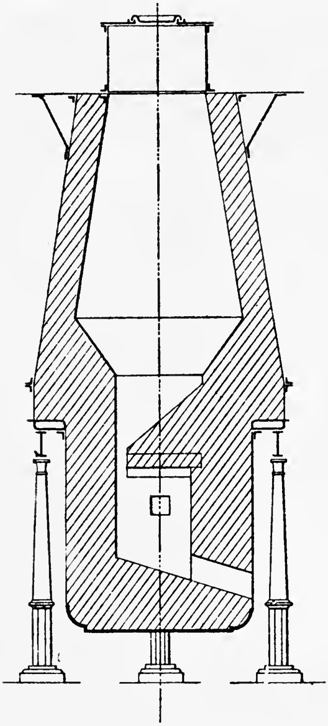
Fig. 18.

Stassano then argued that, if the period when the minimum power is required could be shortened by rapid treatment and removal of the metallic mass during the reducing fusion of the charge, the furnace might be continuously operated at a higher and more uniform load. He had also observed that increased economy was gained, both in time and fuel, by using a furnace with a movable hearth for refining pig-iron, and thereupon conceived the idea of embodying this principle in his furnace to overcome his difficulties.

This was the origin of the Stassano furnace as later used for melting steel scrap, and which in its first form was patented in 1902. In 1903 Dr. Goldschmidt, reporting on behalf of the German Patent Office, stated that mild steel could be successfully made from pure ores, but that the process was too expensive for economic competition.

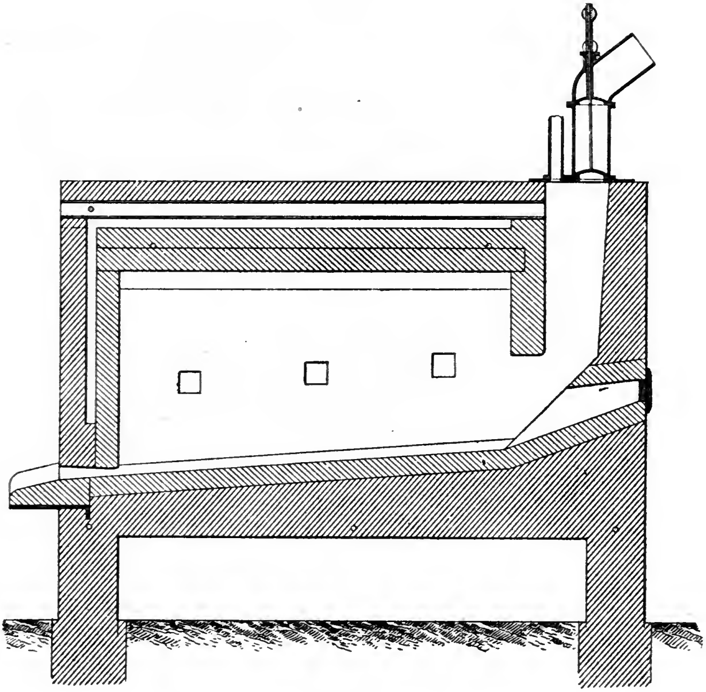

Fig. 19.

Dr. Haanel's report of 1904 contains information communicated by Stassano relative to a furnace of 1000 horse-power rating. The output was given as 4 to 5 tons per day according to the quality of ore used. The furnace was provided with two pairs of electrodes in parallel, each pair supplying an arc with 2450 amperes at 150 volts. Apart from the usual fettling, the lining was estimated to last at least forty days without repair. Later the rotating furnace was further modified to meet the requirements of melting mixed steel and iron scrap, and thus the production of mild steel or iron by direct reduction of ore gave place to simple melting and refining.

While Stassano was endeavouring to apply electric heating to the more primitive method of steel-making by direct reduction from ore, Dr. Heroult, in the year 1899, began his extensive work on the production of steel from pig-iron, and later from common scrap-iron and steel. He saw more prospect of success by following the lines of a well-established process in general use, which consisted of refining pig-iron and scrap in the basic open hearth furnace, and succeeded in solving the problem of carbon contamination by interposing a layer of slag between the bath of metal and the electrodes, and further by the use of a non-carbonaceous lining. This slag covering, which might be varied in composition, had the added advantage of effecting removal of injurious impurities from the bath. The development of the series arc design was both interesting and logical, and entirely fulfilled the expectations of the inventor.

In the year 1899 Heroult was using a single electrode furnace, resembling in principle the direct arc crucible furnace of Siemens, for the manufacture of ferro-alloys. The furnace, lined throughout with carbon, was supplied with current by two electrodes, one embedded in the bottom and the other hanging vertically so as to complete the circuit through the conductive charge. The carbon-lined bottom was satisfactory for the production of high carbon ferro-alloys, but with a growing demand for a low carbon ferro-chrome it had to be abandoned. As a substitute Heroult first employed a bottom composed of chromite ore, surrounding a single carbon pole placed centrally. He expected that the carbon would eventually be consumed, and replaced by metal to a depth where the temperature was sufficiently low to prevent further absorption. This bottom carbon electrode was, however, found unsatisfactory, and the logical conclusion of supplying power to the upper portion of the conductive charge followed, this being accomplished by substituting another vertical top electrode for the one embedded in the bottom. In this way the Heroult steel furnace provided with two hanging electrodes and two arcs in series originated.

The principle, then, of direct arc heating as first used by Siemens is followed, but with the one all-important difference that the electrodes carrying current to the furnace both enter from above the charge and so eliminate all electric connections made with the furnace hearth. Finding the new type of furnace successful for melting low carbon alloys having a high melting-point, Heroult then turned to the problem of melting and refining steel.

The furnace lining could be made similar to that commonly used in the Siemens-Martin furnace, so that there seemed little reason why, given sufficient temperature, the same metallurgical results should not be achieved; not only was this the case but it was soon found that this form of furnace possessed important features which permitted refining operations to be carried far beyond anything hitherto possible in the open-hearth furnace.

Up to the year 1904 nine distinct patents, covering novelties of furnace design and metallurgical operations peculiar to the electric furnace, were issued in different countries throughout the world.

To show how thoroughly the investigations were carried out, and how far the value of the hitherto unknown chemical reactions was understood and appreciated, it is only necessary to quote the titles of the French patents enumerated below:—

I. **No. 298,656.** Nineteen patents issued. "Improvements in electric furnaces, with a view to obtaining soft metals and other materials, in which it is necessary to prevent contamination with carbon from the electrodes.".

II. **No. 305,317.** Four patents issued. "Process and apparatus for the electric manufacture of wrought-iron, steel and cast-iron by electric heating."

III. **No. 305,373.** Patent rights granted in eleven countries. "Process and apparatus to make use of the waste heat resulting from the manufacture of pig-iron."

IV. **No. 307,739.** Thirteen patents issued. "Tilting electric furnace."

V. **No. 318,638.** Thirteen patents issued. "Electric furnace with movable electrodes."

VI. **No. 320,682.** Nine patents issued. "Process for deoxidising and carburising liquid steel."

VII. **No. 328,350.** Six patents issued. "Improvements in the production of iron and steel by electro-metallurgical means."

VIII. **No. 336,705.** Patent issued in France only. "Process for deoxidising and desulphurising steel."

IX. **No. 356,714.** Patent of addition. "Improvements in the production of iron and steel by electro-metallurgical means."

The first patent was applied for in France in March, 1900, and covered the use of a rectangular open-top furnace, provided with two or more vertical electrodes suspended above it and connected in series. Particular mention is made of two voltmeter circuits, connected between the electrodes and the metal in the crucible for the purpose of regulating the voltage of each arc, and so dividing the load equally between them. The electrodes were capable of being raised or lowered, and the use of single-phase and polyphase alternating currents provided for. The crucible could be lined with any known refractory material, and this, together with the use of a slag covering, enabled metals to be melted without the usual contamination with carbon from the bottom or from the electrodes. Heroult's first furnace was fixed, as is shown in Fig. 20,[1] and further comment is unnecessary beyond drawing attention to the power and voltmeter circuits.

In the second patent[2] Heroult describes the application of his first furnace for the production of wrought-iron and steel, and outlines the process to be used. The manufacture of steel was conducted in successive operations in the same furnace. Iron ore was first reduced to pig-iron, which, after removal of the bulk of the slag through the upper tap hole, could be further refined by the addition of ore. The "refining" action, or boiling down, could be stopped at any moment when the desired percentage of carbon had been arrived at, this being determined by constant sampling and fracture of bath tests. By making

[1] British Patent Specification, No. 16,293, A.D. 1900.

[2] *Ibid.*, No. 14,486, A.D. 1901.

the slag thoroughly basic a "purifying" action, resulting in the removal of phosphorus, was made feasible. According to Heroult the "purifying" and "refining" actions—using his own terms—could proceed simultaneously or otherwise as required, and the process could be conducted in the same manner whether the carburised metal was charged molten or in solid pieces. In the patent specification a disclaimer is made pointing out the essential differences between the crucible furnace of Siemens and the furnace in question. The furnace was substantially the same as that shown in Fig. 20, but was provided with a movable roof which confined the heat to the interior and could be readily changed. The next patent may be passed over as it does not bear directly upon the manufacture of steel.

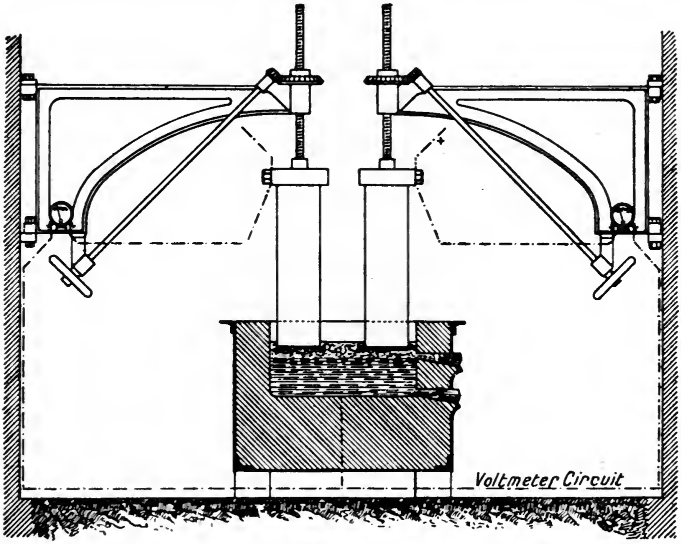

FIG. 20.

The tilting furnace[1] was the next development (Fig. 21). The construction shown embodied all the essential features of the modern furnace, and was designed to facilitate the refining and purifying processes as previously conducted in the fixed furnace. Other novelties were at the same time introduced, including the method of conveying current to the

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[1] British Patent Specification, No. 14,643, A.D. 1901.

movable electrodes, and the use of an air blast to hasten the refining action.

A method of deoxidising and carburising steel produced by the refining and purifying processes is described in a later patent.[1] The method consisted of throwing on to the bath briquettes composed of a carbonaceous material agglomerated with iron and steel filings or turnings, which, by their density, forced the briquettes through the slag into contact with the steel, so that the carbon might be more easily absorbed. The use of this material has not met with much favour, as the carbon absorption was too low and erratic, any variation in the size of the pieces, slag condition, and temperature of the bath influencing the degree of absorption. It is doubtful also whether any deoxidisation was effected except by the subsequent indirect action of any carbon entrapped in the slag.

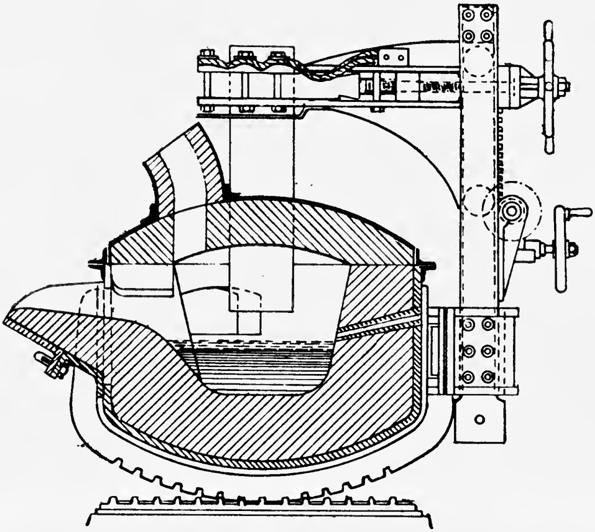

Fig. 21.

It was later realised that the process of decarburising pig-iron by electrical energy was not economical, and it was then proposed to transfer liquid steel, which had been either blown in a Bessemer converter or boiled down in an open hearth furnace, to the electric furnace for the final conversion to finished

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[1] British Patent Specification, No. 6950, A.D. 1903.

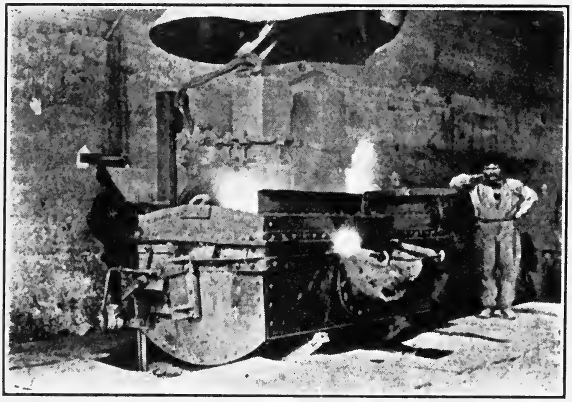

Fig. 22.

[*To face p. 25.*

steel. This method, which amounts to a Duplex process, is described in a further patent.[1]

Lastly, the process for deoxidising and desulphurising steel as is now known was patented in France in November, 1903, so that, in rather more than three years, all those metallurgical treatments essential to the production of high quality electric steel in arc furnaces had been discovered and made use of commercially. These treatments may be briefly summarised as follows:—

I. 2nd Patent.—Melting cold cast-iron and steel scrap by arc heating.

II. 1st Patent.—Regulation of load.

III. 2nd Patent.—Oxidation of carbon and removal of phosphorus and other impurities by a basic slag.

IV. 2nd Patent.—Removal of the foul slag.

V. 6th Patent.—Subsequent carburising of the bath if required.

VI. 8th Patent.—Deoxidation and desulphurising by means of a highly basic slag free from oxides.

VII. 6th Patent.—Addition of alloys to the bath without loss prior to casting.

VIII. 7th and 9th Patents.—Refining liquid steel.

In 1904 the Canadian Government Commission visited La Praz to investigate the Heroult process then being practised in the first tilting furnace built (Fig. 22).

The furnace was connected to a single-phase alternator coupled direct to a water turbine. At normal speed the voltage was 110 and the frequency 33, the power at full load being about 360 kw. and generally much less during the melting down operation. The power consumption, in the case of those heats witnessed by the commission, averaged 1100 units per ton of ingots both for dead soft and high carbon steels. The miscellaneous scrap used was very impure, containing 0·22 per cent. phosphorus, and the period of refining was consequently prolonged. The radiation loss of the furnace was also very high in relation to the total available power, so that the high power consumption was to be expected.

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[1] British Patent Specification, No. 7027, A.D. 1903.

The single-phase rectangular furnace was followed later by the three-phase furnace of circular form, and with this modification the ultimate development of the Heroult furnace was reached, as characterised by its essential features.

At the same time that Heroult was working on the problem of making low carbon alloys, another investigator, Charles A. Keller, was similarly engaged. The early work of the latter, which led to the issue of a patent[1] in France in 1900, was briefly described by the inventor himself in a paper delivered before the American Electro-Chemical Society in 1909.[2] From the description, the furnace closely resembled that patented by Heroult in March of the same year embodying:—

I. The use of two separate heating zones at the upper surface of the conductive charge.

II. Voltage regulation of each electrode.

III. Arrangement of the electrodes in series.

IV. Construction of the furnace to obtain products by the tapping method.

Keller experimented with a furnace of 15 cwts. capacity up to the year 1902 at the Kerrouse Works (Morhiban, France), more especially for the manufacture of steel from cold scrap. The ingots were tested at different works near St. Etienne and, following the favourable results obtained, a larger furnace of 2½ tons capacity was installed at the works of the Société des Etablissements Keller Leleux at Livet (France). This furnace was likewise provided with two electrodes and was experimented with for three years, during which time the metallurgical problems and products made were investigated by J. Holtzer & Co., of the Unieux Steel Works (Loire). Prompted by further success, it was decided to install a furnace of 8 to 10 tons capacity at the latter works, several modifications of interest being incorporated in the design. The furnace body (Fig. 23), consisting of a steel shell lined with a suitable refractory material and mounted for tilting on rollers, was entirely independent of the electrode carrying gear. A novel feature was introduced by interleaving the bus bars and bringing them above and then over the furnace body, where they terminated in a special

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[1] French Patent, No. 300,630, 23/5/1900.

[2] "Am. Electro-Chem. Society," Vol. XV, 1909.

distribution block. The bus bars of different polarity were there divided, and each group provided with two separate sets of lugs, so arranged that the flexible connection to the electrode

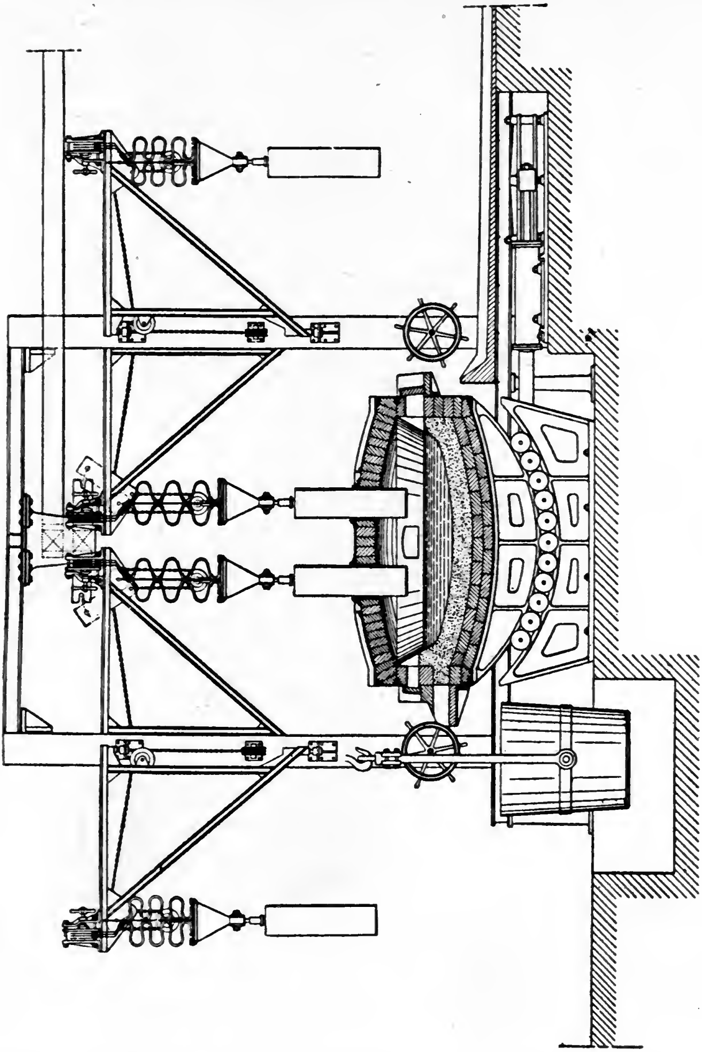

Fig. 23.

holders might be rapidly and securely made when changing electrodes. The latter were suspended from jib cranes, which swung into a position over-hanging the furnace roof. The furnace operated on a single-phase supply, and the load circuit contained two pairs of arcs in series in place of two single arcs, as in the Heroult furnace, the line voltage being equally divided between each pair of arcs. The two electrodes of each group were capable of individual movement and electrically connected in parallel. There were, then, four distinct arcs, one pair of which was in series with the other pair, whilst each pair consisted of two arcs in parallel. The regulation was necessarily complicated, any inequality of current flowing through the two electrodes of either pair being adjusted by the simultaneous lowering of one and raising of the other; balance of voltage between each pair and the bath was likewise effected by raising one pair and lowering the other. Other combinations of movements were also made possible by a system of valves operating hydraulic cylinders. The furnace at Unieux was used for refining open hearth furnace steel already low in phosphorus but high in sulphur. The power consumption per ton of steel averaged 275 kw.-hours, a load of 750 kw. being taken for about 2¾ hours. The experimental work at Livet done prior to the erection of this large furnace at Unieux was concluded in 1905, and in the meantime led to the issue of several French patents.

I. **French Patent, No. 300,630.** 23/5/1900. "Electric furnace improvements."

II. **French Patent, No. 322,700.** 2/7/02. "Process for melting and refining metals and other substances electrically."

This invention was also patented in England[1] and described as follows:—

"Metals or other substances are electrically heated for refining or other purposes by current passing between electrodes E (Fig. 24) dipping into the materials. Preferably several electrodes are used, so that one may be changed without stopping the operation. The heating may be effected in a furnace G, arranged to receive the product of a cupola furnace F; or in an ordinary foundry ladle, which is carried on a movable truck so that it may be charged from one or more furnaces and then placed under vertically movable electrodes. Metal may thus be collected and kept hot for casting, and other materials may

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[1] British Patent Specification, No. 15,271, 1902.

be added to effect refining; waste iron or steel may be added and steel produced."

The specification clearly states that the electrodes dip into the "material" (slag in the case of steel-making), and are arranged in two groups of opposite polarity, each group being built up of two or more electrodes separately movable and connected in parallel.

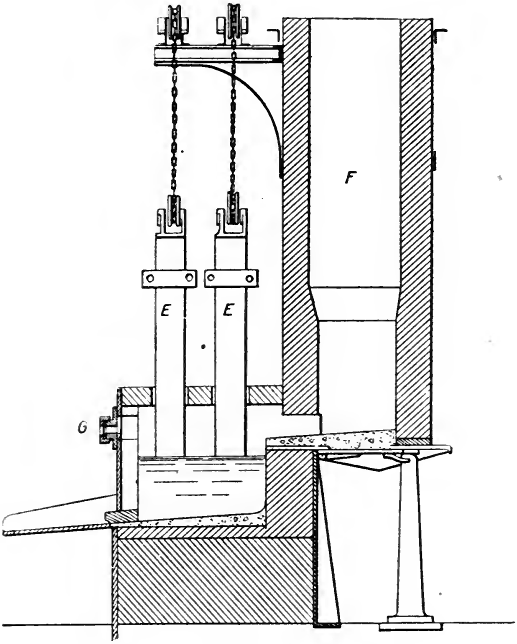

Fig. 24.

III. **No. 329,013.** 2/2/03. "Improvements in the process of electric melting and refining."

The feature of this patent lies in the use of free arcs formed between the electrodes and the slag, as opposed to the electrodes dipping into the slag and heating purely by resistance. In this way alone can an oxidised slag be maintained. Oxidising and dephosphorising could be performed under a suitable slag and then followed by a refining period under a different slag.

IV. **No. 387,461.** 6/5/07. "Process for carburising a liquid metal."

Composite blocks are made by pouring a metal into a mould previously filled with small pieces of broken carbon. Such blocks when thrown on a bath of metal will sink by their weight, and bring the carbon into close contact with the metal to be carburised. In the case of steel, cast-iron is the metal used for making the blocks.

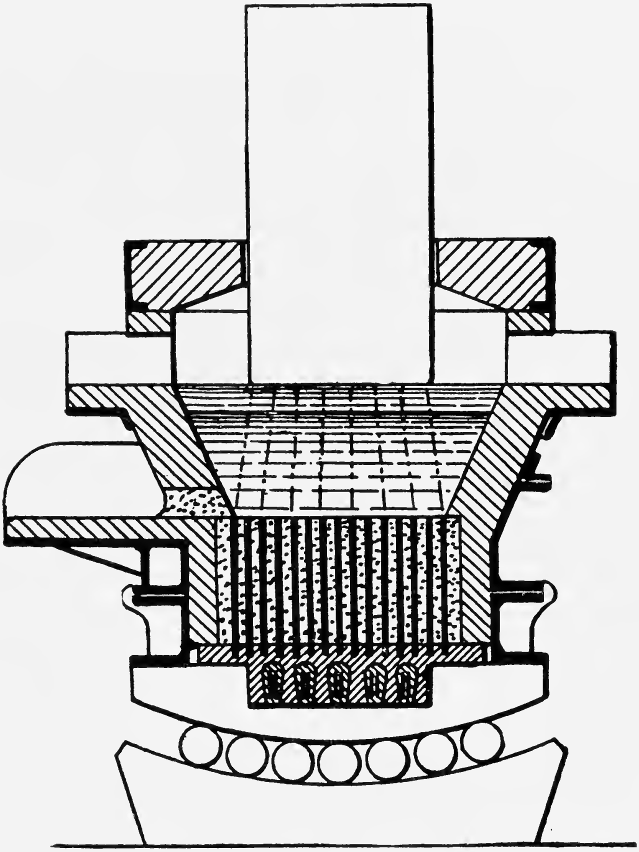

Fig. 25.

V. **No. 393,740.** 4/11/07. "System of conducting hearth for electric furnaces" (Fig. 25).[1]

This invention relates to the use of a conducting hearth, composed of a refractory material suitable for steel manufacture

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[1] "Am. Electro-Chem. Society," Vol. XV, p. 98.

and capable of carrying the full load current. It will be remembered that a carbon bottom, serving as one electrode, had been used prior to 1900 for the manufacture of ferro-alloys, but was abandoned by Heroult for the reason of carbon contamination in cases where low carbon alloys or steel were to be made.

On returning to the use of a conducting hearth, it was necessary to employ a material free from carbon, which was at the same time sufficiently conductive to carry the full current when cold. For this purpose Keller used a composite bottom, consisting of a number of iron rods connected to a fixed bottom plate and embedded in magnesite or dolomite, mixed with a tar or pitch binder rammed in to form a solid plug. The metallic portion of the hearth was relied upon to complete the circuit when cold. The distribution of the current was equalised over the entire section of the hearth, and in this respect differs from the metallic bottom electrodes used by Girod about three years earlier.

VI. **No. 400,461.** 6/6/08. "Process for the exact carburising of steel."

This method consists in the immersion of a carbon block into the bath of steel in such a manner that the exact loss in weight by absorption in the bath is accurately recorded as carburising proceeds.

VII. **No. 399,643.** 27/4/08. "Radiating distribution of conductors for multiple electrode furnaces" (Fig. 26).[1]

This patent refers to a method of interleaving the bus bars of opposite polarity and bringing them to a central point of distribution, whence the current flows through suitable radiating connecting lugs to the four electrodes, as previously described for the 8 to 10 ton furnace built at the Unieux works.

VIII. **No. 400,655.** 15/6/08. "System of regulating the circuits supplying multiple electrode furnaces."

This method was embodied in the four-electrode furnace as described, and consists of voltage and current regulation of the electrodes in series and parallel respectively.

IX. **No. 14,728/393,740.** 21/11/10. Patent of Addition. "System of a conducting hearth for electric furnaces."

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[1] "Am. Electro-Chem. Society," Vol. XV, p. 114.

In this case, the composite bottom previously described is replaced by ramming the entire hearth with a mixture of magnesite and iron filings, bound together by tar or pitch. For operation on a three-phase supply, Keller proposed to connect the three electrode circuits either in delta or star, when, in the latter case, the conducting hearth is made the neutral and connected to the star point of the transformer secondary circuits.

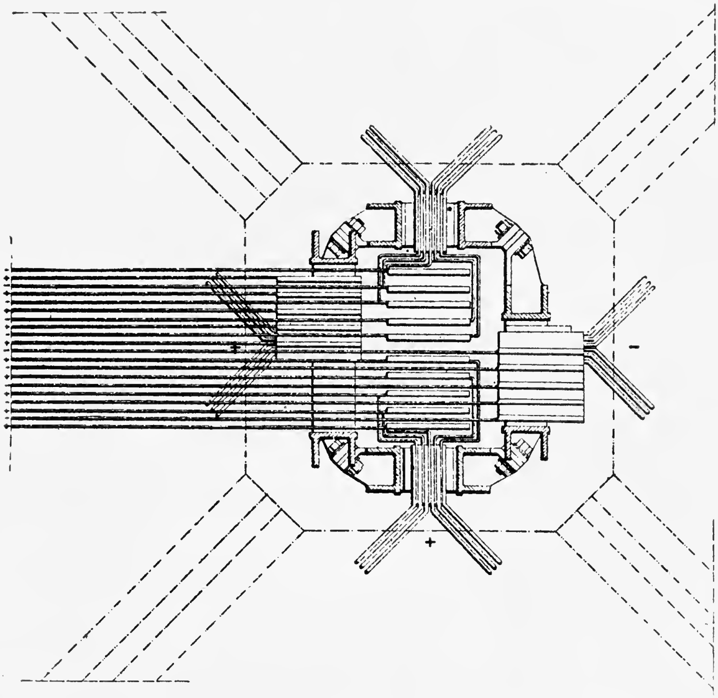

Fig. 26.

In a general survey of Keller's inventions it will be seen that for seven years his efforts were restricted to furnaces employing suspended electrodes operating in series, the bottom electrode or conducting hearth being only adopted towards the end of 1907. Paul Girod had already in 1905 utilised a modification of the old carbon bottom for melting steel, and his contribution to the general study of electric steel-making must next be mentioned.

Girod had been engaged for many years in the use of electric furnaces for the reduction of ores before turning to steel melting. His first invention is particularly interesting, as it was the first application of Siemens' original principle of direct arc heating to the commercial manufacture of steel. Heroult and Keller had overcome the difficulties due to a carbon electrode bottom by eliminating it, whereas Girod preferred to construct a conductive hearth which would prevent all possibility of carbon contamination. The successive steps and improvements, evolved from the time of his first studying the subject, will be again best followed by enumeration of the several patents issued in France.

I. **French Patent, No. 350,524.** 4/1/05. "Electric furnaces."

The nature of this invention is concisely described in the resumé of the specification of which the following is a translation: "An electric furnace in which one of the poles is formed by one or several graphite electrodes, moved mechanically or by hand, the other pole being constituted of several electrodes buried in the furnace lining, and situated at such distance from the hearth, or cooled artificially in such fashion, that at their extremities a certain quantity of fused material is chilled and forms upon them a solid protective layer which prevents contact of these electrodes with the metal being manufactured. Contact between the latter and the movable electrode is avoided by the fact that the said electrode dips only into the layer of slag which covers the metal on melting."

In the subject-matter of the specification further details are given of the manner of constructing the bottom conductors (Fig. 27).

The actual poles, which may be of metal or graphite, are usually water-cooled, and, if insulated, may be connected in parallel or in series to the bus bars. In the case of graphite, the poles are covered at their upper extremities with a metallic pole piece, which either partly melts or becomes enlarged during operation of the furnace.

When the furnace is circular in shape the poles give place to an annular channel, which is filled with cast-iron and similarly water-cooled; this particular construction was especially suggested for making steel from pig-iron (Fig. 28). The furnace

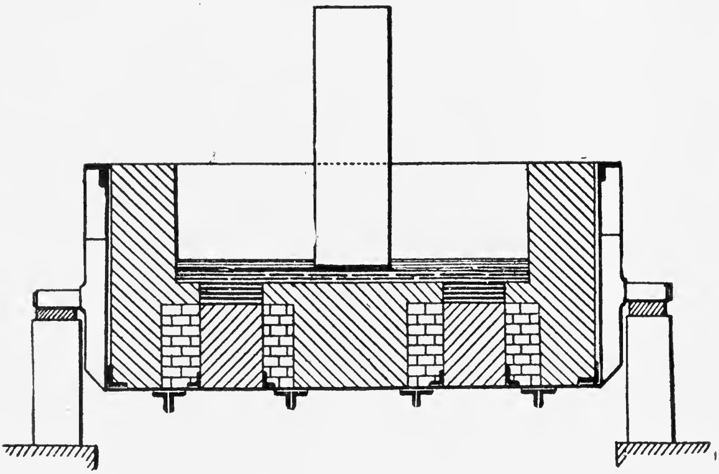

Fig. 27.

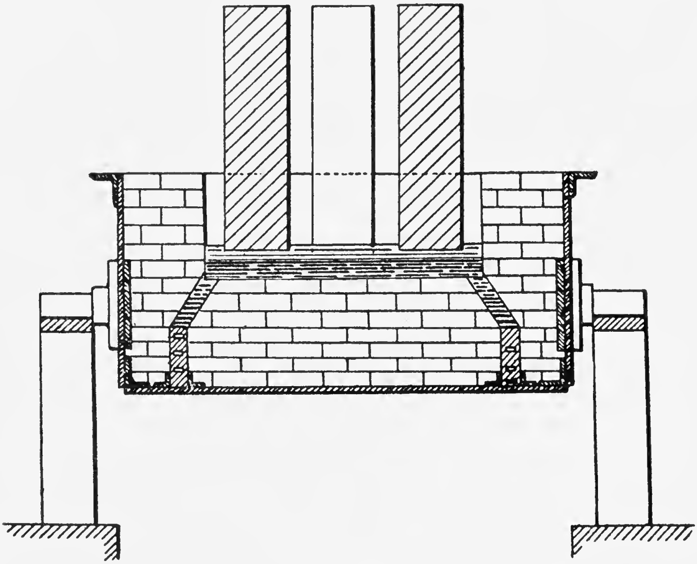

Fig. 28.

can be either fixed or constructed for tilting, and can operate on either continuous or alternating current supplies. With proper choice of voltage the electrodes will form arcs above the slag, when the known processes of decarburising, dephosphorising and desulphurising can be conducted. Preference is given to the use of a non-conducting material surrounding the pole pieces or conductors. In the single-phase design the current from the upper central electrode is forced to take a diagonal path to the two lower electrodes, causing circulation of the metal.

In 1902 Heroult filed a patent application in Belgium which covered the use of carbon-bottom electrodes with metallic end pieces to prevent carbon contamination of the bath. Girod, therefore, was not the first to conceive the idea of such bottom electrodes, but was the first to put it to commercial use.

II. **French Patent, No. 350,802.** 16/1/05. "Tilting furnace with conducting hearth."

In his first patent, Girod suggested that his furnace might be constructed to tilt. In this patent a specific method of tilting the furnace on trunnions is described, and the provision of two tapholes at different levels is mentioned.

III. **Patent of Addition, No. 4829/350,524.** 4/1/05. "Electric furnaces."

In place of the annular conducting ring previously patented, the conducting poles may be formed by leaving channels in the furnace lining extending upwards from the shell plate to the level of the working bottom. These channels are filled up with solid lumps of metal, which, during a preliminary operation of the furnace, become fritted together to form a solid pole.

IV. **No. 388,614.** 4/6/07. "Process for the manufacture of iron and steel in the electric furnace."

This process requires the use of at least two electric furnaces, one being employed to melt and refine cold scrap, the other of much smaller capacity being used solely to finish the oxidised and dephosphorised steel from the former. The large melting furnace is never emptied during continuous operation, sufficient steel being retained to take up an additional charge of cold scrap equal to the quantity transferred to the smaller furnace for finishing. This process saves the time and labour of skimming off the oxidised slag prior to finishing, and, at the same time, obviates the loss of heat when charging cold scrap into an empty furnace.

V. **No. 402,758.** 6/5/09. "Process of refining liquid steel from furnaces other than electric."

Purposely under-oxidised steel from an open-hearth furnace or converter is transferred to the electric furnace, and there cooled down to promote evolution of dissolved gases (hydrogen and nitrogen). This is followed by again raising the temperature, when a further elimination of C, Si, Mn and P is effected in the ordinary manner under a basic slag.

VI. **No. 11924/402,758.** 7/12/09. "Patent of addition."

The process is similar to the above, but is carried further by successive raising and lowering of temperature, the evolution of dissolved gases being assisted by a small addition of carbon before reducing the temperature. This addition is also made after dephosphorising.

VII. **No. 12771/402,758.** 15/6/10. "Patent of addition."

This patent has for its object the removal of dissolved oxides together with the gases. A small quantity of C, Si, and Mn is left in the steel before transference to the electric furnace, and on reduction of temperature reacts with the dissolved oxides to form fusible insoluble silicates, which readily fuse and rise upwards through the bath. C, Si, and Mn may be actually added to promote this reaction.

VIII. **No. 416,927.** 9/6/10. "Method of arranging electric furnaces for three-phase working."

This patent refers to a special method of connecting the three upper electrodes in star fashion, the bottom electrode being connected to the neutral point of the secondary circuits. One phase of the star connection is reversed, which compels the major portion of the current to flow through the bottom electrode, while the load is nearly equally balanced on the supply phases (Fig. 61).

IX. **No. 422,717.** 26/8/10. "Method of arranging electric furnaces for three-phase working."

Three methods are given of connecting a furnace, provided with two upper electrodes and a bottom electrode, to operate on a three-phase supply system. In each case the low tension supply to the furnace is actually two-phase.

Up to the year 1905 the only direct arc furnaces used commercially for steel-making belonged to that class in which the electrode circuits were independent of the furnace lining. The advent of the Girod furnace in that year, and the practical results achieved, led others to modify the hearth construction and to adopt it for use with different systems of electrical connections. Girod restricted the construction of his conducting hearth to metallic or graphite poles, embedded in a material which was preferably a poor conductor at high temperatures, and therefore relied entirely upon primary conductors for completion of the circuit through the charge and furnace hearth. Keller in 1907 modified the above method by using a composite bottom as already described, and the value of a conducting material, consisting of magnesite with a carbonaceous binder, was fully realised by him although used in conjunction with metallic rods as primary conductors.

There was, however, considerable prejudice against the use of metallic conductors embedded in a furnace bottom when directly in contact with the molten bath, and it was suggested that considerable difficulty might be occasioned in repairing a bottom between heats, owing to partial emptying of the holes which normally constituted the metallic electrodes. This prejudice was not always borne out in practice. With a view, no doubt, to avoid such difficulties, a furnace was constructed at the Firminy Steel Works in France and a patent applied for in March, 1908. The hearth had no metallic conductors, and was built up of successive layers of a refractory material, such as magnesite or dolomite, mixed with a carbonaceous binder in decreasing quantities towards the upper surface. It was claimed that a highly conducting hearth could be built up in this manner without fear of carbon contamination of the bath, and with a possibility of restarting a furnace when cold. This type of hearth construction has now been generally adopted in preference to the metallic electrodes used by Girod and Keller, and of modern furnaces using conductive hearths the Electrometals design was the first to gain a wide application.

The early forms of single-phase arc furnace usually required a special generating plant, owing to the impossibility of transforming from either two- or three-phase supplies with static transformers, and it was to overcome this fundamental objection that the Electro-metals design was introduced; the furnace operated on a two-phase low tension supply, transformed from either a two-phase or three-phase high tension system. The original patent[1] was applied for in Sweden in August, 1908, and embodied the use of one upper electrode in each phase circuit, which also included a conductive hearth and a common neutral return conductor, the latter being connected from the hearth to the neutral point of the two phases (Fig. 46). A later[2] patent describes an improved form of rectangular tilting furnace; the electrode holders are mounted on swivel brackets, and so can be raised and swung round clear of the roof. The electrode regulating motors impart movement to the electrode brackets through a system of screw feed and telescopic shafting, and so can be situated away from the furnace body. This construction (Fig. 29) is not, however, adopted in modern types.

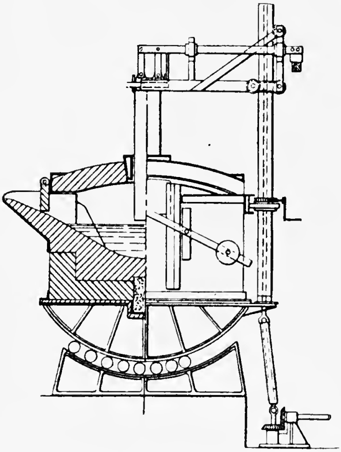

Fig. 29.—Electro-metal tilting furnace.

In 1911 a modification of the Electro-metals furnace was introduced by Stobie,[3] who substituted a return conductor for each phase in place of the common return. The two phases were thus separated and constituted a four-wire two-phase system in place of a three-wire neutral return system; the bottom electrode of each phase was situated diagonally opposite its corresponding upper electrode in place of below it (Fig. 49). In a more recent furnace[4] of larger capacity the two-phase bottom electrode type gives place to a four electrode three-phase star-

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[1] Swedish Patent, No. 28,687, 1/8/08.

[2] British Patent, No. 12,430, 21/5/09.

[3] *Ibid.*, No. 6741, 1911.

[4] *Ibid.*, No. 2081, 1912.

connected type, in which one electrode is connected to the star-point of the three-phase system and serves as a return for any unbalanced current.

An ingenious application of two-phase current to indirect arc furnaces of the Stassano type was introduced by Rennerfelt[1] in 1912 with the object of reducing the excessive wear of the roof and lining. In this furnace two arcs are formed between the horizontal phase electrodes and a vertical neutral, and are deflected downwards by mutual repulsion of the magnetic fields set up.

The value of small electric furnaces for the manufacture of light castings had been well proved, and other new types were introduced to meet the special requirements of the foundry. The Snyder furnace,[2] together with the Rennerfelt furnace just alluded to, may be regarded as falling in this category. The furnace operates on a single-phase supply, and is provided with one top and one bottom electrode, the latter being embedded in the hearth and in direct communication with the metallic charge or bath. The chief feature lies in the use of a high arc voltage, usually about 110 volts at full load, which produces a long arc from which the roof and upper lining of the furnace is only slightly shaded by the electrode itself. By using a high voltage arc the current required for a given power input is correspondingly small, which entails a saving in the cost of electrical equipment. It is also claimed that a charge of steel scrap may be melted quicker, and this is now generally accepted as being true, provided the electrical equipment is at the same time designed to prevent heavy current fluctuations. The Snyder furnace, designed on this principle, enables a very steady load to be maintained even when melting irregular shaped scrap, but this can only be done at the sacrifice of power factor which is considerably lowered by increasing the reactance of the circuit. The construction of the original furnace is shown diagrammatically by Fig. 30. The original specification lays stress on the special construction of the furnace shell, which is composed of a number of laminated steel sheets to prevent the heating effect of eddy currents, and at the same time provides the necessary reactance of the circuit.

[1] British Patent Specification, No. 7367, 1912.

[2] *Ibid.*, No. 25,171, 1913.

In the case of the conductive hearth furnaces so far considered, the current flowing through the hearth has been fixed as a definite proportion of the total current flowing through the upper electrode circuits. This has been considered an objection by some, and attempts have been made to arrange the electrical installation and the method of furnace connections so as to admit of varying at will, within certain limits, the amount of current flowing through the conducting hearth.

The first move in this direction was made by Dixon, whose earliest patent application [1] was filed in 1914. The furnace was provided with four or six upper electrodes and one bottom electrode. By means of switches in the high tension circuit the phase relations of the low tension transformer circuits could be altered to vary the proportion of the current flowing through the conducting hearth; the same result could also be accomplished by reversing one or more of the transformer connections, which had been suggested by Girod in 1911 in the case of three-phase supply. Two later patents [2] referred to special arrangements of a two-phase low tension supply suitably connected to four electrodes so as to render equal distribution of the load between the four electrodes possible by hand or automatic control depending upon current variation only.

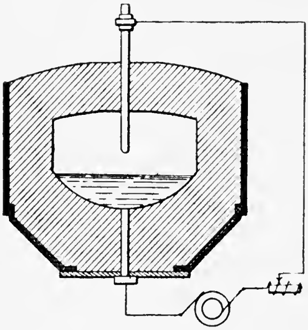

FIG. 30.—Snyder furnace.

A simple single-phase indirect-arc furnace was introduced in Italy in 1914 by F. Bassanese and has there acquired considerable popularity. Certain unusual features were presented by the method of electrode adjustment, but in general appearance the furnace resembles the revolving steel melting furnace of Stassano.

Another furnace of more simple design, but presenting the same feature of a variable bottom electrode current as first conceived by Dixon, was introduced by T. H. Watson & Co.

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[1] British Specification, No. 4742, 1914.

[2] *Ibid.*, No. 111,103, October, 1916; No. 111,104, October, 1916.

(Sheffield), and is better known as the Greaves-Etchells furnace. According to the first patent[1] the furnace could operate on either a three-phase or two-phase supply system, and further by the use of only two upper electrodes a good balance of load could be obtained on the primary supply phases, where static transformers are used. In one case, when the current through the two upper electrodes is equal, the voltage in the secondary windings connected to the lower electrode can be varied to produce the required balance. Further modifications[2] were later made in the use of three-phase low tension circuits for supplying power to the furnace.

In conclusion, mention must be made of the Booth-Hall furnace, which combines the essential features of the Rennerfelt and original Stobie furnaces. In this historical review it has only been possible to mention those types of furnaces which are either in use at the present day, or which have in the past demonstrated certain principles that are now the foundation of more widely known and later types.

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[1] British Specification, No. 106,626, March, 1916.

[2] *Ibid.*, No. 118,233, 1918; No. 121,563, 1918.

CHAPTER II.

ELECTRICAL DEFINITIONS.

I. **Alternating Current Supply.**—A source of electrical energy is "alternating" when the voltage passes through periodic variations of magnitude, together with reversal of sign.

II. **Alternating Current.**—If a source of alternating electrical energy is connected to a suitable circuit, a current will flow the magnitude and direction of which will, at all moments, vary directly as the voltage.

III. **Wave Form or Wave Shape.**—The wave form of an alternating current is found by plotting a curve with instantaneous values of current as ordinates against corresponding time values as abscissæ. The wave form of the impressed voltage may similarly be plotted. Owing to the characteristic design of alternating current generators, the wave form approximates to that of a sine wave, which can be developed by simple geometrical means.

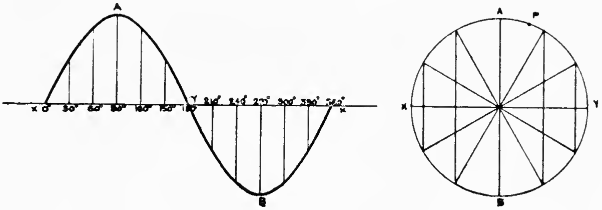


Fig. 31.

Referring to Fig. 31, imagine a point P revolving on the circumference of a circle AYBXA at a uniform speed and in a clock-wise direction from X through AYB, and back again to X; then, by plotting the vertical distance of the point P from the horizontal diameter XY against time, or corresponding angular displacement from a time axis XY, a curve XAYBX will be obtained.

IV. **Cycle.**—Referring again to Fig. 31, the revolving point P, starting from any position with a uniform velocity, will have performed one complete cycle when it has returned to its starting point, having made one complete circuit.

One cycle is therefore completed in 360° of angular displacement. Used in a purely electrical sense, a cycle is regular and periodic. A "cycle" is represented by the symbol ∽.

V. **Electrical Degree.**—One 360th part of a cycle.

VI. **Period.**—The time required to complete one cycle.

VII. **Frequency or Periodicity.**—The number of cycles performed in one second.

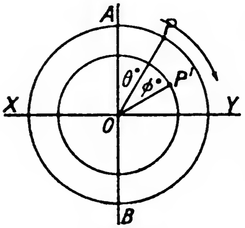


Fig. 32.

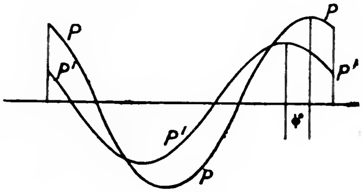


Fig. 33.

In electrical equations frequency is denoted also by the symbol ∽.

VIII. **Phase.**—If a point P revolves about a centre O (Fig. 32) then the phase of that point, relative to any time axis AO, is its angular displacement from that axis.

Again, if two points P and P′ revolve about the same centre O with the same frequency but not necessarily equal radii, and if the angle between OP and OP′ equals $\phi°$, then the points P and P′ are said to be $\phi°$ "out of phase".

By plotting two wave curves for these points P and P′ (Fig. 33), representing a complete cycle and beginning at simultaneous moments when these points are at given vertical distances from XY and $\phi°$ out of phase, it will be seen that the wave curve for point P is always $\phi°$ out of phase with the curve for P′. In this particular instance the plotting of the wave curves has been commenced at a moment when the point P is

θ° and the point $P'(\theta^\circ + \phi^\circ)$ past their maximum positive value at A.

These wave curves might equally well have been obtained by plotting instantaneous values of voltage against time for two separate sources of alternating current of similar frequency, but ϕ° out of phase with one another and having voltages of different maximum values.

It will be shown later how two or more alternating currents of the same frequency, but out of phase, may be combined to produce a resultant effect in the same way that forces of equal or unequal magnitude, acting on a body at different angles, may be compounded by graphical methods to produce one resultant force.

IX. **Root-Mean-Square or Effective Values of Voltage and Current.**—It has been stated (see Def. II.) that, when a suitable closed circuit is connected to an alternating current supply, a current will flow whose magnitude and direction at any moment will depend upon the instantaneous value of the voltage as regards both magnitude and sign.

Since the voltage across the terminals of such a closed circuit undergoes periodic variation of magnitude and reversal of sign, it may be represented by a sine wave curve; likewise the current flowing through the circuit may be similarly represented.

Now the physical effects of any electric current are generally due to the power developed, which is—as will be seen later—proportional to the product of current and impressed voltage. It is necessary, therefore, for purposes of measurement and calculation, to know the value of the average current or voltage which would, in any half cycle, produce exactly the same effect as the sum of all the instantaneous current or voltage values, which rise from zero to a maximum positive or negative value and then fall back again to zero.

Again, since power is proportional to C × E, and either C or E can be expressed in terms of one another and circuit resistance by the equation $C = \frac{E}{R}$, it follows that the physical effect produced is proportional to the square of the current and likewise to the square of the voltage.

$$\begin{aligned}
\text{i.e. if the power W (watts)} &= \text{C} \times \text{E} \\
\text{then W} &= \text{C} \times \text{CR (R = resistance of the circuit)} \\
&= \text{C}^2\text{R} \\
\text{and again, if power W} &= \text{C} \times \text{E} \\
\text{then W} &= \frac{\text{E}}{\text{R}} \times \text{E} \\
&= \frac{\text{E}^2}{\text{R}}.
\end{aligned}$$

From this reasoning it follows that the mean effective value of an alternating current or voltage is proportional to the square root of the mean of the instantaneous values squared, and not to the mean of the instantaneous values themselves.

For any alternating current supply whose voltage may be represented by a simple sine wave, the effective values of voltage or current flowing through a connected circuit are equal to the maximum or crest values ÷ $\sqrt{2}$. Ammeters and voltmeters always indicate the effective values.

X. **Lead and Lag.**—Suppose Fig. 33 represents the voltage or current curves for one complete cycle of two alternating currents of similar frequency, but differing in phase. The maximum values of the curve P′P′ occur exactly $\phi°$ before those of curve PP and are said to "lead" by $\phi°$, whereas corresponding values of PP "lag" $\phi°$ behind P′P′.

If the curves P′P′ and PP represent the voltage and current waves respectively of an alternating current, then the current is "lagging" $\phi°$ behind the voltage, and the "angle of lag" is $\phi°$; similarly the current may sometimes "lead" the voltage and then have a certain "angle of lead".

XI. **Induction and Induced Currents.**—Whenever a current flows through a conductor a magnetic field is produced, which, according to the usual convention, is said to contain a certain number of magnetic lines of force, whose number and direction will depend upon the magnitude and direction of the current and the permeability of the medium through which the magnetic lines of force pass. The conductor, as shown in Fig. 34, may be either straight or coiled, and will produce magnetic fields in the directions indicated when carrying a current likewise shown.

Conversely, if a conductor lies in a magnetic field of varying intensity and in a position other than in the exact direction of the lines of force, then a voltage will be induced within it, the magnitude and sign corresponding at any moment to the rate of change in the number and direction of the magnetic lines of force cutting the conductor; no voltage is induced so long as the magnetic field remains unchanged.

Such a phenomenon is known as "induction," and, according to convention, 1 volt will be induced when a conductor cuts magnetic lines of force at the rate of 10^8 lines per second.

Supposing now that a magnetic field of varying intensity and direction is set up in an iron ring B by a coiled conductor A carrying an alternating current (Fig. 35), then a voltage will be induced in the coiled conductor C, which, if made into a closed circuit, will carry an "induced current" alternating at the same frequency as the current in the coil A. The conductor that sets up the magnetic field is generally known as the "primary winding," and that in which the voltage is induced as the "secondary winding". The induced or secondary volts will bear the same ratio to the exciting or primary volts as the number of turns of the respective windings. Each turn may be regarded as a separate conductor, so that increasing the number of turns of the secondary is equivalent to adding further induced voltage in series, and consequently raising the total voltage for the same magnetic field.

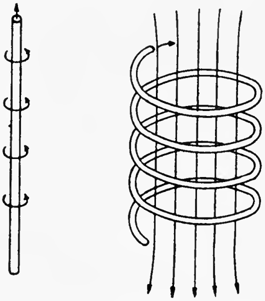


Fig. 34.

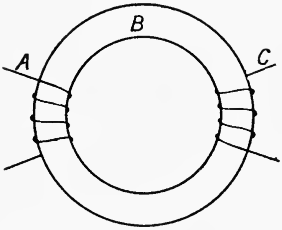


Fig. 35.

If a certain amount of power is utilised in a primary winding, it is obvious that only the same amount of power can be developed by the secondary, even assuming 100 per cent. efficiency of conversion. Therefore, since the primary and secondary voltages are proportional to the respective number of turns, the primary and secondary currents will be inversely proportional, so that the product of volts and amperes or volt-amperes is equal for both circuits. This constitutes the underlying principle of step-down transformers, to which type all induction furnaces belong.

XII. **Self-induction, Reactance, or Reactive Resistance.**—When an alternating current flows through a conductor, either straight or in the form of a coil or solenoid, it will produce a magnetic field constantly changing, as regards both the number of lines of force and reversal of their direction. As explained in Def. XI., such a conductor would be under the influence of the changing magnetic field, and an alternating voltage would therefore be induced within it.

The simple wave form of an alternating current shows that the maximum rate of change of current, and, consequently, of the number of magnetic lines of force produced, takes place at the steepest point of the curve, which is just at the moment of reversal. Although there are no magnetic lines of force at the exact moment of reversal, the rate of diminution or increase in number is then at a maximum, and at this point, therefore, the "self-induced" or "reactive" voltage reaches its highest value. It follows, therefore, that the wave forms of the current flowing and the self-induced voltage, if plotted simultaneously, would be 90° out of phase and have the same frequency.

The result of such self-induced voltage, which is out of phase with the applied voltage, is to reduce the power of the latter to force current through the conductor, and at the same time to cause the wave curve of the current flowing to lag behind that of the applied voltage. This latter effect is important in connection with "wattless current" and power factor. The self-induction, reactance, or reactive resistance of a conductor or circuit is, then, a property by virtue of which it offers resistance to the flow of alternating current. Such resistance is in no way due to the material of which the conductor is composed, but to characteristics which promote the generation of a self-induced or reactive voltage as above described.

The self-induction or reactance of any given circuit is equal to $2\pi \backsim L$ ohms. The co-efficient of self-induction or inductance (symbol L equals one Henry) is present when one ampere produces 10^8 interlinkings of lines of force and windings (i.e. number of turns × lines of force per second).

The number of lines of force or the magnetic flux produced by a current of one ampere may be calculated from various formulæ according to the nature of the electric and magnetic circuits.

XIII. **Reactance Voltage or E.M.F. of Self-Induction.**—The self-induced E.M.F. or voltage may be calculated as the product of current and reactance, and is equal to $2\pi \sim LC$.

XIV. **Percentage Reactance Drop.**—The term "percentage reactance drop" is used for expressing the reactance voltage in a circuit as a percentage ratio of the voltage applied to the circuit, usually at normal full load current.

XV. **Wattless Current.**—When the voltage and current curves of an alternating current are in phase, the entire "effective" or root-mean-square current value is useful for doing work as electrical energy. This is not true for a current which "lags" or "leads" the voltage, when only that component of the current which is in phase with the voltage is capable of doing useful work, the other component being "wattless".

It should be here mentioned that a wave curve of current or voltage may easily be resolved graphically into two components, and in the same way two waves of similar frequency, but differing in phase or magnitude, may be compounded to produce one wave curve.

XVI. **Volt-ampere.**—The product of the effective or measured value of volts across a circuit and the effective current value in the circuit is expressed in terms of volt-amperes (V.A.) or, when divided by 1000, of kilo-volt-amperes (K.V.A.).

XVII. **Power Factor.**—The power factor of a circuit in which electrical energy is transformed or dissipated, is the cosine of that angle—usually denoted by ϕ°—by which the current "lags" or "leads" the impressed voltage. Power factor is of great importance in all electric furnace installations, and, except in certain cases where considerable circuit reactance is purposely introduced, should approach unity as far as possible at normal loads.

The power factor of a circuit indicates the ratio of useful current flowing to the total current, and since the capacity of transformers and generators is limited by the permissible current, it is obvious that the power factor represents the ratio of actual power output to the maximum possible output. In other words, if the power factor is ·6, then only ·6 of the power plant capacity is available for doing useful work.

XVIII. **Watt and Kilowatt.**—The true power absorbed in a circuit is measured in terms of watts or kilowatts, and is the product of volt-amperes or kilo-volt-amperes and power factor,

i.e. Watts $= V \times A \times \cos \phi^\circ$ (where ϕ° is the angle of "lag" or "lead"),

Kilowatts $= V \times A \times \cos \phi^\circ \div 1000$.

If the current and voltage wave forms are in phase, then $\cos \phi^\circ = 1$, and the volt-amperes are equivalent to watts.

XIX. **Surging.**—Strictly speaking, the term "surging" should be used only to express a more or less constant and periodic current or voltage fluctuation, but for furnace loads may be more loosely used to denote persistent current instability or fluctuation of considerable magnitude. In the case of direct arc furnaces it sometimes happens that the nature of the charge is such that a steady load is most difficult to maintain. This condition is most likely to occur with a loosely packed charge of heavy and irregular shaped scrap, and results from the constantly varying resistance of the charge. This is due to local fusion of the metal by the arcs formed at the various points of contact, which may either facilitate or interrupt the passage of current. When the resistance of a charge of scrap is very low, slight movement of the electrodes may be sufficient to cause sudden rushes of current, which may at times reach values 100 per cent. in excess of the normal full load current. Surging is sometimes experienced even when the entire charge is molten, and is then probably due to a phenomenon known as "pinch effect," which occurs in the slag covering; this, however, only happens when the electrode is almost, if not actually, in contact with certain basic and acid slags.

Surging is highly objectionable for many reasons, some simple and some very complex. So long as a current is violently fluctuating it is only possible to prevent it reaching an instantaneous value, large enough to automatically open the main switch, by operating the furnace at a considerably reduced average load. The installation would not then be working at its full capacity, and reduction of output accompanied by an increase in power and other costs would result. Surging or current fluctuation can to a great extent be reduced, if not prevented, by increasing the self-induction of the load circuit, which can be done by the aid of reactance or choking coils.

XX. **Single-phase Alternating Current.**—The alternating current flowing through a single circuit by virtue of an alternating voltage is known as a single-phase current. A source of single-phase current can only have one wave curve for its impressed voltage. (See also Def. I.)

XXI. **Two-phase Alternating Current.**—A so-called "two-phase current" denotes the availability or application of two sources of single-phase current of similar frequency and voltage magnitude, but generally differing in phase by 90°. This particular phase displacement permits of a useful combination of two such single-phase currents flowing through separate circuits for different purposes. The two wave curves of the individual impressed voltages are represented in Fig. 36. Such alternating currents are also said to be in "Quadrature," because the phase displacement is one-quarter of a cycle.

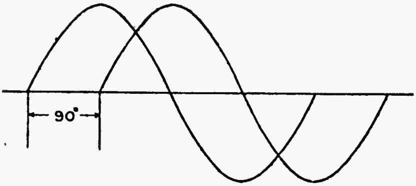


FIG. 36.

XXII. **Three-phase Alternating Current.** — A so-called "three-phase current" denotes the availability or application of three sources of single-phase current of similar frequency and voltage magnitude but differing in phase by 120°. Again, this phase displacement permits of a variety of combinations of three such single-phase currents flowing through separate circuits, which may be for lighting, heating, power generation or conversion to two-phase and even four-phase current. The wave curves of the individual impressed voltages are shown by Fig. 37.

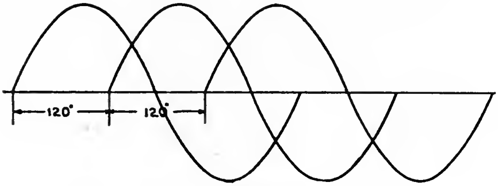


Fig. 37.

XXIII. **Load Factor.**—This term must not be confused with "power factor". The percentage ratio of average power demand to the maximum power demand, during any given period of time, is the "load factor" of the installation during that period. If a furnace installation, which has a maximum power absorbing capacity of 1000 kw., is actually supplied with an average of only 800 kw. during a certain period, then the load factor is 80 per cent. over that period. The load factor is of the very greatest importance in furnace work, as it is only by working as nearly as possible at the full plant capacity that the maximum economy of production can be attained.

The load factor of an installation is sometimes averaged over a period of a month or year, which will include all delays and stoppages. In any case, when referring to load factor, the period upon which it is calculated should be clearly stated, when not otherwise understood.

CHAPTER III.

APPLICATION OF SINGLE AND POLYPHASE CURRENTS TO FURNACE OPERATION.

Application of Single-phase Current.—It has been shown how a single-phase current may be produced in a circuit by a single impressed voltage that is passing through periodic changes of magnitude and sign. An outgoing current must have a return to complete its circuit, otherwise no current flows, so that at least two main leads are required to supply energy to a furnace from any source of single-phase current. The nature of the circuit and the manner in which the power is absorbed depends upon the particular furnace design. The various methods of using single-phase current for arc furnaces from a two-wire supply are shown in Figs, 38, 39, and 40, which also illustrate the principles of direct and indirect arc heating used in certain modern furnaces.

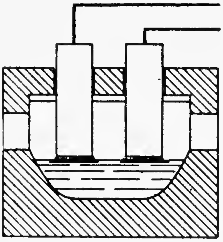


Fig. 38.

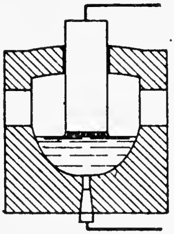


Fig. 39.

Fig. 41 illustrates in plan the heat developing and power supply circuits of a single-phase induction furnace, A being the laminated transformer core in section, B the primary windings from the main supply leads, and C the short-circuited secondary coil in which the low tension current is induced.

A three-wire system was used in the Giffre furnace, the third wire being connected from the middle terminals of two single-phase alternators to a bottom electrode, as shown in Fig. 42. The single-phase alternators, mechanically coupled together, are connected in series and generate synchronously at the same frequency and voltage. If the terminal voltage of each alternator is 50 volts, then the arc voltage will be 50, assuming the two arcs are balanced. If the electrodes are at any time unbalanced, then a current will flow through the bottom electrode back to the middle terminal.

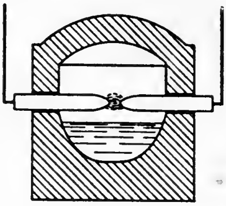


Fig. 40.

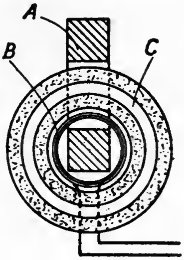


Fig. 41.

Application of Two-phase Currents.—It is more economical to generate and transmit two-phase current, owing to:

(i) The reduced cost of the generating plant.

(ii) The saving of copper in conductors.

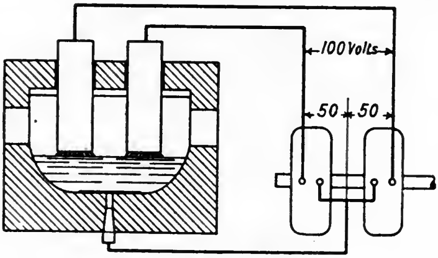


Fig. 42.

It has been explained that a two-phase system is simply a suitable combination of two single-phase currents, 90° out of phase, which can be readily compounded to produce a resultant current in a common return conductor. A two-phase supply, then, might constitute two entirely separate single-phase circuits with impressed voltages of similar frequency and magnitude, but 90° out of phase. If the separate currents were utilised in a machine or furnace and kept distinct throughout, it is obvious that four wires would be required to transmit the two currents. Two such circuits A and B, in which single-phase currents are being generated 90° out of phase, are represented in Fig. 43. If, for example, the voltage across A is at its maximum positive value of 100 volts at terminal D and just beginning to fall off, then the voltage across the terminals of B, if lagging by 90°, will be zero and just reversing to a positive sign at one terminal, say D′. In practice it is customary to connect one terminal of each circuit together, and to connect the remaining "outer" terminals to two corresponding terminals of the power absorbing circuits. The latter are likewise connected together at a common terminal, which is connected to the corresponding terminal of the generating circuits by a common return conductor (Fig. 46). In this way only three conductors are used in place of four, which constitutes a saving of copper, although the "neutral" return conductor carries a heavier current than that flowing through the individual circuits. In Fig. 44 the heavy dotted line EFGH shows the resultant current wave of the two current waves CD and C′D′ for the circuits A and B in Fig. 43. The effective value of this curve equals $\sqrt{2}$ its crest value, and is $\frac{\sqrt{2}}{2}$ times the sum of the effective current values of the wave curves CD and C′D′.

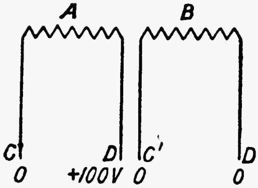


FIG. 43.

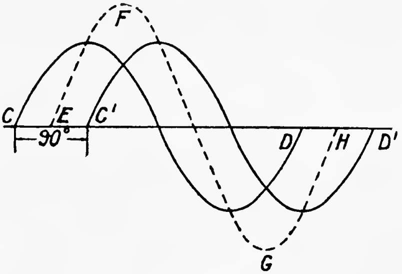


FIG. 44.

Referring again to Fig. 43, if the terminals C and D′, or C and C′ were connected together to form the terminal of the return conductor, the resultant effect for furnace work would be the same. It should be carefully borne in mind that such interchangeability of terminal connections does not apply in the case of three-phase circuit combinations.

It is equally possible to compound two single-phase circuits which differ in phase by 60° in place of 90°. Two such single-phase circuits may be obtained, for example, by taking current from two separate phases of a three-phase system, as was suggested by Girod in 1910.

The various systems of two-phase connections adopted for furnace operation will now be mentioned in the order in which they have been successively introduced.

Heroult Two-phase System.—It was proposed in 1905 to use two-phase current for supplying power to the electric mixing furnace patented in that year, in which case the circuits would be arranged as shown by Fig. 45. As a three-phase system presented considerable advantages over the two-phase, the latter was never applied in practice, but is interesting in view of recent furnace development.

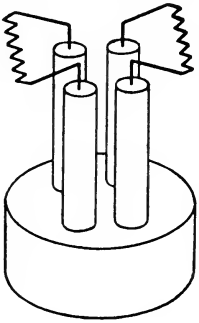


Fig. 45.

Electro-metals Two-phase Three-wire System.—The two-phase low tension circuits are represented in Fig. 46 by the windings A and B drawn at right angles, according to convention, showing them to be 90° out of phase. The two phases are connected at the common neutral point O, from which a neutral conductor goes to an electrode in the furnace bottom. Each upper electrode is connected to the outer terminal of one phase. The hearth only becomes a conductor when hot, and will then carry practically the full return current equivalent to $\sqrt{2}$ of the current flowing through each electrode when balanced. When cold the bottom is non-conductive and the two electrodes then operate in series. The two phases A and B, which have similarly only a series connection, combine to produce a resultant effect equivalent to a simple single-phase power generating circuit, the voltage across the electrodes being 1·41 times the voltage across either phase winding A or B.

The two-phase low tension supply may be generated for each furnace by generators, but is usually transformed down by static transformers to the required voltage from either a two or three-phase high tension system; in the latter case the transformer windings are connected in the well-known Scott method, which gives perfect balance on the three supply phases, provided the secondary side is properly loaded. Figs. 46 and 47 show the loading on the primary side of a three-phase system after and before the hearth becomes fully conductive (Hill and Fleming).[1]

Girod Two-phase Connection (1910).—Instead of using two-phase current 90° out of phase, Girod proposed to transform from two phases only of a three-phase system. Normally the two low tension phases will be 60° apart if connected together symmetrically, but by reversing one phase, the phase displacement will then be 120°. Consequently, the low tension two-phase currents will be 120° out of phase in place of the usual 90° for two-phase systems. By this arrangement (Fig. 48), if the voltage between each top and bottom electrode is 55 volts, then the voltage between the upper electrodes will be 95 volts, and a current will flow not only from top to bottom, but between the upper electrodes also. This results in the load being nearly equally balanced in the three supply phases, when the load is equally divided between the two arc circuits.

FIG. 46.

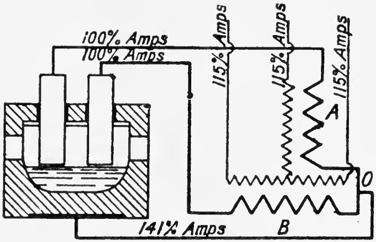
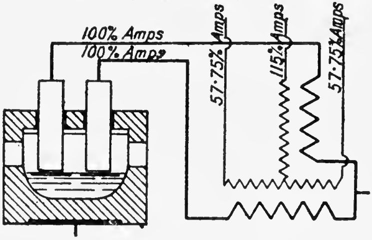


FIG. 47.



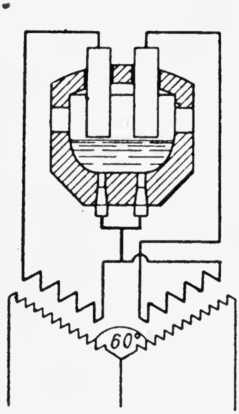


FIG. 48.

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[1] Trans. Faraday Soc., Jan., 1919.

The furnace low tension connections, it will be seen, closely resemble both the Electro-metal and Stobie systems.

Stobie Two-phase Furnace Connections.—In this system the common return wire of the Electro-metals furnace is replaced by separate leads from the two phases to two bottom electrodes, as shown in Fig. 49. A and B represent the two-phase circuits, whose outer terminals are connected to electrodes C and D; the return wires of the phases A and B are separately connected to bottom electrodes E and F respectively, which are embedded in a lining that is conductive only when hot.

This arrangement is specially designed to lengthen the path of current through the bath for the purpose of improving the metallurgical operations by better distribution of heat and circulation of the metal. The return wires are not connected outside the furnace, and there is no neutral point common to the two phases until the lining becomes conductive. Under these conditions the furnace cannot be operated without pre-heating by gas or other means to render the bottom conductive. As soon as the hearth becomes a conductor, the two bottom electrodes are electrically connected, and the conditions are then somewhat similar to those obtaining in the Electro-metals furnace. The path of current, however, from the top to the bottom electrodes must be to some extent different, but the actual direction is problematical, since the bottom electrodes are electrically connected through both the conductive hearth and the bath of metal, which together constitute a neutral point of the two circuits.

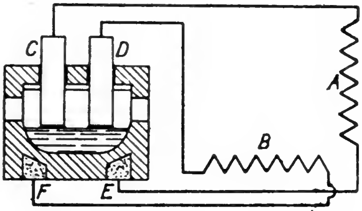

Fig. 49.

Rennerfelt Two-phase Furnace Connections.—Two-phase current is here employed to form two arcs which are entirely independent of the furnace charge or lining for any part of their circuit. The combination of phases is similar to that used in the electro-metals furnace, but the neutral conductor, instead of being connected to a conductive lining, is attached to a third, vertically suspended electrode on to the end of which the arcs strike from two other electrodes. The arrangement is shown diagrammatically by Fig. 50, the two-phase low tension current being transformed from three-phase by Scott-connected transformers.

In certain cases, the neutral electrode C may rest on the furnace charge or actually dip into the bath, when the conditions would be such that two separate direct arcs are formed.

Dixon Two-phase Furnace.—Heroult had originally proposed to split a two-phase current supply into two entirely separate circuits, each having two upper electrodes arranged in series, and thus eliminate all bottom electrode connections. Such a furnace would resemble two single-phase Heroult furnaces assembled in one body. A single-phase circuit with two arcs in series, supplied with power from a source at constant voltage, requires more complicated regulation than if connected to an alternator designed with a "drooping characteristic," since not only must the voltage be equally divided between each pair of arcs in series, but the current flowing through the circuit must likewise be controlled.

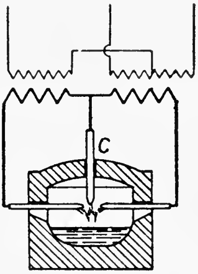

Fig. 50.

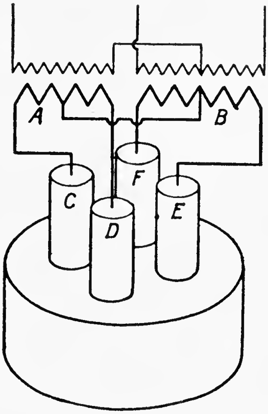

Fig. 51.

The system of connections adopted by Dixon (Fig. 51) was designed to simplify such complicated load regulation of each electrode. The two phases A and B are connected together at their middle points by a conductor, the outer terminal ends of the two phases being connected to electrodes C, D, E, and F. It will be seen that the current leaving any electrode will have more than one return path, so that any variation of the current flowing through one will be distributed between at least two others. By this arrangement the regulation of each electrode can be effected solely by the current flowing in its circuit.

Several methods of compounding two-phase currents supplied to furnaces of the conducting hearth type were introduced at a prior date, and, for a detailed description of the numerous ways adopted, reference should be made to British Specification No. 4742, 1914.

Booth-Hall Furnace Connections.—The electrical features of this furnace (Fig. 122)[1] embody the combined principles of the Rennerfelt and Stobie furnaces. The impossibility of starting a Stobie two-phase conductive hearth furnace when cold has been mentioned, and to overcome this objection a third electrode is introduced, which serves as a neutral return for the current flowing through the phase electrodes. When starting to melt, this auxiliary electrode rests upon the charge, and is later removed as soon as the hearth is sufficiently conductive to carry the full return current.

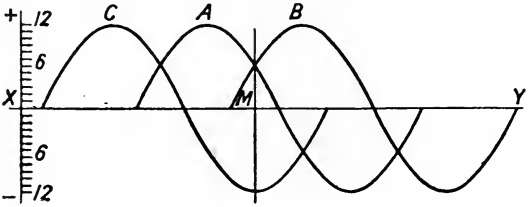

Fig. 52.—Three-phase wave curves.

Application of Three-phase Currents.—It has been shown how two-phase currents may be compounded to produce a resultant flow in a neutral return wire of a three-wire system. The same case applies to three-phase currents, which, however, may be compounded in several different ways.

Usually three-phase currents are primarily generated 120° out of phase, and the voltage curves may be represented by Fig. 52. Three such separate single-phase currents may have entirely independent circuits throughout, and in that case three distinct pairs of wires would be required to carry the separate

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[1] "Met. and Chem. Eng.," Vol. XXIII, p. 212.

currents from the point of generation to a furnace, or other power-absorbing apparatus, and back again.

The curves A, B, and C represent three voltage wave forms of equal magnitude and frequency, but 120° out of phase. At any given moment M on the time axis XY, there will be definite values of impressed voltage for each circuit, which are represented in Fig. 53 as being + 6, + 6, and − 12 for the circuits A, B, and C in reference to the imaginary scale indicated in Fig. 52. The voltage across circuit A is falling, that across B rising, and that across C is at its maximum negative value, but beginning to fall towards zero prior to reversing in sign to positive.

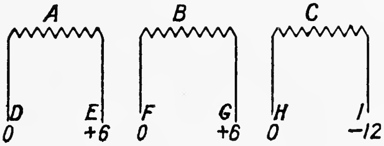

FIG. 53.

The algebraic sum of three such impressed voltages at any given moment is always zero.

Now supposing the terminals D, F, and H are connected together, then the circuits A, B, and C may be represented in the conventional manner as shown in Fig. 54, demonstrating them to be 120° out of phase. The point where the three terminals D, F, and H join is known as the "Star point," and the circuits are said to be "star connected".

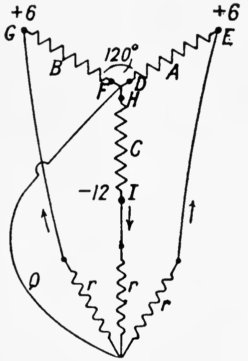

FIG. 54.

If equal resistances *r* are put in circuit between the outer terminals E, G, and I and a common conductor Q which connects the three resistances to the "star point" of the circuits A, B, and C, then at a given moment a current will flow through each circuit whose magnitude and direction is exactly proportional to the magnitude and sign respectively of the impressed voltage. If the voltages at the terminals E, G, and I are + 6, + 6, and − 12 respectively at a given moment, then currents will flow through the circuits ED, GF, and IH in the direction of the arrows, and with magnitudes of 6, 6, and 12, supposing the resistances *r* are each equal to unity; in this case the current through HI will act as the return for the combined currents DE and FG. This is equally true at any other moment when the resistances *r* are equal, and the result is that no current flows through the neutral conductor Q. This means that the "effective" or measured currents, as opposed to instantaneous values, are also equal, and under such conditions the load is said to be "balanced".

Now supposing the resistance *r* in series with the circuit C is increased, then the current flowing in that circuit will be reduced, and will be less than the sum of the currents through the circuits A and B at the particular moment previously considered. The neutral conductor Q will then act as a return for part of the current flowing through A and B, and the system will then be "out of balance," and the three circuits A, B, and C will not be carrying the same "effective" or measured current.

The main line wires of a star-connected power-generating system are connected to the outer terminal of each phase, and may be connected to a three-phase apparatus absorbing the electrical energy so that each power-absorbing circuit of that apparatus is in series with one line wire, the three outer ends of the circuits being themselves connected to a common star point.

It will be seen that the three resistances *r* in Fig. 54 are so connected to a common neutral, and are, therefore, star-connected.

In all three-phase direct arc furnaces the electrodes or line conductors are similarly "star" connected through the arcs to a common star or neutral point, which is the metallic charge or bath of molten metal. In furnace work the charge or bath, which so constitutes the neutral point, is sometimes connected through either a conducting hearth or bottom electrode to a neutral return conductor, which is itself connected to the common point of a star-connected generating system. As has been explained, this neutral return conductor only carries current when the load is not equally distributed or balanced between the three power-absorbing circuits.

If, in a balanced three-phase star-connected system, the effective or measured voltage across the terminals of each power-generating phase—or, in other words, between the star point and the outer terminals of each phase—is equal to 100 volts, then the "line voltage," or the voltage between any pair of the outer terminals, will be $\sqrt{3}$ times 100 volts or 173 volts.

If the three power-absorbing resistances such as arcs are star-connected, as is the case in all direct arc furnaces, the voltage across each resistance or arc will be the "line voltage" $\div \sqrt{3}$, when the load is equally balanced. Knowing this relationship between line voltage and arc voltage, it is easy to calculate the total load on a three-phase star-connected direct arc furnace, if the current flowing through each of the three arcs is known and is the same for all, which indicates balance. Thus, if a furnace is supplied with power from transformers, the line voltage will be invariable for given loads, and the arc voltage will be this voltage divided by 1·73. The total load in K.V.A. will then be the current flowing through each arc multiplied by the arc voltage × 3. This rule only requires knowledge of the line voltage and the current flowing through each arc, and is quite independent of the transformer grouping. Fig. 55 illustrates the nomenclature used, and shows the low tension generating phases star-connected.

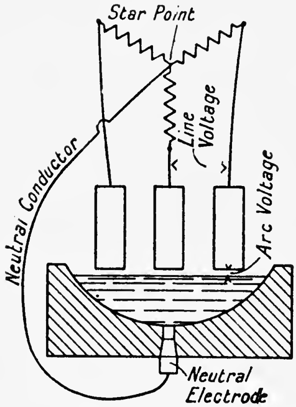

Fig. 55.—Nomenclature diagram.

Delta or Mesh Grouping.—Referring again to the instantaneous voltage values of the three wave forms A, B, and C (Fig. 52) at a moment M on the time axis XY, the three impressed voltages may be graphically represented by a clock diagram (Fig. 56), which shows their instantaneous magnitude and phase displacement. Since the amplitudes, which represent the maximum voltages of the three wave forms, are equal, the radii OD, OF, and OG are equal.

At the moment chosen for setting out the clock diagram the voltage magnitudes of A and B are equal, and therefore each half the magnitude of C. The vertical distances of the points D, F, and G from the horizontal axis is a measure of the magnitude and sign of the voltages at a particular moment, and, if the radii are swinging in a clockwise direction, the magnitude of the voltage in the circuit A is falling, that in B rising, and that in C at a maximum negative value, but falling towards zero. The radii OD, OF, and OG, as set out in the diagram, graphically fulfil these conditions, and, if rotating, may be assumed to represent the constantly varying voltage conditions in the three circuits A, B, and C (Fig. 57). At the particular moment shown in the clock diagram it is assumed that the instantaneous voltage values at D, F, and G are + 6, + 6, and − 12 relatively to the terminals O′, O², and O³ respectively.

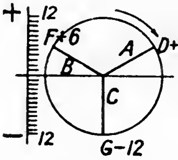

FIG. 56.

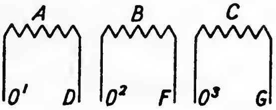

FIG. 57.

The impressed voltages in the circuits A and B both have a positive sign at the terminals D and F, when O′ and O² are connected together; supposing now the terminal O′ is connected to terminal F, it is obvious that it is equivalent to reversing the sign of the voltage impressed in the circuit B, relative to the circuit A, or, in other words, to reversing the circuit B through 180°. Therefore, if B were originally lagging 120° behind A, it will now lead A by 60°, and the radii O²F and O′D may be now shown graphically in their correct relationship by the lines DO′, FO² of Fig. 58, assuming always the radii O′D, FO² to be rotating in a clockwise direction.

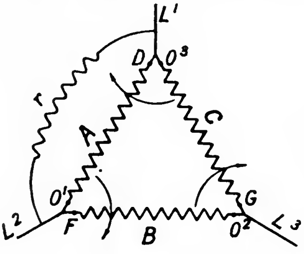

FIG. 58.

In the same way, when the terminals O² and O³ are connected together, the voltage at terminal G is negative, and that at F positive; supposing now that the terminal G is connected to

O^2, then it is again equivalent to reversing the sign of the voltage impressed in the circuit C, relative to the circuit B, and instead of lagging 120° behind B it will now lead B by 60°. By drawing a line GO^3 from the point O^2, to represent the new relationship of the circuits C and B, a closed triangle will be formed which represents graphically Delta or Mesh grouping of three-phase circuits.

The circuits A and B at the moment considered have instantaneous voltage values of $+ 6$ at the terminals D and F, and the circuit C a value of $- 12$ at the terminal G; then, on considering the magnitude and direction of current flow through each circuit resulting from their impressed voltages, it is evident that the circuits A and B together oppose and neutralise the effects of C. This applies at any other given moment, so that three circuits generating alternating voltages, similar in magnitude and frequency but differing in phase by 120°, may be connected to form a closed ring circuit without causing any current to flow. If three line wires L', L^2, and L^3 are taken from the common terminals DO^3, FO', GO^2, and a resistance r, be connected across any one pair L' and L^2, then a current will flow due to the voltage impressed in the circuit A. This also applies to any other resistance circuit connecting other pairs of line wires. If the outside resistances between each pair are equal, then the effective current flowing through each phase A, B, or C will also be equal, when they are said to be balanced. The resistance or load circuits, as distinct from the generating circuits A, B, and C, when connected across the line wires are themselves "mesh" connected.

The resistance or load circuits might also be connected in series with the line wires, instead of across them, and then joined together at a common star or neutral point. This case is analogous to a three-phase direct arc furnace, where three arcs strike on to a charge or bath which constitutes a star point, the line wires being taken from three mesh-connected generating circuits.

With this combination of mesh and star connections, the effective current flowing through each line conductor will be 1·73 times the current flowing through each generating phase, provided the three phases are balanced. The line or, in this case, the generating phase voltage will be 1·73 times the voltage across each arc or other similarly grouped power-absorbing circuit.

Inverted Star Grouping.—It has been shown how three-phase circuits may be connected together in a simple star grouping, represented graphically by three equal radii 120° apart.

Now if one of the circuits is disconnected from the neutral point, reversed and again connected to the neutral point, it will no longer be 120° out of phase with the other two, but will lead one and lag the other by 60°. This, perhaps, may be more simply understood by reference to a wave form diagram. In Fig. 59 three equal alternating voltages 120° apart are represented by their wave forms A, B, and C; now, supposing the wave form C is reversed, which would result by reversing the connection of the phase C relative to A and B, then the wave C′ will obviously be introduced into the system in place of C. It is plain, also, that the phase displacement between A and C′, B and C′ is 60°.

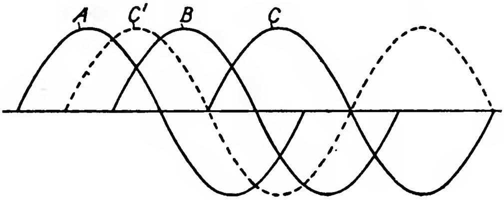

FIG. 59.

The simple inverted star connection has been used for furnace operation, but in its more recent applications a special modification is adopted.

Suppose a circuit represented by the wave form C, after disconnection from the common star point, were split into two parts, then the impressed voltage in each portion would be in the same phase, and might be represented by two separate wave forms of different magnitude, whose sum exactly equalled the original wave form C. Either component might be again star-connected to the other phases A and B so as to be 120° apart, while the other component might be reversed before connecting again to the star point.

The graphical figure would then be a combination of a simple star and inverted star connection.

After this brief description of the commonest methods of compounding three-phase alternating current circuits, their application for furnace operation may now be considered.

Stassano Three-Phase Furnace.—This furnace, being dependent upon indirect arc heating, is provided with three nearly horizontal electrodes, which converge towards a common centre and are set 120° apart. Arcs are struck between the electrode ends, and form an equilateral triangle whose apices are the electrode points. By this formation of the arcs, the electrodes may be regarded as mesh connected at that end where the electrical energy is being converted into heat in the arc gaps. The current flowing through each arc, when all three circuits are balanced, is then equal to the current flowing through any electrode divided by 1·73, so that the load in K.V.A. taken by each arc $= \frac{A}{1{\cdot}73} \times V \div 1000$

where A = current flowing through any electrode
V = arc voltage, which is in this case the line voltage

The total load in the furnace is thus represented by the equation—

$$\text{K.V.A.} = \frac{A}{1{\cdot}73} \times V \times 3 \div 1000$$

provided the value of the current flowing through each electrode is the same.

Heroult Three-Phase System.—When static transformers are used for supplying power, the low tension windings are usually mesh connected, as shown in Fig. 60. They may, in certain cases, be connected to form a "star" grouping (Fig. 54), but without the use of a neutral return circuit from the furnace hearth to the "star" point. These two methods of grouping are sometimes made interchangeable by means of special switchgear for the purpose of effecting a considerable variation of line voltage. The high tension windings are equally well connected in either "star" or mesh grouping, which can likewise be made interchangeable.

The outstanding feature of the Heroult system is the absence of any hearth connection, so that the supply of energy to the furnace charge is entirely independent of the lining. The three arcs strike on to the charge or bath, which serves as the common star point, and are thus star connected. When the electrode circuits carry the same current the total load on the furnace in

$$\text{K.V.A.} = \frac{V}{1{\cdot}73} \times A \times 3 \div 1000$$

where A = current flowing through the electrodes
V = line voltage or voltage between any two electrodes.

The load in K.V.A. taken by each arc is equal to the current flowing multiplied by the arc voltage and divided by 1000, which is represented by $\frac{V}{1{\cdot}73} \times A \div 1000$ in the above equation.

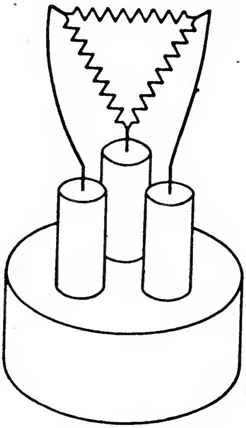

Fig. 60.

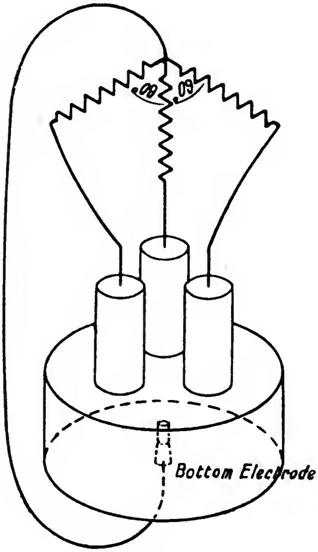

Fig. 61.

Three-Phase Four-Wire System.—This system generally entails the use of a bottom electrode connected by a neutral return conductor to the star point of the supply circuits, as shown in Fig. 55, and has been used by Keller, Girod, and Giffre in various modified ways.

Girod Three-Phase Inverted Star Connection.—This method of connecting the three supply circuits is shown in Fig. 61, where it will be seen that one phase of a simple star-connected system has been reversed. Girod adopted this method in order to increase the current flowing through the bottom electrodes. It is very doubtful whether this was a wise step, and it will be seen later how the modern tendency is rather to restrict the current flowing through the furnace hearth without unbalancing the supply phases:

Stobie Three-Phase Four-Wire System.—The simple case of a four-wire three-phase grouping has already been mentioned, where the neutral return was connected between a bottom electrode and the star point. According to Stobie's method the neutral return is taken from the star point and connected to an upper electrode similar in mechanical operation and dimension to the other three electrodes placed in the line conductor circuits. This fourth neutral electrode carries any out-of-balance current when the three main electrodes are unbalanced, but is otherwise inoperative.

Greaves-Etchells Three-Phase Systems.—In all the previously mentioned methods of supplying direct arc furnaces with power from three-phase circuits, with equal line voltages, an electrode has been interposed in each line conductor so as to produce three arcs star connected to a common point, which is the metallic charge or bath of metal. It has also been shown that the power supply circuits are only balanced when such a star-connected load is equally divided between each line resistance or arc. With three arcs the load can be balanced by adjustment of the arc length, but if the resistance of one arc is fixed, then balance could only be obtained by making the resistance of each of the other two equal to it. In the Greaves-Etchells furnace only two electrodes are used for the purpose of forming arcs, and one fixed electrode is imbedded in a hearth of variable resistance. The hearth resistance may be regarded as being substituted for one arc resistance, but, instead of being adjustable, is dependent solely upon the conductivity of the material used for its construction at different temperatures. The essence of all the Greaves-Etchells patents lies in the special methods of transformer design and grouping, so that a balanced load can be obtained on the primary supply phases when the hearth resistance does not necessarily equal the arc resistances. Provision is also made so that these conditions of balance may be possible over a wide variation of hearth resistance, provided that each arc carries the same current.

A full description of the several methods of transformer grouping possible is beyond the scope of this book, as it can only be comprehensibly studied by the use of complex vector diagrams.

Application of Four-Phase Currents.—In Fig. 62 are represented the wave curves A and B of two single-phase currents 90° out of phase. The curve A may be split up into two equal components C and C′, which, owing to their equality, are superimposed on the diagram. Supposing terminal connections are so made that C′ is reversed relatively to C, then there will be two single-phase waves 180° out of phase, and if the same be applied to the wave B to form two components D and D′, of which the latter is reversed, there will then be four curves 90° apart, arranged in the order C, D, C′, and D′. If these four circuits were star connected, they could be represented in a clock diagram by equal radii 90° apart, which would denote all the relations of one phase to another. It is quite obvious that the algebraic sum of the impressed E.M.Fs. at any moment is zero, so that, if the four individual phases were connected in ring fashion, just as in the case of three-phase circuits, no current would flow. If, however, any outside circuit were connected across any two points on the periphery of the ring, then a current would flow through that circuit.

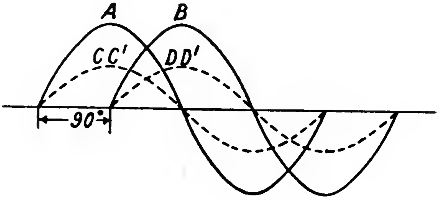

Fig. 62.

The earlier types of two-phase furnaces were designed for small capacities up to about five tons, and for this size two upper electrodes, acting in conjunction with one or more bottom electrodes, are sufficient to convey the required amount of electrical energy to the furnace. With the growing tendency towards larger units, it has become imperative to increase the number of upper carbon conductors, with the logical introduction of four-phase currents. It has already been seen how two-phase currents could be applied in the case of four upper electrodes without a neutral return, and it now remains to describe other methods by which two-phase currents, which have been compounded into four-phases by splitting each phase and reversing one-half of each, can be utilised in either mesh or star grouping (with or without neutral return conductors).

All designers of bottom-electrode furnaces have appreciated the importance of varying the return current without unbalancing the primary phases, and it will be seen how this feature has been introduced into some of the systems of four-phase grouping.

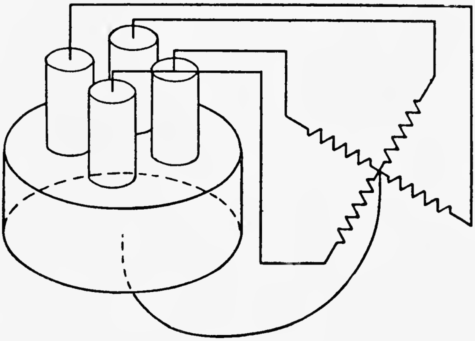

Fig. 63.

Dixon's Four-Phase Star Grouping.[1]—The diagram of connections (Fig. 63) shows a simple star grouping of four-phase circuits, the " star " point being connected by a neutral return conductor to a bottom electrode embedded in a conductive hearth. The four-phase current is derived from the two-phase circuits which form the secondary windings of two groups of Scott-connected transformers. By means of switches in the three-phase primary line conductors the two-phase secondary windings of one transformer group can be made either 180° or 120° out of phase with the other. In the first case the four-phase circuits are 90° out of phase, and no current will flow

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[1] British Patent Specification, No. 4742, A.D. 1914.

under balanced loading through the neutral return. Various other modifications have been introduced for further varying the phase relationship of one pair of circuits relatively to the other, this being done for the purpose of varying the current flowing through the furnace bottom, while still maintaining a balanced loading on the primary phases. The system of connections shown in Fig. 63 is used on a Gronwall-Dixon 5-ton furnace operating at Detroit, U.S.A.

Dixon's Four-phase Mesh Grouping.—This method is far less complicated than the above and consists of a simple ring connection shown graphically in the form of a square (Fig. 64). From each corner is branched off a conductor, which conveys current to an electrode. The load on the supply phases is balanced when the current flowing through each electrode is equal. This system may be regarded as the four-phase counterpart of the simple Heroult three-phase mesh connection.

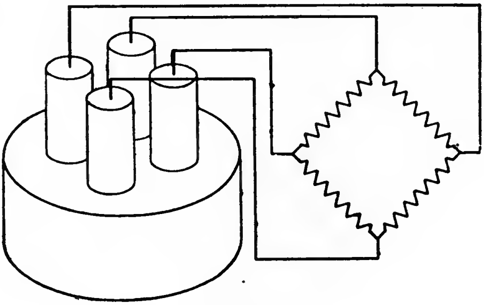


FIG. 64.

Electro-metals Four-phase Five-wire Grouping.—This system of connections embodies a special arrangement of the three-phase secondary windings of a transformer group. Five line conductors are required, four of which are connected to upper electrodes and one to a conductive hearth. It is also a feature of this method that the bottom electrode conductor takes only little more current than any one of the upper electrodes under balanced loading. A full explanation of the manner by which the above conditions are accomplished is only possible by resorting to complicated vectorial or mathematical solutions, so that it is only proposed here to convey the general principle, and to point out the fundamental difference between this and the other methods of grouping previously described.

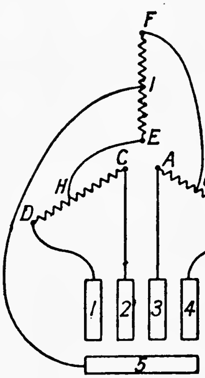


Fig. 65.

The three secondary windings are split up into parts, and are connected to four upper electrodes and a bottom electrode in the manner shown by Fig. 65. AB, CD, and EF represent the three-phase secondary circuits of a transformer group. The transformer windings AB, CD, and EF are respectively tapped at the points G, H, and I. The points G and H are connected to opposite ends of the windings EF, and the point I is connected to the bottom electrode 5. A and B are connected to upper electrodes 3 and 4, and C and D to 2 and 1 respectively. It is clear that the voltage between the bottom electrode or the point I and any one upper electrode is always the resultant of the voltages induced in one section of either winding AB and CD, and one-half of the winding FE. The wave curve representing the voltage between electrodes 1 and 5 is the resultant of the wave curves of the impressed voltages across the windings HD and EI, which are different in magnitude and 120° out of phase. In Fig. 66 the curves A and B, 120° apart, represent the impressed voltages across these windings EI and HD; curve C is drawn by plotting points whose distances above or below the line XY are equal to the algebraic sum of the vertical distances of the curves A and B from XY at several given moments. The curve C represents the resultant of curves A and B, both as regards magnitude of impressed E.M.F. and phase. Since

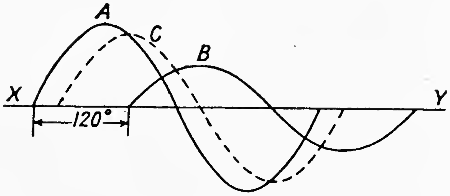


Fig. 66.

there are four distinct electrode circuits built up of two unequal windings in series, and whose phase relationships are in every case different, there will be four resultant curves, and, therefore, a four-phase system, in which, however, the phases are not 90° apart, as is usually the case. The magnitude of each resultant voltage curve must be the same, otherwise the load cannot be properly balanced, and it is for this reason that the windings AB and CD are both unequally divided.

In the diagram the points E and H are connected together, and the voltage curve of EI will then be 120° in advance of HD, assuming EI and HD to be moving about a centre H in a clockwise direction, this being the convention followed throughout. Now EI and HC are similarly connected, but in this case HC is 60° in advance of EI, which is evident since the voltage induced in HC is exactly opposite in sign to that induced in HD in their relationship to EI. The relative phase displacements of the four resultant curves are unimportant, so long as their magnitude is equal. Since the phase displacement between the windings HD, EI, and HC, EI are different, then the only way in which the magnitude of the resultant can be made the same is by making the voltages across HD and HC different, and to effect this both windings AB and CD are unequally divided.

By this method of grouping the secondary windings of a three-phase system can be so arranged that the voltages between four electrodes and a neutral bottom are similar, and the current through the neutral return is nearly equal to the current through each electrode when balanced.

CHAPTER IV.

GENERATION AND CONTROL OF SINGLE AND POLYPHASE CURRENTS.

Single-phase Installations.—The early types of electric furnaces operated on single-phase current supply, and usually derived their power from alternators attached to individual units. Power could equally well be taken from single-phase supplies generated at extra high voltage, and transformed down by static transformers. In this latter case, if the normal furnace load is only a fraction of the total power generated in the system, a heavy load fluctuation will not be accompanied by any serious diminution of the line voltage, and, provided the static transformer is suitably designed, heavy overloads may occur without endangering any part of the electrical installation. This, however, does not apply to alternators separately connected to furnace units, and special provision has then to be made to prevent heavy sudden overloads which might prove disastrous, not only to the alternator, but also to its prime mover.

Single-phase current is seldom generated by Power Companies for local consumption in high power machinery, or for long distance transmission other than for traction purposes. For this reason it is generally necessary to install special generating plant for single-phase furnace operation, since polyphase currents cannot be satisfactorily transformed to single-phase by static transformers. This adds considerably to the capital cost of the installation, and, where motor-generator sets are used, a total loss of about 15 per cent. of the power consumed is incurred. The problem of supplying power to single-phase furnaces stands out as a predominant objection to their use, unless, as is rarely the case, the Power Supply Co. allows the furnace load to be taken from separate phases of a polyphase system.

Single-phase Generators.—Alternators and their prime movers, supplying power to separate furnaces, are usually rated at normal full load capacity, and must be designed to prevent, automatically, heavy overloads which might otherwise prove a constant source of breakdown. The power absorbed in an alternating current circuit is the product of volts and amperes (volt-ampere) multiplied by the power factor, and a variation of any one of these multiples will cause a change in the effective power. It has been explained how the effect of the self-induction of a circuit, which depends partly on the strength of the current flowing, produces an opposing E.M.F. or voltage and at the same time causes a lowering of the power factor, and it is this very property that is utilised in the design of generators supplying current subject to sudden and heavy fluctuation. The alternator is constructed to have a "drooping characteristic," which denotes a rapid falling off of the terminal voltage so soon as the current in the circuit rises above the normal; the power capable of being developed is thereby automatically restricted by a reduction of the voltage, although the actual current will rise, but not to the same extent as if the normal line voltage were maintained. An exciting dynamo is generally mounted on the generator's shaft and is itself excited from an independent source, or it may be self-excited. The excitation current for the field windings of the alternator is controlled by varying the strength of current flowing through the field of the small exciter, and by this means the alternator may be regulated to maintain a constant voltage over a range of different current outputs. In effect then, the alternator may be regulated to supply power at the same voltage for any desired current within its capacity, any increase above this current giving rise to a drop of voltage. Regulation of the furnace load is effected by maintaining the correct terminal voltage, which is done by adjustment of the electrodes. It is also customary to provide the generating set, whether the prime mover be an electric motor, steam or gas engine, with a considerable flywheel effect, which, by its capacity for storing or giving out energy, greatly minimises the result of load fluctuations upon the prime mover.

Miles-Walker Converter.—A special type of three-phase to single-phase converter has recently been introduced for furnace operation by Professor Miles-Walker, which takes the form of a rotating balancing transformer. The converter takes H.T. power from a three-phase supply at unity power factor, and delivers L.T. single-phase current without unbalancing the supply phases. As in the case of single-phase alternators, the balancer is designed to have very considerable self-induction in the secondary or low tension circuit, so that at normal full load the current curve will lag 45° behind the voltage curve, giving a low power factor of ·7. With this arrangement it is impossible on dead short circuit to obtain a current greater than 1·41 times the normal full load value, and this current rise is accompanied by an actual falling off of the power output. The single-phase terminal voltage will steadily decrease when increasing the load from open circuit to normal full load, after which it very rapidly falls away. The load during operation may be varied up to, but not beyond, full load current by lengthening or shortening the arc.

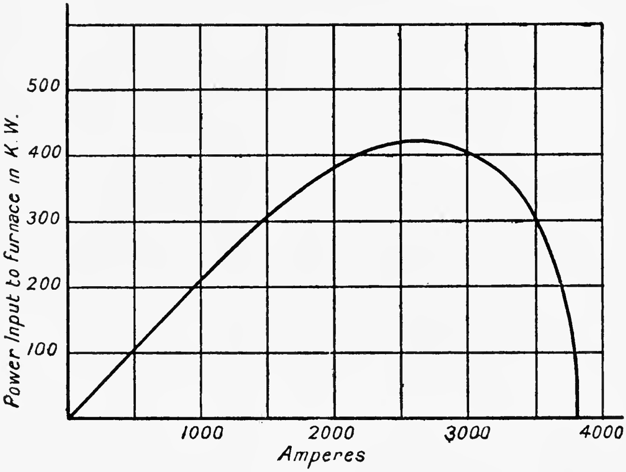


Fig. 67.

The efficiency of transformation is 90 per cent., showing a gain of about 5 per cent. over motor generators. The curve shown in Fig. 67 shows for a specific case how the power output falls off as soon as a full load current of 2700 amperes is exceeded. These balancers have already been in operation in England, and should prove a great advance on the motor generator sets previously used.

Single-phase Static Transformers.—Single-phase static transformers are almost universally used in furnace installations for transforming the high tension voltage down to a voltage suitable for furnace use, and are used either singly or grouped together for operation on single or polyphase supplies respectively.

Power transformers for furnace work require to be specially designed to meet the severe conditions of service, and the following remarks upon their construction apply to single-phase transformers used either singly or grouped together for a polyphase supply.

Internal Reactance.—Direct arc furnaces, when melting cold scrap, are subject to momentary overloads which may even at times approach a dead short circuit, and under such conditions violent surging is likely to be set up unless special provisions are taken to prevent it. There is also considerable risk of the transformer windings becoming displaced by powerful magnetic forces set up by heavy current overloads, and when this happens breakdown of the insulation is likely to occur.

It has been previously explained how the self-induction of a circuit acts as a limiting factor to the degree of sudden current variation, so that by suitable design the degree of current fluctuation and its harmful effect may be considerably reduced. To effect this transformers are usually designed with 8 per cent. to 15 per cent. internal reactance, which is introduced by allowing flux leakage in the main magnetic circuit. The internal reactance of a transformer is regarded as the percentage ratio of reactive volts, at normal full load, to the impressed volts.

Heating.—The fluctuating character of a furnace load precludes the use of any automatic device which will open the switch on other than heavy overloads sustained for a period of one second or more. Nothing, therefore, apart from the care of furnacemen, can prevent a small sustained overload on the transformers, which may be anything up to 15 per cent. or even more over short periods. This demands careful attention from the power transformer designers, who usually make an ample allowance for steady overloads in calculating the temperature rise under ordinary conditions of service. The air-cooled oil-immersed type of transformer is generally favoured, but for the largest units water cooling has been more often used.

The low tension copper bars are frequently interleaved where they emerge from the iron case, and extend at least 18″ away from the nearest point of the shell; any heavy mesh connections can then be safely made between the transformer terminals without fear of local heating in the steel shell.

Automatic tripping devices are sometimes used in conjunction with mercury thermometers, which are immersed in the transformer oil, and complete relay circuits when the temperature exceeds the recognised limit of safety. There can be no doubt that breakdowns are frequently due to prolonged overheating, which in course of time carbonises the oil and causes deterioration of the insulation, so that any safety device, which might at least be used to effect a warning, is a useful accessory in transformer equipments.

Primary Tappings.—In all direct and indirect arc furnaces the arc voltage is to a great extent limited by the damage likely to result to the refractory lining from an exposed arc of considerable length. In the case of direct arc furnaces in which cold scrap is melted and refined, the arc will strike between the nose of the electrode and the charge of scrap or pool of melted metal, and at the same time the heat of the arc will be localised and shielded from the walls and roof by a surrounding wall of unmelted scrap. Therefore, so long as scrap remains unmelted above the bath level, a higher voltage is permissible than when the entire charge is melted and the lining consequently more directly exposed to the arc.

There are certain advantages to be gained by melting at a high voltage, so that it is preferable to work at different voltages during the melting down and subsequent stages of the process. There are various ways of doing this, but the method most generally adopted is to change the connections of the primary windings of the transformer (see Figs. 70 and 71). These windings are tapped at suitable points and connected to separate terminals; by changing the connections between the high tension supply cables and these primary terminals it is possible to alter the transformation ratio and thereby obtain different terminal voltages on the secondary side. The number of tappings is usually restricted to two, which enables either two or three different voltages to be obtained by the use of simple change-over switches.

There are two methods of arranging tappings for changing the transformation ratios, the usual method being to open circuit a number of the coils adjacent to one end of the primary windings, and thus reduce the number of turns. When the voltage variation is small, the number of turns eliminated will be few, so that the point from which the tapping is taken is close to one end of the primary windings. Hill and Fleming have pointed out that at the moment of switching in a transformer there is likely to be a concentration of voltage between the adjacent turns of the primary windings near the ends connected to the line conductors, so that in cases where the tappings are taken out adjacent to one end of the windings there will be a high potential across the terminals of the open circuited portion, which may tend to cause a breakdown of the insulation. To overcome this risk, the primary windings are sometimes split, and tappings are taken out from near the inner end of each half. By suitable switchgear two small sections of the split windings are cut out in place of one large section as in the more usual way. The switchgear for effecting the necessary connections is more complex than the simple change-over switch commonly in use with the system of tapping connection first mentioned.

Transformer Grouping.—The majority of modern furnaces operate on a two-phase or three-phase low tension supply, either of which can be obtained by a simple grouping of single-phase transformers on either two-phase or three-phase high tension systems.

Certain special methods of grouping and splitting up the transformer circuits, peculiar to particular furnaces, have been already dealt with.

Three-phase to Three-phase.—The primary windings of each single-phase transformer may be connected together either in star or mesh grouping, which likewise applies to the secondary windings; star connections of both primary and secondary should, however, be avoided.

Two-phase to Three-phase.—Two-phase current is generally converted to three-phase by the Scott method of transformer grouping. Two single-phase transformers are used in which the primary windings consist of the same number of turns, the secondary windings being in the ratio of 100 to 86·6. Fig. 68 shows the connection of two transformers made so that one end of the secondary winding AB of the transformer "X" is connected to the middle point of the secondary winding of the transformer "Y". The resultant line voltage between A and D and A and C will be equal to the line voltage across DC; in this manner three independent circuits are available in which the impressed voltages are equal in magnitude and are 120° apart.

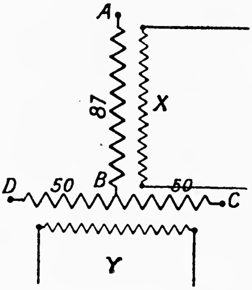


Fig. 68.

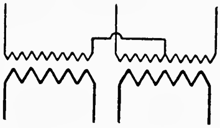


Fig. 69.

Three-phase to Two-phase.—This case is merely the reverse of the above, the transformation being accomplished by designing the windings AB and DBC for the high tension primary circuit in place of the secondary, the ratio of turns remaining exactly the same. The usual diagrammatic method of indicating a Scott-connected transformer group is shown in Fig. 69; this method avoids the confusion of crossed high and low tension line conductors.

Voltage Variation: *Transformer Tappings.*—The provision of tappings taken from the primary circuits of transformers has already been mentioned as a means of altering the transformation ratio and the secondary voltage.

The method of changing the connections between the tappings and the supply cables by means of a special switch depends upon whether the primary windings are star or mesh-connected. Fig. 70 shows a three-phase transformer group mesh-connected to the high tension supply, and the method adopted for changing the secondary voltage by means of a simple two-way knife switch. If a greater range is required, a more complex switch must be used, but generally two voltages are quite sufficient for practical purposes. In Fig. 71 is shown the arrangement when the primary circuits are star-connected; a special switch is required which enables different sets of tappings to be starred, and is exceedingly simple in construction. Change voltage switches are always carefully interlocked with the main oil switch, so that on no account can the primary connections be altered when the transformers are "alive".

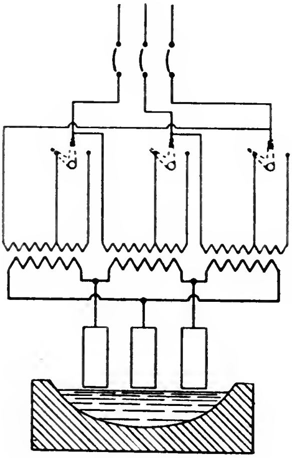


Fig. 70.

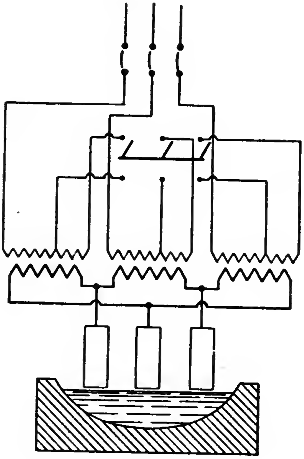


Fig. 71.

By the use of tappings the range of voltage change is usually small, and, where a greater variation is required, recourse is made to alteration of the transformer grouping on the primary circuits. This entails a change from "star" to "mesh" grouping, or vice versa, which, of course, can only be employed on three-phase systems. If the primary windings of three single-phase transformers are mesh-connected, and each has a transformer ratio of 11,000 to 100, then the voltage across each secondary winding will be 100 for an 11,000 volt system. Now supposing the primary transformer circuits are "star" connected, then the voltage across each primary winding will be only 11,000 ÷ $\sqrt{3}$ or 6360, and the voltage across each secondary 57·8, so that in this way the low tension voltage can be reduced in the same ratio of 1·73 to 1. The K.V.A. capacity of the transformer will at the same time be reduced in proportion to the voltage across each primary circuit, but this is seldom a serious disadvantage, as the high voltage arrangement of the transformers would only be used during the melting operation, and the lower voltage for subsequent refining when the amount of power then required is obtainable with the lower voltage grouping. This reduction of available power is unavoidable, since the transformer windings must not be overloaded to counterbalance the voltage reduction.

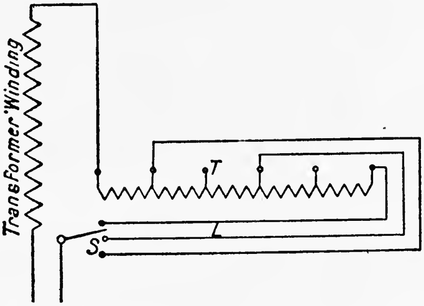


Fig. 72.

Reactance Coils.—In certain types of furnace installations large reactance coils are introduced in either the primary or secondary transformer circuits for the purpose of eliminating load fluctuation. In such cases the reactive voltage drop (at normal full load current) is about 70 per cent. of the line voltage, and the large reactance coils necessary for this purpose can be conveniently used to obtain a small range of voltage variation at the furnace terminals for a given full load current. This is done by varying the number of the windings of the reactance coil, according to the arrangement shown diagrammatically in Fig. 72. In this case a number of tappings T are taken off the windings, any three of which may be permanently connected to leads L passing to a selector switch S. This arrangement enables the reactive resistance in the primary circuit to be varied, so that the actual voltage across the terminals of the furnace is capable of variation. Here again a reduction of voltage results in reduced power since the current must be kept below a fixed maximum. This method of voltage control is not applicable, except in cases where large reactance coils are purposely introduced for controlling load fluctuation.

Reactance or Choking Coils.—A reactance coil may be regarded as a device for introducing or increasing interlinkings of magnetic lines of force with a circuit. It has been explained in Chapter II. that a reactive voltage is self-induced in a coiled conductor carrying an alternating current by virtue of the constantly varying magnetic fields set up. Also, by increasing the strength of the magnetic field, or by increasing the number of turns interlinking the magnetic field, the reactive voltage will be increased if the current flowing and the frequency remain unchanged.

A reactance or choking coil simply consists, then, of a number of turns of a conductor, sometimes enclosing a laminated iron core which may be either closed to form a ring or may be broken to include an air gap. The windings of the reactance coil are connected in series with the circuit in which the reactance is required to be introduced, and therefore form an integral part of the circuit itself. The various power circuits of any arc furnace will include the simple resistance of the conductors, the practically non-inductive arc resistance, and a reactive resistance. The applied voltage across the circuits must, then, be sufficient to overcome the reactive voltage and to force the current through the non-inductive resistance. Since the wave curves of the resistance volts (i.e. the volts forcing the current through the non-inductive resistance, and therefore in phase with the current) and reactive volts are 90° out of phase, the applied volts will not be the simple but the vectorial sum of the two. Since the effect of a reactance coil depends upon an increase of the voltage drop caused by current overloads, it is important in their design to study the reactive drop at full load, and under heavy overloads. Except in special cases the voltage drop at full load is preferably small but increases rapidly under current overloads. To secure this effect the iron core or the magnetic circuit must be so designed that the number of lines of force are increased proportionately to the current flowing; this is only possible so long as the iron circuit remains unsaturated. The effect of a reactance coil so constructed depends upon the ratio of the applied volts across a circuit to the reactive volts induced at normal full load.

If the circuit pressure and the reactive voltage are known for a given current it is easy to calculate the voltage usefully employed in forcing current through the circuit resistance to produce heat. Since the circuit pressure is the vectorial sum of the resistance and reactive voltages which are 90° out of phase, the circuit pressure may be represented graphically by the hypotenuse of a right-angled triangle, the other sides of which represent the resistance and reactive volts. The resistance volts may then be calculated as the square root of the difference between the squares of the circuit volts and reactive volts.

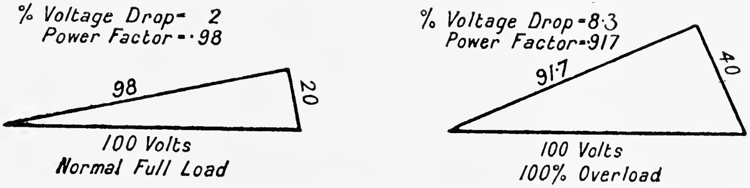


Fig. 73.

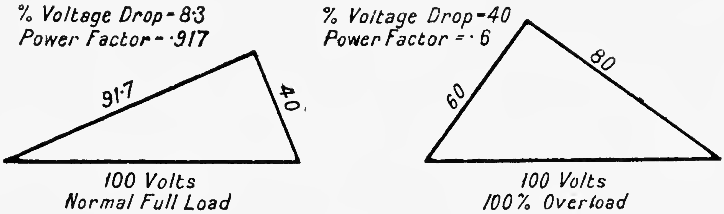


Fig. 74.

The diagrams shown in Figs. 73 and 74 show how the resistance volts are reduced in the case of two circuits by an increase in current from normal full load to 100 per cent. overload; the reactance voltage is in one case 20 volts and in the other case 40 volts at normal full load current, the constant circuit voltage being 100 for both. These diagrams show that the voltage drop increases in far greater proportion than either the current or the reactive volts, and are true provided that the magnetic circuits are such as to allow the number of magnetic lines of force to increase in proportion to the current.

The useful or power-developing current in a self-inductive circuit is in phase with the resistance volts, so that the angle between the sides of the triangle representing the resistance and circuit volts is the angle by which the current "lags" the applied or circuit voltage, and therefore the magnitude of the resistance volts divided by the applied volts; in other words, the cosine of this angle is the power factor. These diagrams also show how the power factor falls on overloads as a result of introducing reactance.

If the fixed voltage across a circuit that includes a reactance coil and a practically non-inductive resistance, such as an arc, is known, it is possible to design the reactance coil to produce a desired reactive voltage for a given current, so that the current fluctuation can be kept within desired limits by the consequent reduction of the volts available for overcoming the non-inductive resistance.

In the case of Snyder furnace installations the reactive voltage is made equal to the resistance voltage at normal full load, so that if the circuit pressure is 100 volts the voltage available for forcing current through the arc resistance is equal

$$\sqrt{\frac{10{,}000}{2}} = 70{\cdot}7 \text{ volts.}$$

With this ratio of reactive to resistance volts at normal full load it is impossible for the current on dead short circuit to exceed 1·41 times the full load current, but this benefit can only be gained at the expense of a low power factor equal to ·707. The curve shown in Fig. 67 is characteristic of the effects of such a reactance.

In some cases the reactance coils contain no iron, the magnetic circuit being in air alone. The advantage of this type is that the magnetic lines of force, and therefore the reactive voltage, are directly proportional to the current; the length of wire used in each coil is, however, much greater than for iron core choking coils.

Reactance coils are used essentially to restrict current fluctuation, and are therefore unnecessary, and in fact undesirable, if the load is steady. Various arrangements have been proposed in which reactance coils are used in conjunction with change-voltage transformer switches, so that they are only included in the power circuits when operating at a high voltage, and at a time when the load is fluctuating.

CHAPTER V.

AUTOMATIC REGULATORS AND ACCESSORIES.

In all arc and arc-resistance furnaces the current flowing through any electrode is dependent, apart from the effects of reactance, upon its circuit resistance, which, being mainly in the arc itself, is always subject to either rapid or slow variation.

The non-inductive resistance of an indirect arc circuit lies in an arc gap, the length of which can only vary by consumption of the electrodes or by deliberate adjustment. The resistance of an arc of given length is also dependent upon the temperature and nature of the atmosphere through which it passes, but if these remain constant the resistance of the arc is also unaltered. For this reason, after striking the arc of an indirect arc furnace, the load will remain fairly steady, and adjustment of the electrodes will only be occasionally necessary to compensate for their consumption and the slowly varying temperature and character of the furnace gases.

The case of the direct arc is very different. Neglecting the minor considerations of furnace gases, the resistance of an arc, and therefore the current flowing through it at a given arc voltage, will depend upon the length of the arc gap as determined by the position of the electrodes, one of which is constituted by the charge being melted or the charge after fusion. During the melting-down operation the length of the arc gap will always be increasing, so long as the metal melted can flow away from the carbon electrode, and this proceeds until ultimately a bath is formed. Therefore, to maintain a constant load it would be necessary, theoretically, to advance the carbon electrode continuously so as to maintain always an arc of uniform length. This is obviously impossible in practice as so many other factors, such as the character of the scrap, the size of electrodes, and the varying resistance to the passage of current offered by the charge itself, influence the distribution of load, so that the problem becomes not merely one of maintaining a constant arc length, but rather one of constantly adjusting the arc length to compensate for a variable resistance in the other parts of the circuit.

If a charge consists of very heavy tightly packed scrap, then the only part of the arc circuit in which the resistance can vary appreciably is in the arc gap itself, and this will remain practically constant so long as the metal melted does not flow away and so lengthen the arc. Such conditions of melting cold scrap are, however, both undesirable and seldom possible in practice, so that during the greater period of the melting-down stage the arc lengths require frequent adjustment, and in fact to an extent that usually renders automatic load regulation desirable in all but small capacity furnaces.

Electric motors are used on all large modern arc furnaces for operating the electrode adjusting gear, and the various types of automatic regulators are all designed to control these motor circuits.

The principle of their operation consists of closing the motor circuits in such a way that the direction of rotation either advances or withdraws the electrode so soon as the K.V.A. load on any arc circuit slightly falls below or exceeds a certain desired figure. In every case the automatic regulator operates by means of an electro-magnetic device, which is actuated by a current that bears a direct relationship either to the current flowing through each arc circuit or to the voltage across each arc itself. The "load" depends both on the current flowing and the voltage across each arc circuit, so that if one of these factors is always constant, it can be regulated in accordance with the variation of the other.

In the original single-phase Heroult furnace, in which two arcs are in series, it is evident that unless the circuit voltage is equally divided between the two arcs the power absorbed in each arc will not be equal; this, of course, follows as a result of the current through both arcs being the same at any moment, since they are in series. In this case, therefore, regulation must necessarily be effected according to the voltage across each arc, but this only ensures equal distribution of load without controlling the magnitude of the load which depends also upon the current flowing. Owing to the difficulties of operating these single-phase furnaces on public supply mains, it is usually necessary to install a special generating plant. The characteristics of suitable alternators, as described in Chapter IV., are such that the terminal or line voltage falls as the current rises and vice versa, and a certain current output corresponds to a fixed terminal voltage. For this reason, voltage regulation is alone necessary to determine the total load and its equal division between the two arcs.

If static transformers are used in place of these generators, then the line voltage would be constant and the current mainly dependent upon the arc circuit resistance. Voltage regulation of the arcs would, under these conditions, only control the division of the total load but not its magnitude, so that adjustment of each electrode according to the current flowing through the circuit is equally necessary. Therefore, in the case of any furnace having two arcs in series and taking power off a constant voltage supply, it is necessary to regulate the electrodes according to both the current flowing through the circuit and the voltage across each arc.

Polyphase furnaces operated by static transformers may be broadly divided into two classes for the purpose of considering the requirements of automatic regulation:—

I. Furnaces in which the voltage across each arc and the current flowing through any arc circuit are entirely independent of the operation of other arcs.

II. Furnaces in which both the voltage across each arc and the current flowing through any arc circuit are dependent upon the mutual operation of other arcs.

To the first class belong two-phase two-arc furnaces of the Electro-metals, Booth Hall and Rennerfelt type, when in their operation the resistance between each arc and the neutral return conductor is negligible, thus rendering each arc circuit entirely independent of the other. In the case of conductive hearth furnaces of the Electro-metals type, this condition is only fulfilled when the bottom is at a high temperature and has very low resistance. Regulation of load under such conditions is then accomplished solely by regulating the electrodes to carry a definite current, the arc voltage remaining fixed within narrow limits and only dependent upon the characteristics of the transformers.

To the second class belong three-arc three-phase furnaces of the Heroult type. Since the three arcs are star connected the arc voltages are equal when the current is equal, but if the condition of balance is upset by the resistance of one arc suddenly being diminished, then the current flowing through that arc will increase and at the same time cause a corresponding but smaller current rise through the other arc circuits ; the voltage across the arc of reduced resistance will also fall, and in this case cause the voltage across the other arcs to rise. This rise of voltage is inevitable since the line voltage is only slightly variable, and accounts for the constant flickering of the arc voltage indicator lamps during melting operations. Regulation by the current flowing through each electrode is generally adopted, but to prevent an overload on one arc from causing displacement of the other electrodes, which may themselves be in correct position relative to the bath, it is necessary to adjust the regulators to operate only outside certain limits of current fluctuation; otherwise, if balance were once upset, all three regulators would be in constant oscillatory operation without ever effecting balance. Further, in furnaces of this class it is possible for an electrode to touch the charge or penetrate the bath without any current flowing until one of the other electrodes completes the circuit, and for this reason automatic regulation dependent upon current flow alone will actually force an electrode downwards until the circuit is completed by one of the other electrodes. To overcome this objection a special form of regulator, which operates in accordance with the voltage between each electrode and the furnace charge, is sometimes employed in conjunction with regulators that are operated according to current flow.

To improve the sensitiveness of automatic regulators it has been proposed to operate them by two electro-magnets drawing current from different sources, the current in one case being dependent upon the current flowing through each electrode circuit, and in the other case upon the voltage across each arc. In this case the regulation of individual electrodes would be more independent of the disturbing effect produced by a varying current through others.

Thury Regulator.—This regulator consists essentially of three distinct parts assembled together on the same framework.

(*a*) An electro-magnetic device actuated by a current which at all times bears a definite relation either to the current flowing through the electrode circuit or to the voltage across each arc. It is also designed so that a moving element of the electro-magnet remains in a position of equilibrium for a definite energising current.

(*b*) A purely mechanical device, which is only caused to operate when the equilibrium of the moving element of the electro-magnet is disturbed by variation of the energising current.

(*c*) Carbon terminals connected to the motor circuits and mounted on the regulator, enabling either one of two possible circuits to be closed so that the motor drive may be in either a forward or backward direction. The proper connections are made to the carbon terminals by the mechanically driven part of the regulator, which also carries carbon terminals connected to the motor circuits.

The above brief description will enable the operation of the regulator shown in Fig. 75 to be more easily understood.

The electro-magnet F is connected to a circuit in which the current flowing always bears a definite relationship to the current passing through the electrode circuit or to the voltage across the arc. The current flowing through the coils of the electro-magnet causes the repulsion of a rectangular copper ring B, which is free to move between the pole-pieces of the electro-magnet F. This copper ring B is attached to a lightly pivoted arm E, which carries at one end a double knife-edged striker C ; to the other end are fixed a light tension spring A and a freely suspended rod, which carries two small collars and terminates in an adjustable piston valve working in a dashpot N. The weight of the arm E and the parts attached are so balanced that the force of repulsion due to a particular current will overcome the tension of the spring and cause the arm to remain in equilibrium in a horizontal position. Movement of the pivoted arm is extremely delicate and responds to small variations of current. The dash-pot N is introduced to prevent the arm responding to momen-

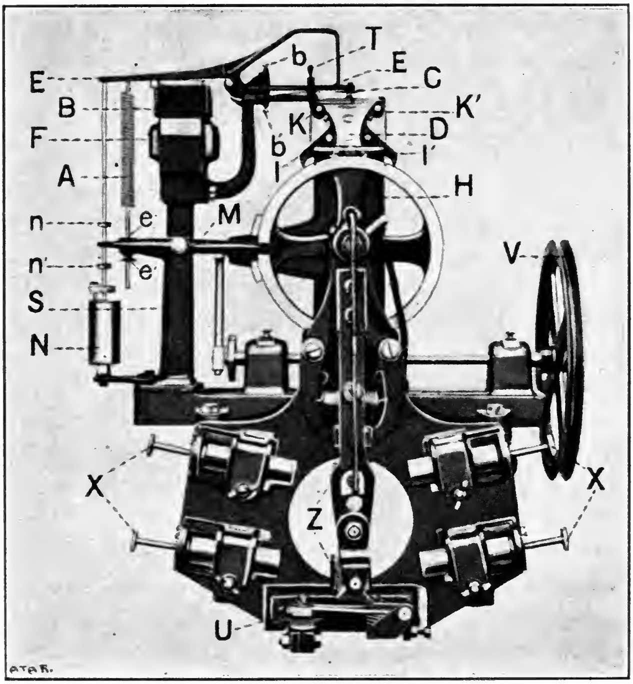


Fig. 75.

[*To face p.* 90.

tary fluctuation of current, and to cause a slow steady movement in either an upward or downward direction instead of sudden jerks. A vertical member mounted on the central spindle is continuously oscillated to and fro by the simple crank drive shown, and the head of this member carries two tappets K, K′ which engage the pawls I and I′. A notched wheel H is also mounted on the same central spindle and actuates a vertically hanging arm to which are connected two insulated carbon terminal blocks Z, Z. The front and lower frame of the regulator carries two pairs of insulated carbon terminal blocks, which, together with the pair on the vertical arm, are suitably connected to the motor circuits.

When the current flowing through the electro-magnet is just sufficient to balance the pivoted arm E in a horizontal position, then the strikers C will just miss the tappets K, K′ every time the latter are oscillated towards and beyond each knife edge respectively, so that no movement of the notched wheel occurs. If the current should increase sufficiently to disturb the equilibrium of the arm E, the striker C will be depressed and engage the tappet K′; at that moment the pawl I′ will have passed the notch during its oscillation towards the left, and being released by the tappet K′ will drop into the notch on its return movement to the right. The notched wheel H will then be advanced in a clockwise direction, and by carrying the vertical arm with it, will close the various circuits to cause rotation of the motor in a direction to reduce the current flowing through the electro-magnet. If this current is in relation to the current flowing through the electrode circuit, then the motor will adjust the electrode to lengthen the arc, but if in relation to arc voltage, the motor will reduce the length of the arc. The time during which the contacts are held may be adjusted by moving the carbon terminals X either towards or away from the central vertical arm and terminal blocks. When the movement of the pawl I′ towards the right has been completed, the direction of movement is reversed and the notched wheel will be restored to its central position; the pawl I′ at the same time will be automatically lifted out of the notch and once more engaged and held by the tappet K′, unless the striker C is still depressed, when repetition of the above movements again proceeds.

The adjustment of electrodes is, therefore, effected by a step-by-step movement, which is intended to prevent the regulation of load being carried too far in the reverse direction, which would then cause continual regulation or hunting. To make this step-by-step movement more pronounced the pivoted arm E, after being raised or depressed to cause movement of the notched wheel, is returned to its mid-position of equilibrium, irrespective of the other balancing forces acting upon it. This is effected by means of a pivoted lever M carrying at one end a pin which projects forward between two studs fixed to the periphery of the notched wheel. The other end of the lever M terminates in a fork, which on its upward or downward stroke pushes the collars *n* or *n′* respectively to a position that brings the pivoted lever E back to its mid-position of equilibrium. This latter movement of the lever M completes the entire cycle of mechanical movements resulting from the release of either pawl I or I′ by means of the striker C and tappets K, K′.

If during one movement of the notched wheel H the current through the electro-magnet is not restored to normal, then the lever E will be again free to move and produce a further cycle of movements. By means of the return action above-mentioned and the dashpot effect which is adjustable, the time interval between each successive step-by-step movement is made subject to control. The small set screws T are adjusted so as to limit the movement of arm E, and so only allow the striker C to rise or fall no further than is just necessary to engage a small V-shaped groove cut in the face of each tappet K, K′. The collars *n* and *n′* should also be fixed to the dashpot rod so that the fork on the lever M just brings the striker C back to mid position at the limit of its stroke.

A furnace regulator must be capable of maintaining a fairly constant load over a wide range, which is generally from full load to quarter load, so that the relation between the current flowing through the electro-magnet device and the current flowing through the electrode circuit must be capable of definite variation.

For purposes of current control a current transformer of fixed ratio is placed in the electrode circuit and connected in series with the electro-magnet windings. The current flowing through the electro-magnet will then be a definite fraction of the current flowing through the electrode circuit, and the regulators would only be capable of maintaining one fixed current value. To increase the range of regulation it is necessary to alter the ratio of the current flowing through the electro-magnet to the current in the electrode circuit, and this is done by placing an adjustable resistance in parallel with the electro-magnet windings. Thus, by varying this shunt resistance, the current generated in the current transformer will be divided between the electro-magnet and shunt resistance circuits according to their relative resistances.

Supposing 5 amperes, flowing through the electro-magnet, are just sufficient to maintain equilibrium, then if this amperage were reduced by shunting a portion through a resistance, equilibrium would be upset and the regulator would operate until the increased load again raised the current through the electro-magnet windings to 5 amperes. In this way, by varying the resistance in parallel with the electro-magnet circuit, the regulator may be set to operate at different loads. A diagram of connections for a Thury regulator controlling an electrode adjusting motor is shown in Fig. 76, which indicates how the circuit connections made in a tramway type controller are completed on the regulator by movement of the two centre terminal carbons; the position of the current transformer relative to the electrode circuit is also shown.

The function of a voltage control regulator is usually to maintain a definite fixed voltage across an arc, and in this respect is only used to divide a load equally between two or more arcs without directly controlling the magnitude of the current flowing; this applies to the regulation of arcs in series, where the current flowing through the circuit is separately controlled by a current control regulator. The Thury regulator is a delicate piece of mechanism and requires very careful adjustment and attention; otherwise correct regulation is not obtained.

Watford Regulators.—The Thury regulator as used on two-phase and three-phase furnaces is essentially a current control regulator, and has the disadvantage that one electrode may be forced on to a charge until its circuit has been completed to permit of the normal current flowing. The Watford superimposed voltage regulator, which overcomes this objection, is purely a voltage control instrument and operates in conjunction with the Thury or other current control type of regulator. This voltage regulator when so used is inoperative until the voltage between an electrode and the furnace charge has fallen below a minimum figure, which might result either from a short-circuited overload, or from contact being made with the charge without any flow of current, as in the case above explained. A sudden drop in the arc voltage causes a switch to open which disconnects the motor from the current control regulator, and at the same time closes another switch which

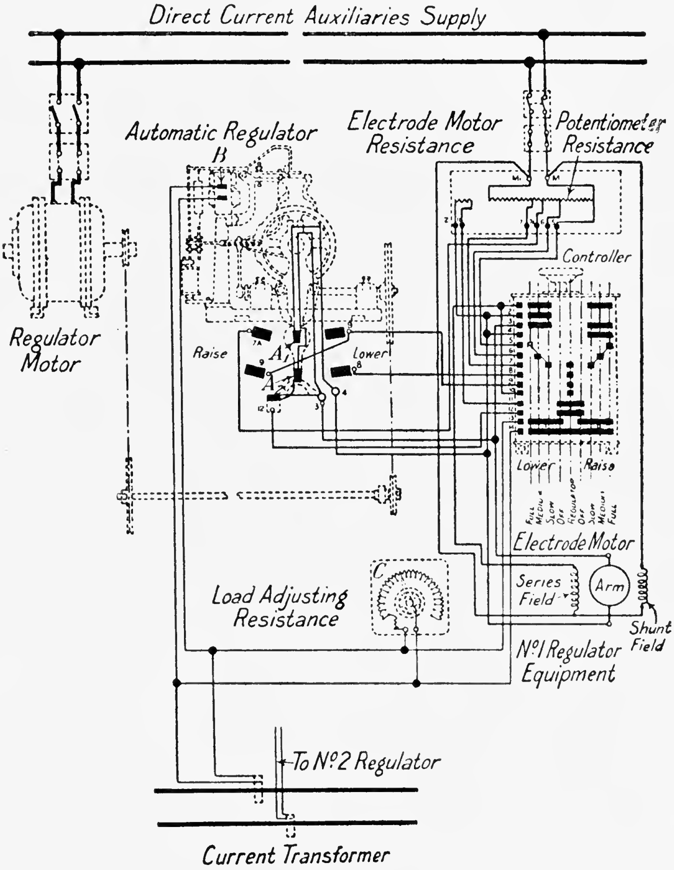


Fig. 76.

re-connects the motor circuits so as to produce rotation in a direction to withdraw the electrode from the charge until the arc voltage is re-established; the withdrawal is continuous and therefore more rapid than with the step-by-step type, so that short-circuit overloads are rapidly corrected. When the arc voltage has been restored, the current control regulator again becomes operative, but only after the motor has been pulled up, which prevents "hunting".

The Watford regulator, as distinct from the superimposed voltage control instrument, performs the dual function of regulating according to the current flowing through each electrode, as in the Thury type, and to a voltage drop below a fixed minimum figure, this latter operation being identical with the superimposed voltage control described above.

The electro-magnetic device, which for current control purposes is connected to the current transformer in the electrode circuit, consists of a pivoted armature capable of rotation between the two poles of an electro-magnet. Two copper bands are recessed in this armature and set up forces tending to rotate it against an opposing mechanical force. The armature assumes a definite position of equilibrium according to the current flowing through the electro-magnet coils. Variation of current causes a slight rotary movement which closes a light relay circuit; this relay then closes another circuit operating a clapper switch, which again closes the motor circuit. According to the direction of rotation of the armature, either one of two clapper switches is thus set in operation, so that the motor is capable of rotation in either direction. By means of other switches and solenoid devices the operation of the clapper-switch is made intermittent, so that adjustment of the electrode resembles the step-by-step movement of the Thury type which prevents "hunting". The voltage regulator is similar in action to that previously described, and operates in conjunction with the current-regulating portion of the apparatus.

General Electric Company, U.S.A., Regulator.—This regulator operates on somewhat similar principles to the Watford instrument, embodying current regulation and no-voltage control devices.

The electrode motors are operated by contactor switches, which are themselves controlled by a contact-making ammeter device or electro-magnet. This latter instrument is connected to a current transformer and assumes a position of equilibrium when energised by a definite current. Current variation either above or below this figure causes displacement of the ammeter's moving element, which closes one of two contactor switch circuits so as to operate the motor either to reduce or increase the current flow through the electrode circuit.

Auxiliary contactor switches interlocked with the motor circuit contactors are also provided, and are closed by the counter E.M.F. generated by the motor as soon as the latter's main circuits are opened after the load readjustment has been completed. In this way the motors are rapidly brought to rest by a dynamic braking effect. Low voltage relays are provided to render the entire apparatus inoperative the moment the voltage between any pair of the furnace conductor bars falls away owing to failure of the power supply; this prevents damage being done by a continuous movement of the electrodes towards a charge in the endeavour to establish a normal current. These protective relay magnets do not, however, respond to a short circuit between any one electrode and the furnace charge, so that, unless such short circuit is accompanied by a current either equal to, or exceeding the normal equilibrium value, the motor will continue to force the electrode in a direction to re-establish normal current. The regulator may be set to maintain fixed current values over a certain range, this being done by cutting out parts of the ammeter windings, and so varying its magnetic force for different energising currents.

Furnace Control Instruments: Ammeters.—Ammeters are used to control the load of furnaces working on constant pressure circuits, and to divide it between the various electrode circuits to secure balance. A separate ammeter is operated by a current transformer placed in each electrode circuit and is calibrated to indicate the main current flowing. These instruments are equally essential for both hand and automatic control, being necessary in the latter case for setting the load-adjusting resistance so that each regulator maintains the desired current. They are always mounted in front of or close to the electrode regulating motor control gear, which may be either a tramway type controller or a push button clapper switch starter. In many furnace installations all the instruments and switchgear are mounted on one board placed in front of the motor controllers, and in some convenient position away from the furnace; in other cases the ammeters are mounted on a small panel attached to the back of the furnace with the controllers fixed in a position whence the instruments are plainly visible.

Current Transformers.—The same current transformers are generally used for operating the regulator electro-magnet together with all control instruments, which may include an ammeter, indicating wattmeter, integrating watt-hour meter and recording wattmeter. These instruments are all placed in series in the current transformer circuit, the load adjusting resistance alone being shunted across the regulator electro-magnet windings.

Current transformers are designed with a fixed current ratio, so that the current flowing through the instrument circuit is always a definite fraction of the current flowing through the electrode circuit. This is true in general practice, since the resistance of the instrument circuit is kept well below a figure which might otherwise alter the current transformer ratio, and so vitiate the instrument readings. Spark gaps are usually provided which short circuit the secondary windings if their circuit is broken anywhere; this is done as a measure of safety to overcome all risk from the high voltage induced across the secondary windings of the current transformer should its circuit be broken. The spark gap simply consists of a small piece of tissue paper placed between two short-circuiting contact studs; should the voltage at any time rise to dangerous proportions the paper is immediately punctured and the secondary windings become short circuited. Puncturing of the tissue paper, which may at times be accidental, is often the cause of ammeters failing to indicate, so that it is always wise to examine the spark gaps in the event of instrument breakdown.

When current transformers are required to operate the electro-magnets of most automatic regulators, the ratio of the primary to secondary currents is so chosen that the secondary current at normal full load is at least three times the current required to maintain the electro-magnetic device in normal equilibrium; under these conditions the regulator will be able to maintain a load equivalent to one-third full load current or less. This is important, as it is often necessary to hold a bath of steel at a constant temperature, which sometimes can only be done at a load equivalent to one-quarter or one-third of normal full load current.

Indicating Wattmeter.—This is a most useful instrument, if intelligently used, since it indicates at a glance the actual amount of power being taken at any moment. The furnace load in K.V.A. can, of course, be calculated when the ammeter readings are all steady and similar and the line voltage known. Provided the line voltage is never changed the furnace load might be controlled by use of the ammeters alone, but even then the load adjustment is only approximate and far more difficult under a fluctuating load than with an indicating wattmeter, which indicates the sum total effect of possibly two or more unbalanced and fluctuating electrode circuits. Wattmeters are usually well damped and their value for power indicating and control purposes is just as great as the other instruments, whose function is more especially for the equal division and balancing of the load. It is advisable that furnace operators should become accustomed to maintain a desired load by use of the indicating wattmeter, and at the same time to balance that load by adjusting the electrodes so that the ammeter readings are approximately similar or within the same range of oscillation, if the load is fluctuating. An indicating wattmeter is usually operated entirely from the secondary circuits of the furnace transformers, the current coils being in the current transformer circuits and the potential coils connected across the line conductors.

Graphic Recording Wattmeter.—This instrument is most useful, as it gives on a paper chart a complete history of the electrical operation of a furnace from the beginning to the end of a heat; the varying average magnitude of the power in kilowatts, the comparative degree of fluctuation, and the number and length of every stoppage are all plainly recorded. This information enables anyone to judge whether the furnace manipulation throughout was satisfactory in point of view of economy of production and maximum output, each of which naturally follows from operation at a highest possible load factor.

The causes contributing to a low load factor will be shown as actual stoppages or low average loads at different periods due either to excessive fluctuation, or simply to poor load control; in each case the chart will indicate the cause, and so enable it to be corrected. A recording wattmeter is not generally mounted near the furnace, as it is only intended for supervision purposes and not as a guide to the furnace operators, the indicating wattmeter being quite sufficient for this latter purpose. The connections are made similarly to the indicating wattmeter on the low tension power transformer circuits.

Integrating Watt-hour Meter.—This instrument is essential as, apart from measuring the units used for payment purposes, it enables the power consumption per ton of steel for melting and for preheating the furnace to be calculated. It therefore provides information of the greatest importance for purposes of checking production costs. Here again, the instrument is best mounted in an office or sub-station where it is unexposed to heat and dust.

The connections are usually made in the same way as for the indicating and recording wattmeters; the readings in this case, although not strictly accurate, are quite near enough for furnace supervision purposes, and may only slightly differ from the readings of a power company's instrument operating off the high tension circuits of the furnace transformers.

Furnace Switchgears.—Furnaces, drawing power from a high tension supply through the medium of static transformers, are connected to and disconnected from the source of supply by an oil switch placed in the high tension circuits. The low tension or secondary terminals of the transformers are severally connected direct to the furnace bus bars, although in certain cases heavy copper knife switches are interposed either for the purpose of isolating the furnace or for altering the low tension cable connections from the transformer.

Oil switches for furnace installations should be of very strong construction, with the oil tanks built of either steel plate or cast steel. The copper contacts are best made of solid copper instead of being built up from copper strips. Switches are always fitted with automatic tripping devices, which cause the switch to open in the event of dangerous overloads.

Hand controlled switches are closed by a handle, which operates through a system of levers and compresses powerful springs while making the necessary contacts in oil. The switch is held in a closed position against the springs by a small catch which prevents movement of the levers; immediately the catch is displaced by hand or other means, the force of the compressed springs is sufficient to open the circuits with great rapidity. The catch is so arranged that it can be released by hand, or automatically by means of a loose iron plunger enclosed in a solenoid and carrying a striking rod. Should the current flowing through any one of the main power circuits exceed a safe value, the solenoid becomes magnetised and attracts the plunger. These tripping coils are connected to a group of current transformers, so that the pull on each plunger is proportional to the current flowing through the main circuit; to render them, however, insensible to momentary current overloads a fuse is connected in parallel with each solenoid coil, and by its low resistance carries practically the full current in the current transformer circuit. In this way the action of the solenoid is prevented until the fuse melts, as the result of either a very heavy instantaneous overload or of a more prolonged overload of less intensity. A time element is thus introduced which prevents the otherwise constant tripping due to momentary overloads. This method of automatic tripping is very simple, but is objectionable owing to the constant renewal of fuses necessary, and to the fact that the tripping gear and fuses are usually in an exposed position near the furnace and liable to misuse.

Automatic tripping is also effected by overload time limit relays, which on heavy instantaneous overloads or sustained overloads of less intensity close an auxiliary circuit through the tripping coils. The main feature of certain types of these relays lies in their being self-setting and easily adjusted to trip over a wide range of instantaneous overloads, which correspond to smaller overloads over certain time intervals. This instrument may be mounted in the sub-station, and so only permits manual operation of the switch from the furnace shop.

The oil in furnace switches should be examined from time to time, as it is likely to become carbonised and sludged if the switch is carelessly used for breaking and making circuit under load.

CHAPTER VI.

POWER CONSUMPTION COST AND CONTRIBUTORY FACTORS.

During the earlier years of the electric steel industry, at a time when the full value and scope of the electric furnace had not been generally recognised, the cost of power for melting and refining was regarded as the predominant factor of economic success or failure. At that time, also, the electrical efficiency of single-phase furnace installations was poor, and as the field of utility was then very limited it is not surprising that all those other economic factors, which are now realised to be of equal importance, were subordinated to the prime consideration of power cost. It was also by no means generally accepted that electric steel was comparable in quality to crucible steel, and when sold in the form of billets or bars it was often placed in the same category as Swedish Bessemer steel. Under these circumstances the power consumption cost amounted to roughly one-quarter or one-third of the selling price, when steam raised power was used.

The same vital importance can no longer be attached to the question of power cost in the case of high-class carbon and alloy steel ingots, which now command a market value considerably more than four times the power consumption cost, notwithstanding the increased cost per unit of steam-generated electricity. Power consumption may now be considered of only equal importance with other items of manufacturing costs, but should receive all possible consideration from both administrative and technical standpoints with a view to reaching a maximum economy.

Power Contracts.—Steam generated electric power is generally purchased under either the "meter rate," "flat rate," or "maximum demand rate" systems. In each case the power company's charges are based upon the "Cost of Service" theory of rate-making, which is so regulated as to include a fair return upon invested capital.

Whenever the supply of power in bulk is contracted for, plant capable of supplying that amount must always be available to meet the demand, so that, whether the consumer is taking power or not, the power company is shouldered with a fixed burden of depreciation, administrative expenses, and interest on capital. On this account, under the different systems of rate-making, the consumer is required to make certain payments which serve as a guarantee to the power company against all such stand-by losses. Power contracts, for this reason, are usually based upon a combination of either flat and meter rates, or of maximum demand and meter rates.

Meter Rate System.—In most contracts for the supply of electric light service, the number of units consumed are metered and charged for at a fixed rate per unit. In such cases the power supply company is well acquainted with the maximum bulk and nature of the load, or, in other words, the approximate number of units used per annum and the load factor. Under these circumstances the meter rate payment can be regulated to cover all estimated stand-by losses.

There are various modifications of this system which either impose an extra charge or allow a rebate, according to whether the load is taken at times when the central station generating plant is fully loaded or partly idle. In the case of lighting services the maximum call for power occurs during the hours of darkness, during part of which the central station load is at its "peak".

Flat Rate System.—According to this system no meter is used, the charge for electric service being based upon the power consuming capacity of the consumer's installation, or upon an agreed fixed sum for each consumer. Such charge would naturally be based upon the possibility of overloads, and upon the average load factor likely to be attained on the consumer's installation, which is a measure of the total estimated number of units consumed per annum.

The generating cost of steam-raised electric power is considerable, and requires the insertion of a special Coal Clause in every power contract. In the case of hydro-electric power, generating cost is practically negligible, so that, if the power charge is based upon a consumer's fixed plant capacity, it is immaterial whether the power is delivered intermittently or under a high load factor.

The flat rate system alone cannot always be satisfactorily applied in the case of steam-raised power, and is therefore usually used in conjunction with the meter rate. The latter covers the central station generating costs and profit, whereas the former is intended to cover the fixed charges on the plant, which must be at all times available for instant use.

Furthermore, when basing the flat rate charge upon the rated plant capacity, it is also customary to make a further charge, which is in the nature of a penalty, according to the maximum number of kilowatts or kilovolt-amperes by which the rated capacity has at any time been exceeded during a week, month, or other period of time; this excess demand is based upon the maximum average K.W. or K.V.A. measured over consecutive quarter or half-hour intervals. The consumer in this case is penalised for a demand exceeding the stipulated amount, and does not receive any rebate should the maximum demand similarly measured be less than the fixed amount chargeable on the rated plant capacity; in this latter respect the payment differs from that made according to a true maximum demand rate.

Maximum Demand Rate.—This rate is very similar to the flat rate, and is charged to the consumer in order to cover the fixed charges on the central station plant kept available for supplying the consumer with power.

The flat rate charge is based on the estimated demand in K.W. or K.V.A., which obviously cannot be favourable to the consumer, who, in some cases, may be further penalised for any demand exceeding the agreed figure. The maximum demand rate is based upon actually measured maximum demands in K.W. or K.V.A., so that the charge payable by the consumer is more strictly proportional to the maximum amount of power demanded from and supplied by the Power Co. during a given period of time. The maximum demand is usually recorded by a special watt-hour meter, first introduced by Merz. The instrument records the maximum of the average number of kilowatts demanded during any one of consecutive quarter or half-hour periods, which are determined by a clock actuating the recording mechanism at regular intervals. The fixed charge is then made upon this maximum demand at an agreed rate, and is usually chargeable on each monthly or three-monthly demand, according to the terms of the contract.

The maximum demand rate of payment is seldom used together with a meter rate, as it is generally alone intended to cover the actual generating cost of power. This combined meter and maximum demand rate of charge (sometimes known as the Hopkinson two-charge rate) is now almost universally adopted both in England and in America for electric furnace installations, and provides a most equitable basis for power contracts. When the supply of power is contracted for by Power Companies with small generating stations, there is usually a supplementary clause by which the consumer agrees to pay a fixed annual payment, and to indemnify the Power Co. in the event of the premature termination of the agreement. Such fixed annual payments are usually based on a demand figure rather less than the rated capacity of the installations, so that, even in this case, the consumer will benefit by paying the kilowatt or kilovolt-ampere charge on the maximum demand rather than on the true flat rate basis.

Load Factor.—Whenever the flat rate or maximum demand rate systems enter into a power contract, it is evident that the consumer shoulders the burden of stand-by losses occasioned by an intermittent demand on the generating plant. Regarding such plant as capable of producing a profit to the consumer while in operation, it is obviously to his advantage to make full use of this profitable asset, and so reduce the proportion of fixed charges on the plant relative to the running costs.

A power bill may be considered as consisting of two items, namely, a fixed charge and a running charge, which are met by the maximum demand or flat rate payment and the meter rate payment respectively. The actual cost to the consumer per B.O.T. unit will vary according to the total units consumed during a certain period, or rather to the load factor, as indicated by the curve shown in Fig. 77. This curve is plotted from the calculated cost per unit taken over one month at different load factors. In this case it has been supposed that the maximum demand is constant at 600 kw. every month, and charged at the rate of 7s. per month for every kilowatt demanded, while a meter rate of ·3d. per B.O.T. unit is charged on the total units consumed during the year.

The load factor of very many electric furnaces is below 40 per cent., taken over a whole year, and it will be seen that it is below this point that the curve begins to rise rapidly. This clearly shows the great saving of power cost to be gained by even a small improvement in load factor. The curve shows a reduction in the cost per unit of 15·5 per cent., resulting from an increase of the load factor from 30 per cent. to 40 per cent.

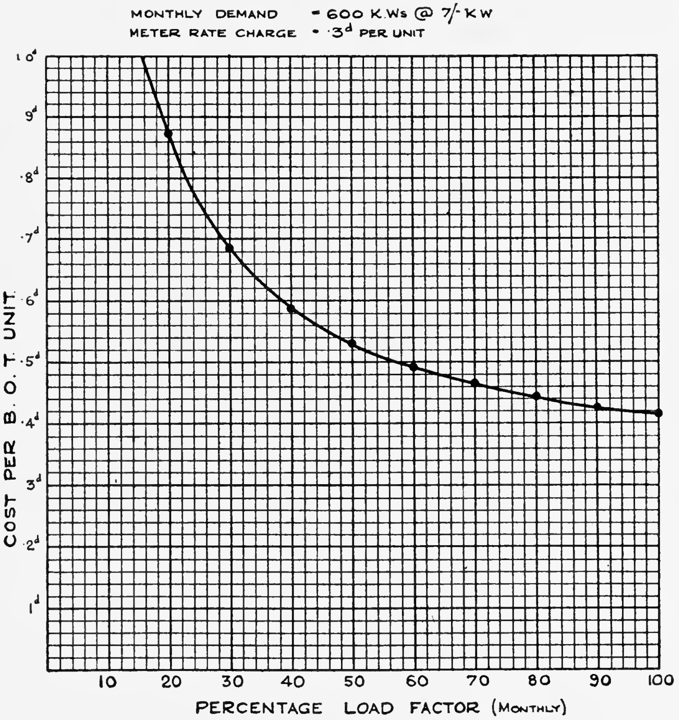


Fig. 77.

The actual load factor of a highly efficient steel furnace installation, working day and night and the usual five and a half day week, was found to be 46 per cent. This figure covered a period of nine consecutive months' operation, which included the total time covered by all repairs and usual holidays, so that under such conditions 40 per cent. may be taken as a fair average figure for the purpose of estimating the actual over-all cost of power per unit according to different contracts.

So far the effect of load factor has only been considered in its relation to the actual cost of power, whereas its influence upon output and reduction of manufacturing charges is of still greater importance.

The total manufacturing cost per ton of steel may be broadly divided between a fixed charge on plant and administration and the actual cost of manufacture, which includes all materials, labour, power, and sundry charges. Of these two distinct charges the fixed charge is distributable over the actual output in a given time, and obviously becomes less per ton as the output increases. The running cost is likewise reduced owing to reduced power cost, and lower labour and repair charges. With regard to the two latter charges it does not follow that an increased output entails a shorter life of furnace lining, in fact, experience often proves the opposite; also, an increased output can usually be obtained without increasing the number of furnace hands. The nett result, therefore, of improving the load factor is to produce an increased output at lower cost, and so doubly increase the net profit per ton of steel made.

Maximum Demand.—As has been previously stated, the maximum demand is the figure upon which the fixed rate power charge is made in order to cover all standing charges on the power plant. The maximum demand figure should be fixed and regarded by the consumer as a power demand which it is not intended to exceed, and may be taken to represent the maximum permissible power capacity of the installation. This being the case it may be considered almost as a flat rate charge based upon a definite estimated demand.

Since load factor, averaged over a period of a year or even over the period of a single heat, is such an all-important element in the economy of manufacture, it is of great importance to take full advantage of the permissible power demand at all times, consistent only with the varying metallurgical and electrical conditions required during the furnace operation. The effect of load factor during a single heat is also very considerably greater than is at first apparent, and is explained at greater length in connection with furnace radiation loss and power input. The maximum demand, although actually recorded as K.W. by the integrating watt-hour meter attachment previously referred to, is usually converted to K.V.A. for purposes of a maximum demand rating based on K.V.A. instead of K.W. For this conversion it is necessary to know the average power factor over the entire period for which the maximum demand is recorded; this figure is ascertained from the combined readings of a kilowatt-hour meter and of an integrating meter which measures the wattless component of the load. The maximum demand recorded in K.W. can then be converted into K.V.A. by using the average power factor so determined as a multiplier.

It should also be recognised that, if the maximum demand should for any reason exceed the normal figure during one-quarter or one-half hour interval, the cost of power per unit during the whole of the period charged for will be increased, which merely corresponds to the effect of a reduced load factor.

Furnace Radiation Loss and Power Input.—The power absorbed in an electric furnace is not all usefully used in supplying heat energy to a furnace charge, a considerable portion being constantly dissipated as radiated heat according to the difference in temperature between the inner and outer surfaces of the furnace lining, its thermal conductivity, and other factors. The consequent effect of this ratio of useful to useless energy was not properly recognised in the earliest electric steel furnace installations in which the proportion was about equal, and for this reason the power consumption was inordinately high, and the lining repairs correspondingly heavy.

The radiation loss may be conveniently recorded as the power in K.W. that is in practice found to be required to maintain a constant temperature of a bath of steel over a sustained period. This figure can be easily determined for every furnace, and will, of course, vary with the condition of the lining and other obvious factors.

In most furnaces of 3 tons capacity the radiation loss, as expressed in the above terms, will be approximately equivalent to 190 to 220 kw.; this being so, there is not such a very considerable margin of power available for melting purposes, even where a normal full load of 600 kw. is provided. During the earlier melting down period the radiation loss will not, of course, be so great, owing to the lower temperature of the furnace interior, but may be estimated at 160 kw. for the purpose of examining the effect of load factor from this particular standpoint during the melting-down period of a heat. The curve shown in Fig. 78 illustrates how the percentage of useful energy in a given power input varies for different values of load factor, assuming an average radiation loss of 160 kw. and a maximum possible load input of 600 kw. The excessive waste of energy

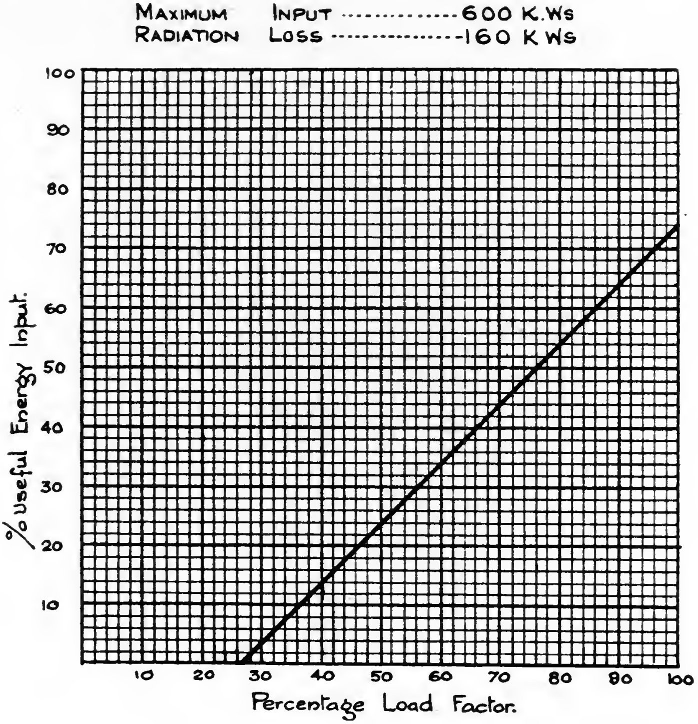


FIG. 78.

is in this way clearly indicated, and emphasises the necessity of operating a furnace at the highest average load permissible.

Another way of illustrating the general influence of load factor is by plotting a curve that shows the variation in power consumption per ton of steel at different load factors. To draw such a curve an actual case must be taken from which the theoretical units for melting and refining a ton of steel may approximately be deduced. The curve A shown in Fig. 79 has been plotted from figures calculated from a specific case, in which 3·2 tons of melted and refined basic steel were produced in six hours using 2453 units. Allowing fifteen minutes for sundry delays, the average power input was 426 kw. The radiation loss was, on the other hand, continuous and constantly increasing during the full period of six hours, and has been estimated for the purpose of this example as averaging 170 kw. throughout. On this assumption the units used usefully for melting and refining purposes amount to 450 per ton of steel, which agrees fairly well with the generally accepted figure. The maximum permissible load was 580 kw., which gives in this case an average load factor of $\frac{426 \times 100}{580} = 73{\cdot}5$ per cent. during the five and three-quarter hours of actual operation.

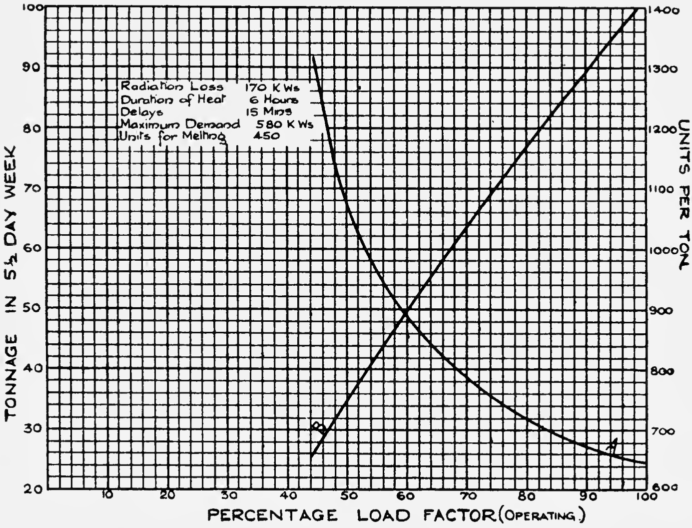


Fig. 79.

It must be borne in mind that as the load factor decreases and so lengthens the period of each heat the average radiation loss will be somewhat greater than 170 kw., and so give power consumption figures slightly higher than those indicated by the curve. Besides this, there are other minor factors which should really be considered, but their introduction would only slightly modify and tend to obscure the result by the many complications involved. The curve B shows still more directly the influence of the operating load factor on output, allowing thirty-six minutes for charging and fifteen minutes delay during each heat, which is, of course, good practice but perfectly possible for this size of furnace.

Such curves can be plotted for any furnace, provided the average radiation loss is based upon an observed figure at a medium bath temperature, and the maximum permissible load fixed at a given figure for the purpose of determining the load factors corresponding to different average loads.

Load Fluctuation.—The effect of load fluctuation on power costs resolves itself purely into a question of load factor and output, as apart from the objectionable results of a badly fluctuating load upon the electrical equipment, there is always the difficulty of maintaining a desired average load. Load adjustment is almost universally effected in accordance with the indications of ammeters, which, owing to the rapid oscillations of their pointers, fail to indicate the average load on the furnace; this inevitably leads the furnace operators to underload the furnace, so as to avoid the heavy apparent overloads inaccurately indicated by undamped instruments.

Everything, therefore, should be done in the way of furnace manipulation to secure a steady load, which can then be more easily maintained at the highest desirable figure. The character of cold scrap used may have a considerable influence on the steadiness of the load and operating load factor, so that those physical qualities conducive to the maintenance of good loads during the melting period should on no account be sacrificed for the sake of the apparent economy to be gained by the purchase of cheap unsuitable scrap. Equal attention should be paid to the manner of charging in order to secure the best possible electrical conductivity within the charge; observations on this point have been previously made.

The introduction of considerable reactance into the load circuit is undoubtedly the most effective way of damping out fluctuations, and so improving the load factor. Unfortunately, this can only be done at the expense of power factor, which, if any really marked benefit is to be obtained from the use of reactance coils, falls below the usually guaranteed average figure of ·8 to ·85 at normal full load.

Power Consumption.—It will be apparent, after consideration of the above-mentioned factors, that the units consumed in melting and refining cold charges are subject to considerable variation, which in a large measure accounts for the very diverse figures obtained with electric furnaces of either similar or different types. It has been found by careful calorimetric determinations that the heat contained in molten steel of average casting temperature is equivalent to about 370 kw.-hours per ton. After making due allowance for the normal extent of chemical refining on a basic hearth, fusion of fluxes, and other minor power-absorbing functions, it is not unlikely that the above figure rises to 400 or 450 kw.-hours per ton of finished steel melted and refined. This figure, then, represents the useful energy input required per ton of steel produced, irrespective of the type of furnace, operating load factor, and other conditions. The curve A in Fig. 79 illustrates how this theoretical figure can easily rise to 1100 units under adverse conditions of operation which lead to a poor load factor, even though the radiation loss is not beyond a reasonable figure. For these reasons, little value can be attached to any power consumption figures other than those that indicate the best performance possible under conditions that should be clearly stated.

A good average power consumption resulting from continuous operation may be taken as 750 kw.-hours per ton of steel produced, but it must be understood that this figure is subject to considerable reduction or increase according to the degree of manipulative skill, the process employed, and the ratio of useful energy input to radiation loss.

When using the acid process the power consumption will usually be somewhat lower than for the basic, other conditions being similar.

CHAPTER VII.

ELECTRO-METALLURGICAL METHODS OF MELTING AND REFINING COLD CHARGES.

Introduction.—The electric furnace is used in steel works to perform a variety of functions. Generally speaking, the smaller units serve to perform all the metallurgical operations for the production of finished steel from a crude charge of miscellaneous steel or iron scrap. Furnaces of the largest capacity are usually employed in conjunction with other steel-making plant, and are then used to perform one or sometimes two distinct operations of a process conducted in several stages. In either case the furnaces, which may be basic or acid lined, are suitable for carrying out operations which are common to the open hearth furnace, but in a modified manner. Owing to the nature of electric heating, which enables chemical reactions to proceed in an atmosphere uncontaminated with oxidising gases and under slags of special character, operations may also be performed which have no counterpart in a gas or any other type of furnace.

The several processes used in the manufacture of steel, made either wholly or in part in electric furnaces, must be broadly classified according to the acid or basic character of the slag employed. Each process must also be studied separately in its application to the treatment of cold scrap and liquid steel.

The basic process has been most generally used for the production of finished steel from cold charges, while the acid process affords special advantages for foundry practice and for finishing semi-refined liquid steel. The basic process, as applied to the open hearth or Bessemer converter, is far more limited in its application than when practised in the electric furnace. Since electric heating can be applied to a furnace charge in such a manner as to exclude all oxidising gases, it is not only possible to maintain a reducing atmosphere, but also a powerfully reducing slag, exceedingly basic or limey in character, which has power both to deoxidise and desulphurise a bath of semi-finished steel. This property of deoxidation is entirely absent in open hearth and converter slags, which contain at least 16 per cent. of FeO, and whose range of chemical action is limited to one of oxidation alone. It is true that a basic open hearth slag, to which has been added a quantity of calcium chloride according to the Saniter process, has a certain power of desulphurising, but the degree of sulphur removal does not equal the extraordinarily low figures obtainable with electric furnace basic slags.

With regard to the acid process, there is a far closer identity of behaviour of the acid slags used in converters and in electric and open hearth furnaces, as in each case the slag will only remove carbon, silicon, and manganese, without any reduction of sulphur and phosphorus. In certain instances, the acid slag in an electric furnace may be partially reduced by carbon, when it then helps to accelerate the deoxidation of the bath; this applies more especially to liquid refining. In this latter respect it differs from open hearth and converter slags, which are not subjected to the same intense local heat.

The general principles governing the selection of one or other process for use in the electric furnace are different from those which might be applied in the case of the open hearth or converter. In the latter cases, apart from the consideration of available pig-iron, the quality of steel required usually defines the choice of process. Acid steel made from high class raw materials is admittedly superior in quality to basic steel of similar composition, but this does not apply to electric furnace steel and, if anything, is the reverse. It is now generally accepted that basic electric steel, if properly made, meets all the requirements of crucible steel, whether it be plain carbon, simple alloy or high speed, but this is only possible if the phosphorus and sulphur are exceedingly low and at a figure which acid electric steel, made even from a good class of steel scrap, could not approach.

The acid electric process, however, has been used largely for finishing semi-refined steel. In this respect, the furnace merely serves as a convenient internally heated receptacle, in which oxidised steel may be brought up to a casting temperature and deoxidised under slightly reducing or neutral conditions. Necessary additions of alloys may be also conveniently made prior to casting. Of recent years considerable attention has been directed to the beneficial effects derived by holding steel in a perfectly tranquil state for a short period before teeming. This procedure enables minute slag particles in suspension, and possibly gases, to rise through the steel and escape. The conditions required to promote this simple physical action are perfectly fulfilled by the electric furnace, and it is probably some such secondary effect, proceeding simultaneously with the process of deoxidation, that helps to impart the high qualities to carefully refined acid steel. This, of course, equally applies to basic electric steel.

Owing to the extravagant cost of electric energy as a source of heat, metallurgical operations, which may be satisfactorily and economically performed in either the open hearth furnace or converter, are at once precluded from electric furnaces. Hence the electric furnace is used only where the others fail, and is therefore usually limited to the further refining and finishing of comparatively pure raw materials. The charge may equally well be cold or liquid, but in either case will not require more than a small degree of purification. Although the actual reduction of impurities may be very slight, it is just that final degree of refining which justifies the use of the electric process. As a general rule, therefore, it may be assumed that the charge will consist of liquid steel or steel scrap, and average only a small percentage of carbon, phosphorus and sulphur. Excess of the two former elements may prolong the period of chemical action considerably beyond that of melting, in which case the economic advantage of the electric process will be seriously impaired.

Basic Process.

This process, as usually conducted, may be briefly divided into three distinct stages:—

(i) Melting down under oxidising conditions.

(ii) Skimming and carburising.

(iii) Refining under powerfully reducing conditions and finishing with alloy additions, etc.

General Outline.—The process, as conducted in the various types of basic lined furnaces, either with or without conductive hearths or bottom electrodes, is the same in general principle. The details of operation, more especially charging, may be slightly different, but the method of working will not deviate far from the following description, which applies more strictly to direct arc furnaces without conductive hearths. For the operation of indirect arc furnaces, slight modifications in methods of charging will suggest themselves, and are here purposely omitted so as not to confuse or destroy the continuity of the following description of the process. The various steps in the mechanical operation, together with the chemical changes that occur, may be briefly summarised in consecutive order:—

1. Hand, or mechanical charging of fluxes and scrap into the furnace, which has been previously heated.

2. Load put on to the furnace and melting begins.

3. Scrap melts under the electrodes, which bore downwards until pools of metal are formed on the bottom with a slag covering.

4. Melting and further slag formation proceeds until all fluxing materials in the original charge are fused.

5. A considerable portion of the scrap melted, forming a bath; appearance of slag observed, and additions of ore or lime made if necessary. Feed of scrap given, filling furnace as far as possible. Removal of carbon, manganese, silicon, and phosphorus proceeds, provided the slag is sufficiently oxidising and basic.

6. Second feed given. Chemical reactions still proceed.

7. All the charge melted. Bath hot and showing very slight boil. Slag appearance correct.

8. Load off, skimming started.

9. Bath skimmed. Carbon additions made to naked bath if necessary, followed sometimes by a small addition of ferro-silicon, and then a mixture of fluor spar and lime. Bath becomes partly deoxidised by the ferro-silicon addition.

10. Load on. Fluxes begin to fuse and correct slag formation is promoted by an addition of carbon dust. Bath fairly hot.

11. Slag fused; bath well rabbled. Slag pale in colour. Bath sample taken for analysis, if required. Slag thickened by

further lime additions. Carbon dust added in small quantities, as required, to complete the removal of metallic oxides from the slag, and to promote the formation of calcium carbide.

12. Sample taken and a small addition of ferro-silicon made, if necessary. Further deoxidation of the bath proceeds, and if the slag is nearly or quite white, desulphurisation begins.

13. Heat tests and bath samples taken from time to time until the steel lies quiet in a sample mould. Deoxidation and desulphurising action finished.

14. Bath at casting temperature; alloy additions made if required. Bath rabbled after five minutes, and final sample taken.

15. Load off. Steel poured.

The above summary must not be regarded as a strictly accurate history of a basic heat, it being obvious that the chemical changes are progressive and gradual, and overlap the manipulative operations. It is only intended to assist in piecing together the detailed description of the different phases of the process, and to serve as a guide for examining the relationship between the various operations and the chemical reactions which they produce.

Choice of Scrap.—The selection of scrap has an important bearing on the economic operation of any electric steel furnace. In many cases it may be impossible to use a type of scrap that is suitable in every way, but careful judgment in the selection and mixing of inferior grades may lead to equally satisfactory results, not only as regards the quality, but also the quantity of steel made. It should be remembered that chemical analysis is not the only standpoint from which the value of scrap can be judged, since the shape and size have considerable influence on the electrical conditions during melting. A suitable class of scrap should be such as will give no trouble with the metallurgical operations, or in the maintenance of a steady electrical load. The employment of light scrap of irregular shape or very light turnings is sure to cause considerable difficulties in maintaining a full and steady load, unless judiciously mixed with some class of heavy scrap, which may help to eliminate the disadvantages otherwise experienced. An unsteady and low load, due to the irregular resistance of such scrap, results in a poor load factor, which in itself is a fundamental cause of poor output, thermal inefficiency, and increased cost of production. It must be remembered that electrical energy equivalent to the radiation loss of the furnace constitutes a large percentage of the maximum power input of the furnace (more especially in the smaller sizes), and, therefore, inability to maintain the power available for useful heating at its maximum results in considerable loss of time and failure to operate the plant at its full capacity.

The nature of the scrap from a chemical standpoint is also important. The phosphorus and sulphur contents should be reasonably low, so as not to unduly prolong the period of the refining action of the basic oxidising and reducing slags. Generally speaking, almost any class of steel scrap is sufficiently low in these elements and will be satisfactory, provided the percentage of carbon is also low. Carbon, either chemically combined or mechanically mixed in the form of coke, cinders, or oil, affects the suitability of ordinary steel scrap more than any of its other chemical constituents. Coke and cinders are very commonly found in turnings which have been dumped on to freshly made ground, but of all forms of carbonaceous foreign matter, the worst is oil. It is generally inadvisable to melt oily turnings as the sole constituent of a charge; the action of heat decomposes the oil, which leaves a fine deposit of carbon on the turnings, the greater part of which is absorbed and passes into the bath of steel. For this reason, the bath, when otherwise ready to skim, will contain too much carbon, which must then be boiled out to ensure complete removal of phosphorus. When a charge consists of low carbon scrap, the process of decarburising and dephosphorising should proceed as fast as the melting operation, so that when all the charge is melted and hot enough for skimming, the carbon, phosphorus, manganese, and silicon contents will be as low as required. This is not always possible when the scrap is over-charged with carbon, owing to the difficulty of maintaining a strongly oxidising basic slag during melting. The metallic oxides are rapidly reduced and the conductivity of the slag is thereby lowered; in order, then, to maintain a full load current the electrodes may be forced to dip into the slag, and this under certain conditions gives rise to heavy current fluctuations. The electrodes, again, should never under any circumstance dip into an oxidising slag, as any addition of ore is rapidly reduced by the action of the carbon electrodes rather than by the action of the carbon in the bath. These facts demonstrate the excessive waste of time involved in boiling down carbon and ensuring the removal of phosphorus, which can only be accomplished in extreme cases by frequent skimmings and constant additions of ore. The use of light, stringy turnings should also be avoided, as the loss of time and heat occasioned by the frequent charging of such light, bulky scrap usually represents considerably more money than the extra cost of a more suitable scrap. It may also be pointed out that, when charging stringy turnings into a small furnace using graphite electrodes, there is considerable risk of breaking the latter; also, the melting loss by oxidation is excessive.

The following observations will serve as a guide in the selection of suitable scrap or mixtures of scrap for melting:—

I. Scrap should be of such size and form as to pack closely in the furnace and render charging operations capable of being conducted with minimum expenditure of time and labour, and without risk of damaging the door jambs or electrodes.

II. Scrap should be sufficiently heavy (*a*) to enable the entire charge to be introduced into the furnace without more than two feeds after the initial charging; (*b*) to form pools of metal under each electrode of sufficient size (i) to prevent entire rupture of the electric circuit, which might otherwise occur—more particularly with small graphite electrodes—should the electrodes bore down to the bottom and then lose contact with the unmelted charge, (ii) to prevent the electrodes arcing on to the bottom in the case of conductive hearth furnaces.

III. It should be moderately free from rust to prevent over-oxidation of the bath; in reasonable amounts rust is useful, as it will assist the removal of carbon and maintain a slag charged with oxide during melting operations without the addition of iron ore.

IV. The use of high carbon scrap alone is generally avoided, as, when melted, the resulting bath will probably contain carbon, and require further boiling down with ore additions; at the same time there is danger of imperfect removal of phosphorus. Dead soft steel or wrought-iron scrap is also objectionable if melted alone. The bath is invariably over-oxidised and cold melted, and the erosion of the dolomite banks and wear on the roof and walls are considerably increased. The best average carbon content of a charge of scrap is from ·3 per cent. to ·4 per cent.

V. The presence of carbon in scrap as foreign matter, such as unconsumed cinder and especially oil, has already been dealt with.

VI. Wet scrap is objectionable as it is liable to cause explosions when fed into a bath with a slag covering; this, besides being a source of danger to the furnacemen, impedes the rate of charging, and consequently increases heat losses and reduces output. Wet turnings fresh from the machine shop should be allowed to drain for a week or more before being melted.

VII. A high manganese content is sometimes imperfectly removed—particularly if the carbon is also high.

It must not be supposed that scrap, which does not fulfil all the conditions enumerated above, is unsatisfactory; as previously stated, a judicious selection and mixing of grades, unsuitable by themselves, is quite capable of producing the very best results both as regards quality and output.

Method of Charging Scrap.—Scrap should invariably be charged in a manner that will offer the least resistance to the passage of current through the charge. Rail and bar ends and other such shapes should be charged side by side as closely as possible, and not in criss-cross fashion. A little extra time and trouble spent in careful charging will often save considerable loss of time caused by a fluctuating load. Heavy solid scrap, such as ingot heads, when mixed with very light scrap, should be charged under the electrodes on the bottom; this will prevent any possibility of the electrodes boring down through the light scrap to the bottom and losing contact with the charge.

The initial charge should be of such quantity that one or, at the most, two subsequent feeds will suffice to complete the charging operation. Constant opening of the charging doors for this purpose is a practice to be avoided, owing to the considerable loss of heat from the interior of the furnace, more especially when the bulk of the charge is melted.

Care is always taken to charge scrap in a manner that will afford maximum protection to the furnace banks ; in cases where turnings constitute a large portion of the charge, it is an easy matter to effect this by reserving the turnings for covering the banks from the slag line upwards. In the immediate vicinity of the arc the turnings will rapidly melt, and during subsequent feeds fresh quantities should be charged, as far as possible, between the electrodes and the nearest portion of the banks. Unmelted scrap remaining on the banks is left undisturbed until the charge is almost ready for skimming, when it may be gently pushed into the bath with the aid of a hooked bar.

Irregular shaped heavy scrap, if used in small quantities, is preferably kept out of the initial charge and added to the bath during subsequent feeds to avoid disturbance of the furnace load.

Scrap should not be fed into a furnace until the initial charge has been melted to form a fairly hot bath of steel between the electrodes. The practice of pushing in fresh scrap as soon as there is any available space leads to all the troubles associated with cold melting. The slag, being thus constantly chilled, does not promote the rapid elimination of carbon and the metalloids from the cold bath of metal, as evidenced by a dead-looking appearance and absence of boil. The bath of metal is also likely to chill on the bottom, and will then require a small addition of pig for its removal, which could otherwise only be effected by prolonged heating. There is also the possibility of the bath becoming unduly charged with dissolved oxides, and this adds to the difficulty of the subsequent carburising (if any), deoxidising, and desulphurising treatments. Provided the composition of the slag is correct, a slight boil will indicate that the temperature of the bath formed from the initial charge is suitable for fresh additions of scrap to be made.

If a furnace is properly charged and left undisturbed while the load is maintained, the portion of the charge lying between the electrodes will frequently, and especially in the case of turnings, frit or weld together and ultimately collapse into the bath beneath it, causing a violent boil and heavy current overload. This usually occurs rather suddenly, and, unless the melter is on the alert, may cause the automatic trip to operate and cut off the supply of power. A skilful melter will usually anticipate such an occurrence by gently pushing that portion of the charge into the bath just before it is likely to collapse. This point marks a favourable time for feeding, which may be done a few minutes after the slight resulting boil has subsided. A furnace charged and fed in this manner will not have acquired a high roof temperature by the time the bath is ready for skimming. Some of these remarks may not apply to, or may be modified to suit various types of direct arc furnace, but the general principles are the same for all.

In the case of indirect arc furnaces, such as the Stassano, care has to be taken not to break the long high voltage arcs formed between the electrode tips by throwing cold scrap directly across their path. If only one of the three arcs formed in a Stassano furnace is broken, the other two remain nearly unaffected and the load is not entirely ruptured; but the reduction of power and the resultant out-of-balance load should be avoided.

Basic Oxidising Slag.—*Function.*—In this case the basic slag will contain a large percentage of iron oxide for the purpose of eliminating carbon, manganese, silicon, and phosphorus from the bath of metal by oxidation. The presence of a high percentage of lime in the slag is also essential for the satisfactory removal of phosphorus, which, after oxidation to phosphoric oxide by the oxides present in the slag and dissolved in the metal, is fixed by the lime to form a stable phosphate. The duty then of the oxidising slag, which is somewhat similar in character to basic open hearth slag, is to reduce the quantity of all impurities in the scrap other than S to a required degree, and by so doing, to produce a bath of steel as low as possible in C, Mn, Si, and P, without excessive oxidation of the steel.

Fluxes.—The fluxes used for the formation of such a slag are in most cases iron ore and lime. Limestone is sometimes used, but, although initially cheaper, proves less economical after the carbon dioxide has been driven off at the expense of electric energy. It is always advisable to use the best raw materials, which in their purest forms require the least amount of heat energy for fusion, and owing to their degree of concentration have the maximum power of chemical action.

Lump ore is preferable, as in this form its action is more rapid when thrown on to a bath of steel for boiling down purposes.

Character and Appearance.—A satisfactory slag will usually contain about 25-30 per cent. combined FeO and Fe_2O_3, and 35-40 per cent. CaO, the remainder being MgO, SiO_2, MnO, etc. A slag of such composition may be recognised by its appearance when molten, as also by the colour and fracture of a sample removed from the furnace. The molten slag should never appear glassy and reflect the light rays of the arc, but should have a dull matt appearance, indicating a sufficiency of lime. A more positive examination of the slag can be made by observing the colour and fracture of a cooled sample. Such a sample may be conveniently taken by dipping the end of an iron bar into the slag several times and momentarily withdrawing it, so that successive layers may chill and accumulate to form a thick covering to the bar. On removal of the bar the slag will rapidly set and cool, exhibiting a smooth shiny grey skin, and cracking off readily. The fracture should be close and stony in appearance and have a grey-black colour. Insufficient oxide is always evidenced by a brownish shade, which is not always easy to detect in artificial light. Infusibility, or a frothy pastiness, is also a positive sign of lack of oxide in the slag, and is generally accompanied by a high percentage of carbon in the bath and a copious production of pale coloured smoke and luminous flame. A sample of such slag will be brown or yellow, and generally full of small cavities. An over-oxidised slag is very fluid, black in colour and has a distinctly crystalline fracture. Excess of silica will give a glassy appearance to the slag, either when molten or chilled, although the colour may be black. Experience soon enables the different characteristics to be recognised and slag formation to be regularly controlled in each heat from the time that the slag can first be observed. Inexperienced melters may not observe the unsuitable character of a slag until the bath is almost ready for skimming, and have then to consider whether it is better to risk a high phosphorus content in the finished steel, or to adopt the safer measure of correcting the slag composition at the expense of time and labour.

Formation and Control.—The fluxes, consisting of burnt-lime and iron ore, are usually charged on to the furnace bottom before the scrap, but this practice is sometimes modified so that only a portion is at first charged, the remainder being added while the charge is melting down. The proportion of lime and ore will depend upon the chemical composition of the scrap, besides its physical condition and freedom from rust, dirt, and carbonaceous matter. Scrap containing much carbon, either chemically combined or mechanically mixed in the form of oil or cinders, will require an increased addition of ore before or after the melting. The quantity of ore required also depends upon the extent to which the scrap is oxidised in the furnace either before or during melting operations. The quantity of ore and lime used in a charge should be so judged as to form a sufficient slag covering to the bath; if the slag blanket is too thin the arc will tend to strike on to the metal and cause a fluctuating load, and if too thick, the electrodes will touch the slag without forming a free arc.

Between two and a half and three and a half hundredweights of lime and ore combined will usually be sufficient to form a good slag covering for a three-ton furnace. The quantities for smaller and larger furnaces may be based on the ratio of the respective bath areas. If, however, the scrap itself contains slag-forming foreign matter, the weight of ore and lime must be accordingly reduced. The correct quantity of ore is best determined by trial for a given class of scrap and method of working, and it is better, when in doubt, to begin a heat with an insufficiency of ore and to make additions from time to time until the character of the slag is correct, as judged by its appearance. The total weight of ore used will serve as a guide for more accurately proportioning the ore and lime constituents of subsequent charges. The proportion of ore and lime should be such as to maintain a good black basic slag from the time of its first formation; in this way the elimination of C, P, Mn, and Si will proceed simultaneously with the melting, and be carried sufficiently far when the bath is hot enough for skimming. Such conditions are easily maintained if the class of scrap and other conditions of working are not frequently altered, but uncontrollable circumstances, such as the fracture of an electrode, are always liable to vitiate results in the case of individual heats.

Satisfactory removal of P, Mn, and Si can only be effected provided the slag conditions have been correct for a certain length of time, so that it is not sufficient merely to finish with a slag of proper composition before its removal from the furnace. This question should always engage the careful attention of the operator, whose skill and experience will enable him to judge the amount of ore or lime to add for correct regulation of slag composition during the process of melting, and to estimate the extent of P elimination at a time when the bath of steel is ready for skimming.

Low bath and slag temperatures sometimes produce a misleading appearance of the slag, which may be thin and dead-looking with apparent indication of over-oxidation. An increase in the temperature will usually suffice to improve its character, and to produce a slight boil, provided there is still sufficient carbon in the bath.

An excessive quantity of slag, due either to foreign matter in the scrap or to the addition of large quantities of ore for boiling out carbon, should be avoided by pouring off the excess at intervals. A thick blanket of slag increases the resistance to the passage of current between the electrodes and the metal bath, and may at times prove so considerable that the electrodes either touch the slag or dip into it. When this happens, the reduction of the metallic oxides by the carbon electrodes lowers the conductivity of the slag, and further intensifies the difficulty of carbon and phosphorus removal. In extreme cases, the phenomenon of "pinch effect" may also develop and give rise to a strongly fluctuating load.

Slag Reactions: I. *Removal of Carbon.*—Carbon in a bath of steel is oxidised both by the action of iron or other metallic oxides dissolved in it, and by the influence of a covering slag rich in either FeO or Fe_2O_3, or these two combined as Fe_3O_4; in either case the speed of reaction is dependent upon temperature. A cold bath of steel will not boil even under a properly composed slag unless the temperature is high enough to promote the reaction, and it is, therefore, necessary to melt a charge of scrap in such a manner that the increasing bath of steel is always hot enough (except during feeding) to allow the elimination of carbon to proceed. The best method of charging and feeding cold scrap to ensure this has already been given.

The iron oxide constituent of the slag is mainly present as FeO, and results from the partial reduction of the higher oxides derived from either rust in the scrap or added in the form of iron ore or mill scale. This partial reduction of the higher oxides accompanies the oxidation of carbon in the bath of steel by the available oxygen, the reaction being in this case exothermic. The oxidation of carbon by FeO is endothermic, so that heat must be supplied to the furnace both to promote the reaction and to maintain the necessary fluidity of the steel as its melting-point rises. The carbon is oxidised to carbon monoxide, which either produces a frothy condition of the slag or bursts through it as numerous isolated bubbles. The escape of carbon monoxide in this latter way produces the appearance of boiling and the bath of steel is commonly said to be "on the boil".

The elimination of carbon can be regularly carried further in the electric furnace than is either usually safe or desirable in open hearth practice. The limitation of bath temperature in the latter case is a question of heat application, and there is thus grave risk of the bath becoming pasty should the carbon be removed too far. The electric furnace, on the other hand, provides a source of heat, by which the temperature of a bath can be raised to a point only limited by the fusing temperature of the refractory lining.

II. *Removal of Phosphorus.*—The removal of phosphorus from a bath of steel takes place in two stages. In the presence of oxide of iron, either dissolved in the metal or present in the slag, the phosphorus is oxidised to P_2O_5, which may, in the absence of another suitable base, combine with excess oxide to form an unstable ferrous phosphate. This compound is again split up by metallic iron to form a stable iron phosphide Fe_3P, which remains in solution in the metal. Lime serves as a most suitable base for fixing the P_2O_5 originally formed, provided it is present in the slag in a sufficient degree of concentration. The combination of CaO as a base with the P_2O_5 is accompanied by evolution of heat, and therefore results in the formation of a stable compound which has the chemical formula $4CaO . P_2O_5$.

The extent to which phosphorus elimination can be carried in arc furnaces is in great measure due to the localisation of heat, which by raising the slag temperature enables the reactions to proceed to a far greater extent than is possible in other furnaces. A slag that may appear pasty or thick will, in the vicinity of the arc zones, be perfectly fluid; this enables the composition of a slag to be varied in order to increase its power of furthering chemical reactions.

If a charge of scrap, lime, and ore has been properly proportioned, the carbon, as has been pointed out, should at no time be high enough in the bath, during the melting down period, to prevent elimination of phosphorus and other metalloids. The removal of phosphorus proceeds gradually from the time a bath is first formed, and should therefore reach a high degree of elimination by the time the entire charge is melted and hot enough for skimming; this is, of course, subject to the maintenance of correct slag conditions, which have been dealt with.

III. *Removal of Manganese and Silicon.*—Manganese and silicon, which may be present in the bath in solution or as definite compounds, are removed by the oxidising action of iron oxide dissolved in the bath or present in the slag. The oxidation proceeds from the initial stages of the melting down period, and is influenced by the amount of carbon present in the bath and the bath temperature. As a general rule there is no difficulty in removing the small percentages of Mn and Si present in ordinary carbon steel scrap under the influence of a basic oxidising slag.

IV. *Removal of Sulphur.*—It has been shown (Dr. A. Mueller) that there is a removal of sulphur during the refining operation under a basic oxidising slag. The reaction which may be expressed by the equation—

$$FeS + 2FeO = 3Fe + SO_2,$$

can, however, only proceed when the $FeS + FeO$ are present in a sufficient degree of mutual concentration.

The figures in the table below (Mueller) show the degree of sulphur removal from three charges of liquid steel during the oxidising and dephosphorising operation :—

	C.	Mn.	P.	S.	Si	Per Cent. S Removal.
Liquid steel charged	·19	·76	·02	·045	trace	
After ore addition	·13	·48	·01	·044	,,	
Before slagging	·09	·30	trace	·033	,,	26·7 per cent.
Liquid steel charged	·2	·64	·03	·056	,,	
After ore addition	·2	·52	·02	·056	,,	
Before slagging	·12	·28	trace	·039	,,	30·4 per cent.
Liquid steel charged	·23	·62	·03	·059	,,	
After ore addition	·22	·44	·01	·048	,,	
,, ,, ,,	·21	·42	·01	·045	,,	
Before slagging	·21	·38	·005	·045	,,	23·7 per cent.

Conditions of Bath before Skimming.—From a chemical standpoint sufficient has been said, both in regard to slag formation and bath reactions, to show that the process up to the time of skimming is conducive to the maximum removal of P, Mn, and Si with reduction of the carbon to a low figure.

Carbon.—The carbon content of the bath should be about ·06 per cent. to ·08 per cent. and if the slag composition and temperature are correct, may be easily judged by the very slight boil which will still be exhibited. Further removal of carbon, accompanied by excessive over-oxidation, is likely to cause trouble in the subsequent deoxidation and desulphurising treatments, and will also cause irregularity of carbon absorption, if carburising with anthracite or other form of carbon is to follow the skimming operation. A spoon sample is usually taken and poured on to an iron plate to form a thin narrow strip, which should bend double without fracture after being quenched in water if the carbon is sufficiently low. A small piece of aluminium thrown into the spoon before pouring will produce a strip of sound steel free from blow holes; powdered ferro-silicon is not satisfactory for this purpose as an excess is likely to cause brittleness, and so vitiate the test.

Phosphorus.—Phosphorus should not exceed about ·012 per cent. in the bath for tool and special alloy steels; this will be usually equivalent to a final percentage of ·018 per cent. to ·02 per cent. in the finished steel. It is difficult to explain the discrepancy, since the increase cannot possibly be due to a reduction of phosphorus from the inappreciable quantity of the oxidising slag left behind after a careful skimming. For castings there is seldom any necessity for reducing the phosphorus below ·03 per cent. in the finished steel.

After working through a few trial heats with a given quality of scrap and correct slag conditions, experience will show whether the phosphorus is sufficiently low when the bath is otherwise ready for skimming. If there is any doubt, a bath sample may be taken at least half-an-hour before it is expected to skim the bath, so that no delay will be occasioned pending the result. A charge composed of ordinary steel scrap will usually be low enough in phosphorus when entirely melted and ready for skimming. Should, however, the phosphorus be too high after melting down a charge, it is often best to pour off the bulk of the slag and add fresh lime and ore to effect the final elimination.

Manganese.—Manganese in the bath may vary from a trace up to 0·4 per cent. according to the carbon and manganese content of the steel scrap. It is preferable that the manganese be 0·15 per cent. or under, although a higher figure is quite permissible, provided it is not the result of improper slag conditions. It is often an advantage to be obliged to add 0·3 per cent. Mn before casting in the form of either spiegel or ferro-manganese, as this generally enables a final adjustment of the carbon content of the steel to be made before pouring without recourse to white iron or hæmatite pig. Scrap of medium carbon and 0·8 per cent. to 1 per cent. Mn will usually melt down to a bath carrying 0·3 per cent. to 0·4 per cent. Mn, unless the slag is carefully watched and kept up to a proper degree of oxidation. It does not always follow that a percentage of manganese in the bath above 0·2 per cent. signifies also a high phosphorus content due to insufficient oxidation, but, generally speaking, the possibility is greater, and caution should be exercised.

When using scrap that leaves over 0·15 per cent. Mn in the bath, it is advisable to determine the manganese in a bath sample taken at a later period, especially where the manganese has to be within close limits in the final specification.

Silicon.—If carbon and phosphorus are properly eliminated as indicated above, the silicon will invariably be reduced to a trace. An unexpectedly high percentage of silicon in the finished steel is sometimes accompanied by high manganese and phosphorus contents, which indicate improper slag conditions before skimming.

Temperature.—Temperature is an important factor in skimming operations, and should always be carefully observed immediately before skimming. Lack of heat often renders the slag thin and difficult to gather, and will also cause the steel to set on the skimming tools and slagging spout. A bath that has not a sufficient reserve of heat to resist scumming over after removal of slag will, if carburising is to follow, result in a low and variable absorption of carbon. This makes the subsequent operation of finishing more difficult and uneconomical, especially when further carbon has to be added in the form of high grade white iron or hæmatite pig. The chemical and physical conditions of the bath should be as nearly as possible the same for successive heats, this being one of the chief factors which lead to economy of production and uniformity of the chemical and physical properties of the product.

Removal of Oxidising Slag.—If the correct slag and temperature conditions are fulfilled, the slag can be removed without any difficulty. The furnace is tilted, and the bulk of slag poured off until the arcs become broken and snappy. The main switch is then opened, and the electrodes raised to allow free movement of the skimming tools over all parts of the bath; neglect to do this often leads to a broken electrode—particularly in the case of graphite electrodes of small diameter—with the probable result of unknown absorption of carbon by the bath. The remainder of the slag is then gently skimmed off, the last remnant being more easily removed after thickening with a few shovels of fine dolomite or lime. Care is taken throughout the operation to prevent an accumulation of slag on the lip of the spout, which makes further removal very difficult.

Skimming appears to be a simple operation, but if properly done, requires considerable skill. Experience alone will enable a man to skim in the shortest time without loss of metal and to the necessary degree of removal, which is a matter of judgment according to the operations that follow. After skimming has been completed, the furnace is again tilted to the normal position and is ready for the next operation.

Carburising.—Carburising is best done by adding to the skimmed bath a calculated quantity of anthracite, or other form of carbon low in ash and volatile matter. The material chosen should contain a maximum percentage of carbon, and be in a physical condition conducive to rapid absorption by the steel. A high percentage of ash will form a slag covering, which mechanically prevents absorption of the last additions made; this applies particularly to high carbon steels, and to furnaces in which the bath area bears a small ratio to its volume. A high percentage of volatile matter (10 per cent. and over) in anthracite is equally objectionable, as it causes the small particles to cake together as soon as they touch the bath, which prevents proper contact and mixing.

If the conditions of the bath are correct before skimming and this latter operation is properly carried out, the addition of carbon will be accompanied by a boil, which is sometimes considerable at first but gradually subsides as the last additions are made. This reaction, at the beginning, must be carefully watched and kept under control by adding only small quantities until the period of violent ebullition is passed; the carbon may then be added more rapidly without fear of a boil-out. This ebullition is particularly useful as it serves to keep the carbon and steel continually in movement. When the practice of adding ferro-silicon is adopted, it is preferred by many to destroy this effect by making the addition before, instead of after, the carbon is added; this method may be followed quite satisfactorily, provided only a small addition of carbon is to be made, but in the case of high carbon steels the slag formed by the ferro-silicon addition, together with the lack of mechanical mixing, make it far more difficult to effect a regular and maximum degree of carbon absorption. The violent boil after the first carbon addition does not take place immediately, but only proceeds after the carbon has reached a certain degree of concentration in the bath, when it then reacts with the dissolved oxides present. The violence of the reaction falls off as the reduction of oxides becomes more complete. With increasing weights of carbon added to the bath, the proportion actually absorbed will similarly increase; this is obvious, since the first addition made may be regarded as lost during the oxide reaction. The ratio of the actual amount of carbon absorbed to the weight of carbon added is conveniently, but perhaps incorrectly, expressed in terms of percentage efficiency, so that for small additions the efficiency will be lower than for large. The "efficiency" will vary between 40 per cent. to 55 per cent., and will be lower still for very small additions. The conditions that favour maximum absorption are:—

(*a*) good bath temperature,
(*b*) freedom of bath from slag,
(*c*) steel not too over-oxidised,
(*d*) low ash and volatile content of the carburiser employed.
(*e*) physical state of the carburiser, such as to prevent loss by dusting and to offer a maximum surface of contact.

Anthracite of good quality is usually employed, and should be carefully sized by rejecting all particles passing through a 30 mesh sieve, together with the oversize from a quarter-inch or, better still, a $\frac{3}{16}$ inch mesh sieve. Certain varieties of Welsh anthracite are very suitable, an excellent sample of which has the following analysis:—

Fixed carbon	91·6	per cent.
Volatile matter	5·3	,,
Sulphur and phosphorus . .	0·8	,,
Ash	about 1·75	,,

The ash and volatile matter are usually rather higher, rising up to 5 per cent. and 7 per cent. respectively. Sulphur and phosphorus should be as low as possible. Anthracite should be stored in a dry place as, if exposed to weather, it will carry sufficient moisture to considerably vitiate the calculated weight. After the last addition of carbon has been made, it is advisable to wait until all action has ceased, as shown by the entire drop of flame from the furnace, before giving a final stir to free any carbon held together by slag. The bath is then ready for the next and final stage of the process, conducted under a powerfully reducing slag.

Basic Reducing Slag.—*Function.*—Up to the time of carburising the process is usually conducted under conditions that exert an oxidising action on the bath of steel. The act of carburising undoubtedly removes a portion of the iron oxide dissolved, but is incapable of carrying the reduction past that point at which there is a chemical equilibrium between the carbon and oxide still remaining dissolved in the steel. It requires, then, a far more powerful reducing agent to deoxidise the steel to such an extent that it will produce a perfectly sound ingot. For this purpose a strongly reducing slag composed of lime, fluor spar and carbon is used, and this is frequently supplemented by ferro-silicon added to the bath after skimming and in very limited quantities during the process of slag deoxidation. Deoxidation is, however, possible without the use of ferro-silicon, but the period of refining is more prolonged.

Another important function performed by this slag is the removal of sulphur, which can be carried to a considerable degree. The slag also serves as a perfectly neutral covering to the bath, so that the latter can be held at any desired temperature without changing its chemical composition; this is exceedingly useful, as it enables the steel to be held in a tranquil, inert condition pending analysis of bath samples, or in the event of shop delays. These conditions are also conducive to the elimination of slag suspended in the bath as minute particles. The slag, when of correct composition, will not contain more than 0·5 per cent. of metallic oxides, so that ferro-alloy additions may be made according to calculation without any allowance for oxidation loss.

Fluxes.—(*a*) *Ferro-silicon.*—Although ferro-silicon cannot be strictly regarded as a flux it is convenient to consider it under this category, as, apart from the part it plays in the deoxidation of the bath, it supplies at the same time a large part of the silica content of the reducing slag. The greater part of the total ferro-silicon addition is made to the bath of steel immediately after carburising. A rich alloy, containing at least 45 per cent. Si, should be used for this purpose, otherwise the chilling effect of the larger quantities required for a poorer grade will prevent proper absorption by the already cooled bath. The alloy should be crushed small to expose as large a surface as possible to the steel and so hasten absorption.

For use after the reducing slag is formed, the ferro-silicon should not contain more than 45 per cent. to 50 per cent. Si, otherwise its specific gravity will be so low that the alloy will only with difficulty pass through the slag and enter the steel. These latter additions of the alloy should also be made in lump form, avoiding small pieces which might be held up in the slag. The most convenient grade is undoubtedly the 45 per cent. to 50 per cent. Si alloy, as it serves equally well for both purposes; the dust and smaller pieces are added to the naked bath, while the lumps are reserved for later use.

(*b*) *Lime.*—Hard-burnt lime is most commonly used, although crushed limestone is occasionally employed. In the latter case the same objections arise as when it is used for the oxidising slag formation; also, the formation of the reducing slag is delayed until all the carbon dioxide has been driven off. The lime should be used in small pieces, or broken up just before use by moistening with water; this is important, as fusion of the fluxes and formation of the slag will otherwise be delayed, more especially if the bath has been well-skimmed before carburising. It is advisable to select a lime that is not too high in sulphur or phosphorus, but apart from this almost any burnt lime is suitable.

(*c*) *Fluor Spar.*—Fluor spar is a most useful flux both for promoting slag formation and for adjusting the fluidity and general character of the slag once formed. Its action is entirely mechanical, and only serves to diminish the stiffness of a limey and almost mono-basic slag by its extreme fluidity when molten. Its action is powerful, and so it is only used sparingly when rectifying the character of the slag after initial formation. Fluor spar is often associated with galena (PbS) and should be examined for this latter mineral, which, if present in any quantity, will render the spar unfit for use. Iron has a greater affinity for sulphur than lead has, so that simple replacement will follow direct contact with the bath.

(*d*) *Silica.*—Sand is sometimes employed as a substitute for fluor spar, either wholly or in part, but is liable to produce a slag too rich in silica, which is usually in the neighbourhood of 25 per cent. when ferro-silicon is used for deoxidising the bath.

(*e*) *Carbon Dust.*—This is added to the slag for reducing the metallic oxides which are present, and for the subsequent formation of calcium carbide. Anthracite, electrode carbon, or petroleum coke is used for this purpose in the form of fine dust, which reduces the risk of carbon passing into the steel and effects reduction in the minimum of time.

Character and Appearance.—The physical characteristics of a proper finishing slag are very well defined, and undergo marked and rapid changes with slight variation in its condition; it is, therefore, an easy matter to adjust the composition and degree of fluidity from time to time, so as to satisfy the conditions most favourable to deoxidation and desulphurising.

The slag first formed by fusion of the lime and spar, and on to which a quantity of carbon dust has been thrown, will at first be brown or brown-yellow in colour and rather stiff; on rise of temperature it will become much thinner, and, if there is sufficient carbon present for reduction of oxides, the colour will change to a pale yellow, and subsequently to white. During this change, a pronounced reaction takes place in the slag, giving it a frothy appearance due to the evolution of carbon monoxide. The surface should at all times appear quite dull if not frothing, a glassy appearance indicating presence of metallic oxides in the slag, or lack of basicity. Experience alone can indicate to which cause the faulty character is due. As a general rule the slag should have a creamy consistency, be white or greyish in colour, fall to a fine powder on cooling from redness, and, when moistened with water, exhibit the presence of free calcium carbide by the smell of evolved acetylene. The peculiar property which the slag possesses of falling to a fine powder is not necessarily an indication of the presence of calcium carbide, since the slag will often fall at almost a red heat before any decomposition of CaC_2 is possible. It is rather an indication of a high lime content and freedom from metallic oxides.

Analyses of Reducing Slags :—

CaC_2	2·1	9·77	5·74	4·59
CaF_2	20·7	20·0	22·6	20·5
CaS	·74	·51	·73	·47
CaO	53·0	46·7	50·1	50·8
SiO_2	14·17	16·21	14·26	15·95
Al_2O_3	3·18	4·69	3·95	3·96
MgO	2·95	1·99	3·65	2·95
Free coke	·80	·66	1·24	1·32

The above are analyses of slags taken from a 15-ton furnace used for refining liquid steel, but usually the SiO_2 is somewhat higher, being about 20 per cent. or more when ferro-silicon is used to effect preliminary deoxidation of the bath. With such strongly reducing slags almost the entire deoxidation can be effected by the calcium carbide in the slag without the use of ferro-silicon, provided the bath is not in a highly oxidised condition. Slag samples rich in calcium carbide usually have a pale grey colour and smooth skin, and fall to a greyish-white powder.

Slag Formation and Control of Bath Deoxidation.—The addition of ferro-silicon immediately following carburisation of the bath has already been mentioned. When this practice is followed, it is usually quite safe to add a quantity equivalent to 0·2 per cent. Si in the bath without any risk of unoxidised Si remaining in the steel. This applies to charges that have been melted and worked down normally from a fairly low carbon scrap, yielding an oxidised bath at the time of skimming. After the ferro-silicon has all worked through, which may be aided by rabbling, the fluor spar is added together with about half the lime, this being done to promote more rapid fusion and formation of a slag covering. After partial fusion of the fluxes added, a liberal quantity of carbon dust is thrown on and allowed to work through for a few minutes. A slag sample is then taken, and if found to be pale brown or yellow, a bath sample can be taken for the analysis of carbon, manganese, and other constituents, after well stirring the bath. However perfect the slag may be, this sample should on no account be taken unless the bath is fairly hot, but well below the temperature of casting, otherwise the sample cannot be accepted as truly representative owing to the non-homogeneity of the bath. By retaining one-half of the lime from the first addition of fluxes, time is saved by the more rapid formation of a slag covering, which, when well fused, enables a spoon sample to be taken without risk of spoiling the spoon. The remainder of the lime is added as soon as the bath sample has been taken, the increasing temperature of the slag being sufficient to maintain the necessary degree of fluidity. The slag should be carefully and frequently observed by taking a sample on an iron bar.

Any tendency to revert to a darker shade of yellow or brown demands an addition of carbon dust; if too thick, a small quantity of fluor spar should be given, and if too thin, lime is required. The change in the appearance of the slag is accompanied by a simultaneous change in the appearance of the smoke issuing from the furnace doors or roof. After the first addition of fluxes, a copious quantity of a pale grey-yellow smoke will be evolved together with much luminous flame; as slag formation proceeds and the oxides become reduced, the smoke will diminish in volume and become whiter and less dense, finally assuming the appearance of a thick, white haze. The flame also will subside and become less luminous. Carbon dust may be used liberally without fear of carbon entering the steel, and is, moreover, essential to the formation of free calcium carbide. The total quantity of fluxes used should be sufficient to form a good covering to the bath, which will prevent carbon absorption by the steel from the carbon dust added.

The process of deoxidation of the bath by slag reaction is a function of time dependent upon the degree of oxidation, slag composition, and temperature, and is more prolonged than when aided by other deoxidisers. Whether ferro-silicon is used or not for effecting a preliminary and partial deoxidation after skimming, it is a common practice to make small additions of this alloy, equivalent to ·05 per cent. to ·1 per cent. Si, at a later stage to assist deoxidation by the carbide bearing slag. These small additions may be made from time to time according to the condition of the steel, as judged by the degree of "wildness" shown on solidification of a spoon sample poured into a small mould. When ferro-silicon is used purely as a deoxidising agent, the Si is oxidised and unites with the unreduced metallic oxides still present to form silicates. The greater part of these silicates certainly passes from the steel into the slag, but the remaining portion will exist either in a state of fine suspension or true solution. In either case the silicates which remain in the steel at the time of pouring will segregate on solidification and be distinguishable under microscopic examination. For this reason, when making certain special classes of steel, it is preferable to use ferro-silicon and other slag forming deoxidisers as sparingly as possible and to utilise rather the deoxidising powers of the calcium carbide bearing slag to its utmost extent.

Slag Reactions.—(*a*) *Removal of Oxygen.*—The removal of dissolved iron oxide from the bath results from the action of ferro-silicon, when used, and of the calcium carbide formed in the slag. In the case of the former the reaction takes place between the Si and O in the bath itself, whereas in the latter case it is purely a contact reaction. The reducing action of ferro-silicon is simply due to the fact that Si has a greater affinity for O than Fe, the reaction taking place with evolution of heat. With regard to the true slag reaction, the power of deoxidation is due to formation of calcium carbide, which, when in contact with a bath containing dissolved oxides, is immediately decomposed with liberation of CO and CaO, as may be expressed by the equations :—

$$(1)\ 4CaC_2 + 3Fe_3O_4 = 9Fe + 8CO + 4CaO,$$
$$(2)\ CaC_2 + 3FeO = 3Fe + 2CO + CaO.$$

The reaction is accompanied by flame resulting from the combustion of the CO liberated. Owing to the liberation of carbon monoxide the amount of flame issuing from the furnace becomes insignificant when the slag is white and the steel freed from oxide, and this is also a sure indication of the satisfactory completion of the reaction. The elimination of oxygen solely by means of the carbide reaction is necessarily slower than when aided by ferro-silicon.

(*b*) *Removal of Sulphur.*—Sulphur can be eliminated to a remarkable degree by the action of a finishing slag having the characteristics previously described. It has been shown that the removal of sulphur is comparatively slow until the slag is free from oxides, as indicated by its white appearance or the presence of calcium carbide; at this point the sulphur reaction proceeds with great rapidity, and its elimination is soon complete. This is clearly demonstrated by the analyses of slag samples taken at different periods during the reducing stage (Dr. A. Mueller) :—

	Colour.	FeO and Fe_2O_3	MnO.	S.
After carburising . .	Greyish-brown	1·25 per cent.	1·44	·39
,, final flux addition .	White granular	·27 ,, ,,	·52	·9
Before tapping . . .	,, powder	·13 ,, ,,	trace	1·22

There is no difficulty in reducing the sulphur in the steel to below ·02 per cent., the sulphur being fixed in the slag as CaS, which can only exist as such in the absence of manganese and iron oxides. This accounts for the negligible reduction of S before the slag becomes white. The reactions which take place may be represented by the following equations, into each of which carbon, either free or combined, enters (Amberg):—

$$(1)\ FeS + CaO + C = Fe + CaS + CO.$$

$$(2)\ 3FeS + 2CaO + CaC_2 = 3Fe + 3CaS + 2CO.$$

The removal of sulphur, according to the first equation, is probably what also takes place in a blast furnace, where the production of low sulphur pig-iron is favoured by a somewhat basic slag and a high furnace temperature. There seems little doubt, therefore, that the first equation represents the correct nature of the chemical reaction. At the higher temperatures of the arc zone, where CaC_2 is formed, the reaction may take the form of the second equation. In the same way, silicon dust or other reducing agent present in the slag will suffice to prevent the reversible reaction according to the equation:—

$$FeS + CaO \rightleftarrows CaS + FeO.$$

Alloy Additions.—The final addition of ferro-manganese and ferro-silicon for specification purposes should not be made until after the steel has been "killed" by slag reaction alone, or with the aid of small ferro-silicon additions, and then not less than five minutes before actually casting. When ferro-silicon has been used to assist deoxidation, it is usual to base the calculation of the final addition of this alloy, for specification purposes, upon a bath content of 0·1 per cent. Si. Nickel, ferro-chrome, and other alloys should be added before the manganese and silicon, and the bath rabbled to ensure thorough mixing and homogeneity. When large additions are to be made the alloys should be added in small quantities at a time, and it should be ascertained that the bottom is perfectly clear by feeling it with an iron bar before each addition is made. By taking this simple precaution there is never the slightest risk of the bath chilling and setting on the bottom of furnaces in which no hearth heating is developed. Aluminium, if used at all, should not exceed 4 oz. to the ton when making ingots, but for foundry work it is often advisable to increase this proportion considerably as a precautionary measure against subsequent oxidation, especially in the case of green sand casting.

Temperature Control.—Temperature control is an important factor in refining and finishing electric steel. After skimming and carburising, the bath temperature will be very low, and it is advisable, on again heating, to operate at a load that will fuse the fluxes added and raise the bath temperature to a degree suitable for spoon sampling at the end of about 15 to 20 minutes. The ratio of this load to full load will be found by experience, and will vary according to the furnace capacity and the temperature of the steel after carburising. The temperature of the bath when sampling should be moderate, but not hot enough for casting, and should then be held with little further rise until the steel is nearly ready for casting, or for receiving large additions of ferro-alloys. In the case of furnaces of small capacity—30 cwts. or less—it is by no means easy to so regulate the temperature, as a small margin of power over and above that equivalent to the constant radiation loss will suffice to raise the temperature rapidly. When small additions of ferro-silicon are made during the process of deoxidation, it is of the greatest importance that the bath temperature should not exceed, but preferably remain below, normal casting temperature, otherwise difficulty may be encountered in "killing" the steel, and high silicon contents result.

Various methods of judging temperature are employed. Some merely take a sample in a spoon about 3 inches in diameter, allow it to remain a few seconds, and then upset the steel on to an iron plate; if the steel runs freely, and leaves the spoon perfectly clear, it is deemed hot enough for casting. Another method commonly adopted is to cast a small flat rectangular ingot about $\frac{1}{2}$ inch to $\frac{5}{8}$ inch thick, which is quenched in water and then broken in half; if the steel is sufficiently hot, the fracture will exhibit a needle-shaped structure radiating from the bottom and sides, and showing the usual plane of junction running at an angle of 45° from the bottom corners of the fractured surface. Both these methods have the disadvantage that their accuracy is entirely dependent upon the manner in which the spoon sample is taken, and are therefore too subject to possible error. The best method, which, however, is less economical, consists of plunging a clean ½-inch diameter iron rod into the bath to touch the bottom, and holding it immersed for a period of five to seven seconds according to the character of the steel, at the same time gently moving it in an axial direction. The rod, when rapidly withdrawn, will indicate the temperature of the bath from top to bottom by the extent to which the steel has either adhered to it or cut into it, as the case may be. Generally speaking, for low medium to high carbon steels the bar should be just left clean after immersion for five seconds. The first methods of taking temperature may well serve as a guide during the refining operation, but the actual casting temperature is more accurately judged by the rod test.

Calculation of Additions.—As a starting point for calculations of carbon and metallic additions, the approximate weight of the bath of steel must be known. There is always a certain melting loss which will vary, according to the nature of the scrap used, from 3 per cent. up to 13 per cent. or even more, the average for heavy turnings being roughly 7 per cent. This melting loss can only be determined by observation of losses in previous heats when similar scrap was melted. The gross weight of steel cast, less the weight of all ferro-alloy additions made, when deducted from the weight of scrap charged, will give the loss incurred during the entire operation with sufficient accuracy. The following examples show the method of calculating the additions for a high carbon and a chrome steel heat, the same system being adopted for any variety of alloy steel.

Example I.—It is required to make a high carbon steel to the following specification :—

C	1·0	per cent.
Mn	·25	,,
Si	·15	,,
P below . . .	·02	,,
S ,, . . .	·02	,,

Weight of scrap charged . . . . . . 6000 lb.
Assume a 7 per cent. melting loss.
Deduct weight lost in melting (calculated) . . 420 ,,

Actual weight of steel in bath at skimming . 5580 ,,
Estimated carbon content of bath . ·06 per cent.
Required to carburise up to . . ·95 ,, ,,
(i.e. ·05 per cent. less than specification).
∴ increased per cent. C to add to bath ·89 per cent.

$$\text{i.e. weight of carbon to be added to bath} = \frac{\cdot 89 \times 5580}{100} \text{ lb.}$$

If anthracite containing 90 per cent. fixed carbon is to be used for carburising the bath, and the carbon absorption is assumed to be 50 per cent.,

then weight of anthracite to be used

$$= \frac{\cdot 89 \times 5580 \times 100 \times 100}{100 \times 50 \times 90}$$

$$= \underline{110 \text{ lb.}}$$

Supposing the bath sample taken for carbon and manganese analysis after proper formation of the finishing slag is found to contain,

C . . ·92 per cent.
Mn . . ·08 ,,

then the carbon will have to be raised a further ·08 per cent. and the manganese ·17 per cent. before tapping.

For the addition of manganese there is the choice of either spiegel or ferro-manganese, in which the relative percentages of manganese and carbon contained are as 4 to 1 and 11 to 1 respectively.

Since only ·17 per cent. Mn is required to be added, it is obvious that spiegel is preferable to use, as roughly ·04 per cent. carbon can at the same time be added.

Increased per cent. Mn to add to bath = ·17 per cent.
Per cent. Mn in spiegel . . . = 21·0 ,,
Per cent. C in spiegel . . . = 5·0 ,,

$$\text{Then weight of spiegel required} = \frac{\cdot 17 \times 5580 \times 100}{100 \times 21}$$

$$= \underline{45 \text{ lb.}}$$

The bath will now contain ·92 per cent. + ·04 per cent. C or ·96 per cent. C, and is therefore still too low for the specification.

The most accurate means of increasing the carbon content is by the addition of white iron or hæmatite pig-iron, which can be obtained with not more than 1·5 per cent. Si, although 2·5 per cent. is a more general figure. Assume, however, that white iron is to be used containing 4·5 per cent. C, then since the finished steel is to contain 1 per cent. C the white iron only contains 3·5 per cent. C available for addition to the bath.

Now the calculated carbon content of the bath after the addition of spiegel . . . . . = ·96 per cent. C.
∴ per cent. C to be added by the white iron . . . . . = ·04 per cent. C.
∴ the wt. of white iron required . = $\frac{·04 \times 5580 \times 100}{3·5 \times 100}$ lb.
= 64 lb.

The weights, therefore, of carburisers and ferro-manganese required are :—

Anthracite (added after skimming) .	110 lb.
Spiegel (added 5 minutes before tapping)	45 ,,
White iron (added before the spiegel) .	64 ,,

In the above calculations it will be seen that no allowance has been made for the increase in the weight of the charge due to the anthracite and ferro-manganese additions; the increase is so small that the error is negligible. In the case of the white iron addition, the carbon content is only just over four times the specification figure, so that the calculations have to be based on the carbon available for carburising. The same remarks might equally well apply to the carbon added by spiegel, but the carbon content is, in this case, slightly higher and the actual weight of alloy less, so that the discrepancy is negligible when compared to the crude estimation of melting loss which always varies from heat to heat. It has been previously stated that no loss of alloys, added after the formation of a white finishing slag, is allowed for in the calculations.

Example II.—It is required to make a chrome steel to the following specification:—

C . .	1·2 per cent.
Cr . .	3·0 ,,
Mn . .	·4 ,,

Assume the same initial charge and melting loss as in the previous example:—

The actual weight of steel in the bath at skimming = 5580 lb. For the addition of chromium suppose there are two grades of ferro-chrome available, both containing 65 per cent. Cr, but one with 5 per cent. C and the other 9·2 per cent. C. These are the two commonest brands of ferro-chrome made, and it will be seen to what use the varying carbon contents can be put for the final adjustment of carbon, while adding the correct amount of chromium. Suppose equal chromium equivalents of the two alloys are used, and the per cent. carbon added by them is calculated, then, if the bath sample after carburising has a carbon content which, together with the carbon added by one grade of the ferro-chrome, would be either above or below the specification figure, the correct adjustment may possibly be made by varying the proportions of the high and low carbon grades used. In this example, however, an addition of manganese is necessary, so that there is a still further means of adjusting the carbon by the addition of either spiegel, ferro-manganese, or a mixture of the two.

It will be supposed that it is preferable to use only the 4 per cent. to 6 per cent. C ferro-chrome, which for this specification will have 65 per cent. minus 3 per cent. Cr and 5 per cent. minus 1·2 per cent. carbon available for addition to the bath.

Before calculating the quantity of anthracite for carburising, it is necessary to know about how much carbon will be added by the ferro-chrome and spiegel. Spiegel has, in this case, been chosen in preference to ferro-manganese for the purpose of calculation, so that in the event of the carbon found in the bath sample being lower or higher than was intended, there is still a possibility of adjustment by using the high carbon ferro-chrome in place of the 4 per cent. to 6 per cent. C grade, or ferro-manganese in place of spiegel respectively.

If both the lower carbon alloys were used, and the carbon was found to be too high in the bath after carburising, then there would be no means of reducing it to within the specification limits. Therefore, since carbon can easily be added by white iron or hæmatite pig additions, it is always a safer policy to purposely under-carburise slightly with anthracite, and rely upon alloys and, if then necessary, white iron or pig for the subsequent and final carburisation. Calculating the ferro-chrome first:—

Wt. of steel in bath . . . = 5580 lb.
Per cent. of Cr required in bath = 3 per cent.
Then weight of Cr to add (using the 4 per cent. − 6 per cent. C grade) . . $= \dfrac{3 \times 5580 \times 100}{100 \times (65 - 3)}$
= 270 lb.

Since the available percentages of Cr and C in the ferro-chrome are as 62 to 3·8, it follows that an addition of 3 per cent. Cr is accompanied by an addition of $3 \div \dfrac{62}{3{\cdot}8}$ or ·184 per cent. C.

Since spiegel is being used for the addition of say ·3 per cent. Mn (assuming ·1 per cent in the bath),

Then the carbon added will equal ·07 per cent. (about).

Assume per cent. C in the bath before skimming = ·06 per cent.
and per cent. C added in ferro-chrome . . = ·18 ,,
and per cent C added in spiegel . . . = ·07 ,,

Then the total carbon in the bath, if the above additions are made without any addition of anthracite, would be ·31 per cent.

Now per cent. C required by specification . = 1·2 per cent.
Total per cent. C in bath with alloy additions = ·31 ,,

∴ per cent. C to add as anthracite . . = ·89 ,,

Actually it is safer to aim rather lower than the calculated figure. Then per cent. C to be added . . = ·8 (say)

Now weight of steel in bath at skimming = 5580 lb.
Per cent. C required to add as anthracite = ·8 per cent.

Then wt. of anthracite required, assuming 90 per cent. C in the anthracite and 50 per cent. C absorption

$$= \frac{\cdot8 \times 5580 \times 100 \times 100}{100 \times 50 \times 90}$$

$$= \underline{99 \text{ lb.}}$$

Supposing the bath sample, taken after formation of the finishing slag, was found to contain—

C . . ·79 per cent.
Mn . . ·12 ,,

The per cent. C in the bath is lower than was expected from calculation, and it will be necessary to use the high carbon ferro-chrome in place of the lower carbon grade either wholly or in part. The spiegel addition must therefore be calculated first.

Per cent. Mn found in bath . . = ·12 per cent.
,, Mn required by specification = ·4 ,,
∴ ,, Mn to be added as spiegel . = ·28 ,,

It is known that about 120 lb. of ferro-chrome will be added later, so that this weight may be added to the weight of the bath.

Original wt. of bath at skimming . = 5580 lb.
wt. of ferro-chrome added . = 120 ,,
wt. of bath for Mn calculation = 5700 ,,

The above allowance is really unnecessary in practice, and is done in this example to convey the principle which must be applied in the case of large additions of alloys.

$$\text{Wt. of spiegel required} = \frac{\cdot28 \times 5700 \times 100}{100 \times 21} \text{ lb.}$$

$$= \underline{76 \text{ lb.}}$$

Since the Mn content of spiegel is about four times the carbon, the per cent. C added by spiegel . . . . . = ·28 ÷ 4
= ·07 per cent.

Now per cent. C found in the bath sample = ·79 per cent.
and per cent. C added as spiegel . . = ·07 ,,

∴ Total carbon without addition from ferro-chrome . . . . = ·86 ,,
Per cent. C required by specification . . = 1·2 ,,

∴ C to be added by the ferro-chrome . . = ·34 per cent.

From this figure it is obvious that the bulk of the ferro-chrome must be added as the high carbon alloy in which the available Cr and C are (65 – 3) per cent. and (9·2 – 1·2) per cent. respectively, the available carbon being therefore practically one-eighth the chromium. If this alloy only is used, and per cent. Cr added as ferro-chrome

= 3 per cent. (i.e. specification figure),

then per cent. C added
by this alloy = 3 ÷ 8
= ·37 per cent.

But only ·34 per cent. C is actually required, so that a portion of the high carbon grade must be replaced by the lower grade. Since the per cent. Cr is the same for both, the weight of the alloys, if mixed, will still be the amount previously found, i.e. 127 lb. Trying a proportion of 100 lb. high carbon alloy, and 27 lb. of the lower carbon grade, and knowing that 127 lb. adds 3 per cent. Cr to the bath,

then per cent. Cr added by the higher C grade

$$= \frac{100 \times 3}{127} = 2{\cdot}36 \text{ per cent. Cr},$$

and per cent. C added by the higher C grade
= ·29 (i.e. one-eighth the chromium).

Again the per cent. Cr added by the lower C grade

$$= \frac{27 \times 3}{127} = {\cdot}635 \text{ per cent. Cr},$$

and per cent. C added by the lower C grade = ·04 per cent.
∴ Total carbon added by the mixed alloys = ·33 per cent. C.
The above proportion of the two alloys is near enough.
The additions, then, for the charge will be :—

Wt. of anthracite for carburising . = 99 lb.
Wt. of 8-10 per cent. C ferro-chrome = 100 ,,
Wt. of 4-6 per cent. C ferro-chrome . = 27 ,,
Wt. of Spiegel . . . . . = 76 ,,

Tapping.—There are a few points, which might be mentioned in connection with pouring, that may materially assist in preventing any deterioration in the quality and composition of the steel during transfer from the furnace to the ladle. A basic reducing slag, if in proper condition, is creamy in consistency, and has no power of cohesion, as in the case of a vitreous or siliceous slag. The particles are easily broken up, and, owing to their high degree of infusibility, do not readily escape from a mass of molten steel, when once entrapped. It is, therefore, preferable to hold back the slag when pouring into a ladle, and only allow it to pass over the spout with the last portion of steel. To effect this without running any risk of the slag adhering to the banks and remaining in the furnace, a small addition of fluor spar may be given just before pouring, in order to mechanically increase its fluidity at a reduced temperature. In many cases a special brick or a tapping spout (Fig. 80) is used, which holds back the slag until the steel has passed over into the ladle. It is also a good practice to hold the ladle for five minutes before teeming, as this will offer an opportunity for entrapped slag and gases to rise. The spout should always be perfectly dry, so that there may be no risk of the steel boiling on it, and becoming oxidised.

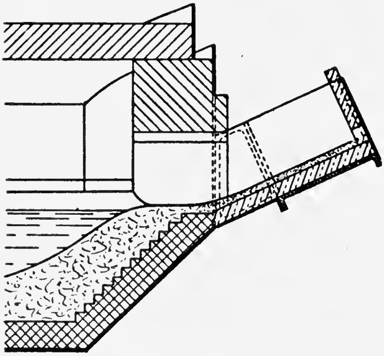


FIG. 80.

Furnace Tools and Manipulation.—The tools shown in Fig. 81 are used to conduct the operations of fettling, skimming, slag sampling and charging heavy pieces of scrap. The dimensions shown are suitable for a 7-ton furnace, and will be rather less for smaller furnaces. The skimming rakes should be used with care, as under the best conditions they require frequent renewal of blades. If the bath is cold, the rakes will become covered with steel, and, if too hot, will be badly cut away unless withdrawn in time. The hooked bar is useful for clearing scrap off the banks, and for taking slag samples. The fettling shovel is best made as shown, so that it can be drawn back on to the furnace door sills without lifting.

Fettling.—Fettling is a most important operation, upon which not only the life of a lining depends, but also the ease with which the proper metallurgical conditions can be maintained. Fresh dolomite crushed to pass a $\frac{3}{4}$ inch or $\frac{1}{2}$ inch mesh, and free from slaked dust, should be used. Care must be taken that the dolomite thrown on to the slag line does not roll down the banks and build up the bottom, which is a common error with unskilled furnacemen. The slag line is always fettled as far as possible, and then the furnace is charged with scrap up to that line. Turnings or other small scrap, when used, are best charged round the foot of the fettled slag line, and will form a seating on which more dolomite can be banked up. Dolomite can then be used liberally, and, with judicious fettling, may be built up as a facing to the lower part of a badly-cut silica wall and so greatly prolong its life. The door jambs and spout also require very careful attention, the former being kept in shape with crucible ganister or magnesite powder mixed with just enough clay to make it bind; the spout is best lined with any kind of moulding sand, which is thrown on after removal of the slag, and beaten down to the correct shape. Ganister, although largely used, is not so good, as it requires far more drying and cracks when dry, while moulding sand, with a small percentage of water, only requires

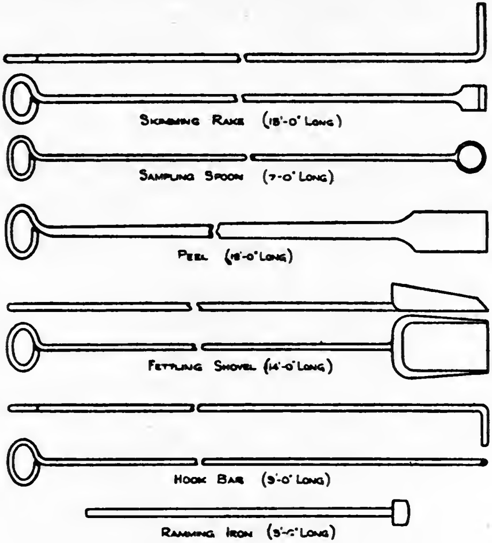


FIG. 81.

a good skin drying and will not cause the steel to boil up on passing over.

ACID PROCESS.

The chemistry of steel-making by the acid process is very similar, whether it is conducted in electric or gas furnaces, or even in the converter. In each case the removal of carbon, silicon, and manganese from the raw material, whether it be pig-iron, steel scrap, or a mixture of the two, follows from simple oxidation, which, except in the converter, proceeds under the direct influence of iron oxides and—to a more limited degree—oxygen in the furnace atmosphere. The acid process, as in the case of the basic process, may be employed for the conversion of steel scrap into ingots or, more generally, castings, as well as for refining molten oxidised steel.

The process, as applied to the working of cold charges, may be briefly divided into three stages:—

(i) Melting down under oxidising conditions.
(ii) Boiling out carbon under an oxidising slag.
(iii) Addition of alloys and finishings.

General Outline.—In the acid process there is no reducing period during which dissolved oxides and sulphur are removed, and it is therefore far more important than in basic working to melt under conditions that do not conduce to over-oxidation of the bath. One great advantage is the economy resulting from the use of only one slag, which makes it possible to produce a rather larger output than is possible with basic furnaces of similar power capacity. If, on the other hand, it is necessary for purposes of carburising to remove the first slag and then form another, this economy will naturally disappear.

The acid process is more especially suitable for the manufacture of castings from raw materials (usually steel scrap) sufficiently free from both phosphorus and sulphur to meet the required specification. Steel made for foundry purposes will not contain as a rule more than ·35 per cent. C, so that its manufacture is possible without any carburising addition other than ferro-alloys; this makes it possible to operate with one slag, so that the full benefit of the process is derived in this particular application. To prevent over-oxidation of the bath during melting, it is advisable that the charge should contain a sufficient quantity of carbon, so that when entirely melted there may still be at least ·3 per cent. C in the bath. Mild steel scrap, which would melt to form an over-oxidised bath with a carbon content below this figure, is usually mixed with a small quantity of pig-iron or preferably carbon dust, which, being absorbed on melting, helps to limit the extent of bath oxidation. Manganese and silicon are removed by the oxide of iron, which is either added to the slag in the form of ore, or results from oxidation of the scrap in the furnace. Carbon will also be reduced until the slag becomes so impoverished in metallic oxides that further reaction becomes impossible. The carbon remaining in the bath after fusion of the charge may be further reduced by additions of ore to the slag, until it is sufficiently low for the final addition of spiegel or ferro-manganese to bring the bath within the specification figures for carbon and manganese. Before describing in detail the different operations during the melting down and finishing of a charge of scrap, it may be advisable to indicate briefly the consecutive steps in the furnace manipulation, and the order and nature of the chemical reactions that take place.

The following description will apply more particularly to furnaces having a capacity of not more than three tons, but, with slight modification of the details of manipulation, will be equally applicable to larger furnaces:—

1. Hand charging of scrap into the previously heated furnace.
2. Doors closed, load on and melting begins.
3. Scrap melts under the electrodes, forming pools of metal covered by a slag formed from the siliceous fettling material, dirt, etc. Oxidation of carbon, manganese and silicon begins, if sufficient oxide present.
4. Small quantity of ore and other fluxes added to the slag, if necessary; chemical reactions proceed slowly.
5. Melting proceeds until bulk of scrap has melted to form a large bath.
6. Further addition of scrap made and melting proceeds.

7. Repetition of (6), if necessary.

8. Entire charge melted; carbon in bath too high for finishing, and Mn and Si low. Addition of ore made.

9. Boil begins and carbon is removed, until the slag becomes impoverished in iron oxide and the boil subsides.

10. Further ore added, if carbon is still too high.

11. Carbon content of bath sufficiently low; iron oxide in slag reduced to a figure incapable of producing further oxidation of carbon.

12. Temperature of the bath adjusted as required, and finishing additions made before casting.

13. Load off; steel poured.

Choice of Scrap.—Scrap suitable for the acid process must not contain, when melted, more phosphorus and sulphur than the specification of the steel demands. Carbon contamination is not such a serious matter in the case of acid scrap for the reasons already indicated, but should not exceed a degree that would unduly lengthen the boiling down operation after complete fusion of the charge. Foreign matter and dirt present in carelessly collected and stored scrap will usually be siliceous in character, and is therefore less harmful to an acid lining. On the other hand, the scrap should preferably be clean and free from rust, so that it may be melted to a bath containing not less than ·3 per cent. to ·4 per cent. C and a minimum amount of dissolved oxide of iron.

With regard to the shape, size, and other physical conditions which influence the ease of the furnace manipulation and mechanical control, the remarks which have already been made in reference to scrap suitable for the basic process will equally apply.

Method of Charging.—The method of charging miscellaneous scrap in a manner most favourable to the maintenance of a steady load and other desirable conditions is substantially the same for both acid and basic processes, but it should be noted that pig-iron or carbon dust, when used, is generally mixed with the scrap near the bottom in the initial charge. This is done so that the carburising action may proceed as soon as the metal forms a pool, and thereby prevent an undue absorption of oxide as melting continues.

Formation and Function of Acid Slags.—The functions of an acid slag are twofold :—

(i) To reduce the carbon, manganese, and silicon in the bath to a desired percentage.

(ii) To serve as an inert covering of the bath of steel while ferro-alloy additions are being made, or whenever it is desired to prevent further chemical reaction taking place before pouring.

The essential constituents of an active acid slag are SiO_2, FeO and, generally, small quantities of CaO, Al_2O_3, MnO, and Fe_2O_3. The SiO_2 is derived from the acid hearth and the loose sand or ganister used as a fettling material. A small quantity of lime is sometimes added after the preliminary formation of the slag, generally for the purpose of thinning it when the FeO content has fallen to a low figure.

There is usually sufficient iron oxide, derived from the rust on the scrap or from oxidation while melting, to combine with the silica for the formation of a slag covering. Other bases, such as Al_2O_3 and CaO, will be readily absorbed by such a ferrous silicate slag which, owing to its high silica content, is powerfully acid in character.

If the scrap used is exceptionally clean and the oxidation loss during melting very small, it may be necessary to add iron ore after a small bath of steel has been formed. This is done to provide the iron oxides necessary for the removal of carbon, silicon, and manganese, and at the same time to open out or thin the highly siliceous, pasty slag; the free oxygen of the Fe_2O_3 plays an important part in the oxidation. The removal of carbon from the bath, which is always evidenced by a distinct boil due to evolution of carbon monoxide gas, only proceeds where the temperature of the slag and steel is above the reaction temperature. Accordingly, the oxidation of carbon only occurs in the neighbourhood of the arcs, until the entire bath with its slag covering is hot enough for the reaction to take place over the whole surface. An absence of "boil" during the melting down period does not, then, necessarily indicate lack of iron oxides in the slag. This latter condition can best be judged by the colour of a broken sample. It has been previously stated that the carbon content of the charge and the amount of FeO and

Fe_2O_3 passing into the slag or added to it as ore, is usually so balanced that the carbon content remaining in the bath after complete fusion of the charge will be about ·3 per cent., this being done to prevent over-oxidation. As the removal of carbon from the bath proceeds, the slag becomes impoverished in metallic oxides, until finally it may be no longer capable of promoting oxidation, and the boil ceases. This reduction of the oxides is accompanied by very marked changes in the appearance of the slag when cold, and enables any subsequent addition of ore, for the further removal of carbon to a desired figure, to be correctly estimated.

An acid slag capable of boiling out carbon will usually contain over 25 per cent. of combined FeO and MnO, which is rather less than in open hearth practice. The percentage of silica in the final slag will also be much higher, so that rather more lime is sometimes needed to obtain the required fluidity. The pastiness of the slag at tapping is a quality that is very useful in foundry practice where ladle lip-pouring is adopted, as the slag can be so readily held back in the ladle or skimmed off.

The varying percentage of carbon in a bath during the "boil" can be judged by observing the fracture of bath samples; such tests are generally made until the carbon is thought to be nearly low enough, when a test sample is sent to the laboratory for a carbon determination. Under proper conditions of working, the slag should cease to be active and become almost incapable of further carbon oxidation, just when the percentage of carbon in the bath has fallen to the required figure. This, of course, requires considerable skill and judgment when making the additions of iron ore during the boiling down period. If the carbon is reduced below the desired figure it can at once be raised by the addition of pig-iron, a sufficient quantity being added to increase the carbon in the bath by the desired amount, and at the same time to compensate for any subsequent loss by slag reaction; by this means the correct bath and slag conditions may be simultaneously obtained.

The elimination of carbon by boiling gradually proceeds in successive stages, following each of the several small additions of iron ore which are made from time to time to promote the reaction. These additions are made more cautiously as the carbon approaches the desired percentage.

Summarising the foregoing description of slag formation and reaction, it will be seen that the necessity of carefully controlling the metallic oxides present in the slag is even greater than in the basic process, where over-oxidation of the bath can be almost completely corrected under a reducing slag without the aid of the final ferro-alloy additions.

Physical and Chemical Characteristics of Acid Slags.—The colour and fluidity of electric furnace acid slags change very markedly during the period of boiling. Assuming there is ·3 per cent. to ·4 per cent. carbon in the bath of steel after complete fusion of the charge, there should then be almost sufficient available metallic oxides in the slag to reduce the carbon to the desired percentage. The oxides present in such a quantity will colour the slag light or dark brown, but, as the slag becomes impoverished in these oxides, the colour of the slag fracture will progressively alter. In the regular operation of a furnace carrying the same quantities of steel and slag in successive heats, the colour of the slag may be used to indicate the percentage of carbon which it is still capable of boiling out without further ore addition. This being the case, it is important to examine the slag at frequent intervals and, by knowing the approximate carbon content at a particular moment by a fracture test, it is possible to estimate fairly well just how much ore to add for the further removal of carbon required. In colour the slag successively passes through various shades of yellow, until it finally becomes very faintly blue-green, and is then no longer capable of producing a vigorous boil. The viscosity will become greater by the partial removal of the iron oxide base, and may then be adjusted by small additions of some other base, such as lime. The considerable viscosity of electric furnace acid slags is mainly due to the high percentage of silica, which frequently reaches 65 per cent.

A typical analysis of an electric furnace acid slag just before tapping is as follows :—

SiO_2	64·0	per cent.
FeO	12·36	,,
Al_2O_3	7·27	,,
MnO	10·62	,,
CaO	5·9	,,
MgO	Trace	
	100·15	per cent.

The combined FeO and MnO are considerably lower than in acid open hearth slags, which indicates that the oxidation of carbon may be effected with a less oxidised slag and, probably, with less tendency towards over-oxidation of the steel.

Ferro-Alloy Additions.—Alloy additions are calculated and made in the same way as in the basic process, with the exception that 10 per cent. to 20 per cent. loss of Mn and Si must be allowed for, without, however, counting on any loss of the carbon added. The addition of spiegel, ferro-manganese, and ferro-silicon is made only after the slag is pale green, and low in active oxides. The bath will then exhibit only a very slight boil, and the carbon content, either determined or calculated, is no longer liable to appreciable variation. The carbon added to the bath as a constituent of the ferro-alloys will not suffer any loss, but, owing to the slight oxidation of the bath at the end of the boil, there will be a small loss of manganese and silicon, which is allowed for as above. The proper allowance to be made for loss on this account and by a slight slag reaction can easily be determined by experience.

CHAPTER VIII.

LIQUID STEEL REFINING.

In the preceding chapters, the methods by which high grade steel is made from miscellaneous steel scrap have been dealt with and classified according to the acid or basic character of the slag used. The chemical reactions upon which the basic and acid processes depend are mainly due to the interaction of molten slag and metal, and therefore proceed independently of the action of fusion. The conditions under which melting takes place certainly influence the chemical character both of the bath of metal and of the slag, and in this way alone does the process of melting affect the chemical reactions which follow, when the temperature and chemical conditions of the bath and slag are satisfied.

For these reasons, the process of making highly refined steel from cold charges may be regarded simply as a process of melting, followed by a period during which conditions suitable for chemical reactions are maintained, although in practice the two phases proceed simultaneously. In liquid refining the melting phase is eliminated and the chemical phase, when conducted under oxidising or non-oxidising slags, may be considered as identical in its functions, irrespective of how the molten steel has been produced. The advantage of liquid refining is economic rather than technical, as cheaper methods of producing liquid steel can be employed than is possible by melting scrap with electric energy. The electric furnace, if basic lined, may be used for purposes of dephosphorising followed by carburising, desulphurising and deoxidising treatments, and when acid lined, is used for the sole object of deoxidising liquid steel already low enough in phosphorus and sulphur.

Liquid Refining in Basic Furnaces.—The refining operations usually include removal of phosphorus as a preliminary to carburising and the subsequent treatment for the removal of sulphur and dissolved oxides. Sometimes the liquid steel may not require further elimination of phosphorus, in which case it is carburised in the transfer ladle, or in the electric furnace itself, prior to the addition of fluxes for the formation of a reducing slag. When cold blown acid bessemer steel is electrically refined for further removal of phosphorus, it is sometimes necessary to add about ·3 per cent. carbon to the steel in the transfer ladle, so as to minimise the risk of skulling by causing a slight lowering of the melting-point; this practice is generally followed when the time taken in transfer is considerable.

Both bottom teeming and lip pouring ladles are used for transferring the liquid steel, the latter type being used only when the slag covering is viscous and easily skimmed. In either case, a special launder is used to convey the steel from the ladle to the electric furnace, so that the steel may flow clear of the door sills on to the hearth. When a dephosphorising treatment is necessary, both iron oxide, in the form of iron ore or mill scale, and lime are shovelled into the furnace whilst the steel is still being poured. These fluxes quickly fuse and form a suitable basic oxidising slag under which phosphorus removal is rapidly promoted. When the phosphorus removal has been carried far enough the bath is skimmed and carburised, if necessary; fresh fluxes are then added, and the process of deoxidation and desulphurising followed in the same manner as described in Chapter VII. The table on the opposite page gives particulars of the materials used, time occupied, and power consumption per ton of metals charged for three typical heats, in each of which about 11 tons of liquid bessemer steel were dephosphorised and finished in a basic lined electric furnace.

The power was supplied up to the time of skimming at about 2000 K.V.A., and reduced to a much lower figure during the last part of each heat.

The following typical analysis of the steel, before and after refining, shows the extent to which the quality is improved.

	Analysis of Bessemer Steel.		*Analysis of Finished Steel.*
C	·05 —·10	per cent.	
Mn	·05 —·10	,,	
Si	·005—·015	,,	·09 per cent.
S	·035—·07	,,	·025 - ·035 per cent.
P	·095	,,	·015 - ·04 ,,

	I.	II.	III.
Weight bessemer steel in lb. .	26,860	25,060	25,900
,, Scale ,, ,,	700	800	600
,, Ore ,, ,,	100	900	400
,, Lime ,, ,,	800	800	600
Current on	8 hrs. 15 mins.	4 hrs. 30 mins.	3 hrs. 10 mins.
Began skimming . . .	9 hrs. 5 mins.	5 hrs. 30 mins.	3 hrs. 50 mins.
Finished skimming . .	9 hrs. 15 mins.	5 hrs. 40 mins.	4 hrs. 10 mins.
Time for skimming . .	10 mins.	10 mins.	20 mins.
Weight crushed electrode (for carburising) . .	150	—	—
,, Lime	700	800	600
,, Fluor spar . .	325	290	275
,, Coke dust (added to slag) . . .	200	270	150
,, Ferro-manganese .	270	200	150
,, Ferro-silicon, 50 per cent. . . .	86	90	20
,, Nickel . . .	—	820	—
,, Other ferro-alloys .	—	—	580
Time tapped	10 hrs. 10 mins.	7 hrs. 0 mins.	5 hrs. 0 mins.
Total time	1 hr. 55 mins.	2 hrs. 30 mins.	1 hr. 50 mins.
Units consumed . . .	3100	4200	2600
Unit per ton charged . .	255	360	218

The removal of sulphur and phosphorus can always be carried to the lower limit when specified, and the silicon can also be kept low if necessary. The physical properties of electrically refined mild steel show an increase of 15 per cent. in the ultimate strength as compared with basic open hearth steel, but at the same time the elongation is decreased by 11 per cent. ; this comparison is shown in a table of tests compiled by C. G. Osborne for a paper read before the American Electrochemical Society in 1911.

Liquid Refining in Acid Furnaces.—The theory and practice of deoxidising liquid steel in an acid-lined electric furnace has been carefully studied both in America and Germany. Various theories have been advanced to explain the exact manner by which deoxidation proceeds, but the several suggested chemical reactions responsible for the removal of oxygen by this process are not substantiated by any conclusive proof. Thallner, in 1913, advanced an ingenious theory to explain the physical characteristics of cast steel, based upon physical suppositions rather than upon chemical composition. According to his theory the quality of steel was greatly influenced by the size of the molecules when in the liquid condition, the physical properties being improved the smaller their size. This particular condition of a bath of steel was, he considered, not only a function of temperature, but also dependent upon the combined effect of several chemical reactions, in which carbon plays an essential part. The reactions which promote the deoxidation of steel in an acid-lined furnace, according to this theory, are briefly as follows:—

(*a*) Reduction of silicon from the acid hearth, with the formation of silicon carbide.

(*b*) Decomposition of the silicon carbide by metallic oxides dissolved in the steel and present in the slag, with formation of iron carbide.

(*c*) Partial decomposition of the iron carbide by the oxides of the slag.

The silicon reduced from the lining does not then remain in the steel, but indirectly causes the formation of iron carbide to which is ascribed the specially fine grain obtained. This is only possible so long as oxide of iron is present in the slag or bath in sufficient quantity to split up the silicon-carbide first formed, otherwise silicon will be reduced from the lining in increasing quantity and remain in the finished steel. According to the process evolved on the above assumption and practised at the works of the Lindenburg Steel Company at Remscheid-Hasten, Germany, the carbon necessary for the above reaction is introduced into the bath in the form of briquettes, consisting of carbon and iron filings or borings. The reduction of silica to silicon is mostly from the hearth lining and only to a lesser degree from the slag, the reduction in this latter case being influenced by the temperature, the amount of carbon present in the bath, and the percentage of silica in the slag.

From the results obtained by melting and refining cold charges in an acid-lined three ton Girod furnace at Gutehoffnungshütte, the following conclusions have been drawn regarding the conditions which influence the reduction of silicon during the deoxidising period:—

(*a*) Provided that carbon is present in the bath the silicon reduction will be almost entirely derived from the acid hearth, and only to a lesser degree from the slag.

(*b*) The reduction of silica to silicon from the slag may be increased by raising its silica content.

(*c*) The reduction of silica to silicon from the slag may be considerably influenced by any excessive rise of temperature.

(*d*) The reduction of silica to silicon is considerably influenced by the amount of carbon in the bath, subject also to temperature.

The process practised at these works consisted of cold melting, followed by a careful refining. The first or oxidising slag is similar to acid open hearth slag containing a high percentage of mixed iron and manganese oxides. The charge after melting down is skimmed and fresh slag added consisting of about 75 per cent. crushed silica brick and 25 per cent. lime, to which is later added sufficient manganese oxide to give about 10-15 per cent. in the final slag; this practice of using a finishing slag low in metallic oxide is comparable to true liquid refining of steel transferred from some other steel furnace, and is quite distinct from the cruder method of melting and finishing cold scrap charges under one slag as described in Chapter VII. The bath of steel should contain rather less than the final required percentage of carbon before addition of the deoxidising slag.

The following analyses are given as typical of the deoxidising or finishing slags used at Gutehoffnungshütte:—

SiO_2	54·4	52·6	54·2	66·27
CaO	11·5	13·7	23·2	15·26
MgO	5·2	3·2	3·05	2·03
Al_2O_3	1·72	1·52	1·86	·15
FeO	3·35	3·85	4·05	2·81
MnO	23·55	23·7	11·15	11·68
S	·29	·13	·4	·51
P_2O_5	·03	·12	—	—

The advantages of acid refining lie in the rapidity of deoxidation, reduction of lining repair costs due to the cheaper price of acid materials used, and the prolonged life of the silica roof. The power consumption is also less than for basic refining, being about 100 kw.-hours per ton for a 15-ton furnace, the period of refining being about 1½ hours.

It has been generally admitted that deoxidation, promoted under an acid slag by actual silicon reduction, is far more complete and produces better results than when done in the more rapid and cruder manner by additions of ferro-silicon. Deoxidation in the acid furnace is comparable to the "killing" action or "dead melting" associated with crucible steel manufacture. Thallner's theories, which are based upon the initial production of silicon carbide, might reasonably be applied to the crucible, but he points out that the "killing reaction cannot be so complete in the latter case owing to the much lower silica content of the clay material".

The silicon-carbide theory is not generally accepted, and it is difficult to reconcile it with the fact that the minimum temperature of silicon carbide formation is not less than 1800° C., a temperature that can only be reached in the arc zones.

The theory generally favoured depends rather upon the alternate formation and dissociation of ferro-silicon due to the combined interaction between silica, carbon, and iron oxide. It is probable also that metallic iron itself, when in contact with a highly siliceous slag and in the presence of carbon, may promote the reduction and absorption of silicon at temperatures below the reduction temperature of silicon from pure silica. In either case the silicon would not be reduced in its elemental form, but as a compound of iron and silicon. The silicon thus entering a bath of steel will immediately reduce any metallic oxides in solution, and in this manner cleanse the bath of these impurities.

Acid refining based upon the above theory has been largely practised for the deoxidation of basic open hearth steel. Liquid steel, carburised to within a few points of the specification figure, is transferred to the electric furnace, and fluxes consisting of iron ore, lime, and sand are charged; the greater proportion of the silica, however, is derived from the fettling material, which becomes detached from the banks and hearth. The slag becomes bluish when the steel is hot, and the silicon in the bath should not be above ·05 to ·08 per cent. After the bath becomes deoxidised the silicon content will rapidly rise, especially if the steel should be very hot, and for this reason the charge should be tapped without delay. This method of refining enables any class of carbon or alloy steel to be made, which is perfectly deoxidised and free from segregation in the ingot.

Apart from the chemical effect of liquid refining, the improvement due to purely physical reasons must not be disregarded. It has been stated elsewhere that considerable importance is now attached to allowing finished liquid steel to remain in a perfectly quiescent condition for some time previous to casting, this being done for the sole purpose of allowing finely suspended slag, or other foreign matter of low specific gravity and gases, to rise and pass out of the steel. The deoxidation of steel in electric furnaces is not accompanied by any commotion due to evolution of carbon monoxide, so that during the process of refining the bath is also in a condition physically suitable for the free separation of minute slag particles.

Scope and Application of Liquid Refining.—Liquid refining in electric furnaces frequently constitutes the final stage of what are commonly known as Duplex and Triplex processes. These processes, as their names imply, embrace two or three distinct operations for the manufacture of finished steel, each operation being conducted in separate furnaces. When basic lined electric furnaces are used, the liquid steel may be simply crude blown metal produced by the acid or basic bessemer process; in either case the blown steel may possibly require further dephosphorising, which can be done either in the basic electric furnace prior to deoxidation and desulphurisation, or in some other furnace before transfer to the electric. In the former case the process is Duplex, and in the latter case Triplex.

The Duplex or Triplex process is used for the production of high class steel from unrefined liquid steel, such as is made by the basic bessemer process in Europe or the acid bessemer in America. In both cases the liquid product may be electrically refined to produce a steel equal in quality to that made by the more general method of melting and refining scrap. The basic open hearth furnace has also been used for producing cheap liquid dephosphorised steel, requiring subsequent deoxidation and desulphurising only in the basic electric furnace.

The application of electric furnaces to liquid refining is essentially suited to large outputs and rapid operation. The electric furnace should always be operated at a high load factor, and to render this possible a regular and frequent supply of liquid steel must be provided for. The bessemer process fulfils these conditions, and it is in conjunction with this method of steel-making that the electric furnace, as used for liquid refining, has been most generally applied. Tilting basic open-hearth furnaces working a continuous process, such as the Talbot, are equally, if not more suitable than bessemer converters, since the phosphorus can be sufficiently reduced to meet any acid open-hearth carbon or alloy steel specification, which thus shortens the period of subsequent refining in the electric furnace.

From technical standpoints a Duplex or Triplex process, which embodies a final refining treatment of semi-finished steel by the electric process, is perfectly feasible, as has been conclusively demonstrated both in America and Germany. The possibilities of liquid refining must be studied rather from an economic standpoint, and in this direction they will be dependent upon the following factors:—

I. The production of cheap liquid steel which can be enhanced in value by further refining.

II. A frequent and regular supply of liquid steel to the electric furnace, so that the load factor and output may be raised to a maximum, and all overhead charges correspondingly reduced.

III. A suitable market for the electrically refined steel, which in the case of large outputs must be able to compete favourably with the higher grades of bessemer and open-hearth steels.

CHAPTER IX.

INGOT CASTING.

Theory of Ingot Formation.—The art of steel-making, as applied to the manufacture of ingots, has for its ultimate object the production of steel in a crude form that will submit to subsequent physical or mechanical treatments, such as forging, rolling, machining, and heat treatment, without exhibiting or developing any structural defect. Liquid steel of excellent quality may be rendered quite unsuitable for the purpose intended by improper methods of casting and handling, and it is, therefore, of the utmost importance to adopt a method of casting that is satisfactory for each particular class of steel and shape of ingot. For instance, the existence of a long pipe may be harmless for one variety of steel, but might entirely ruin an ingot of another variety having imperfect welding properties. The solidification of steel in a cast-iron mould has been the subject of considerable investigation and discussion of recent years. More and more attention is being paid every day to this particular branch of the art of steel-making, and even now the theories advanced by those who have long and carefully studied the matter are by no means unanimous in all respects.

Iron is an element that possesses a definite crystalline form, so that solidification of steel first proceeds by crystallisation of the elementary iron from the molten metal, which contains other elements in solution. It is first necessary to study the process of solidification of a body of liquid steel under different thermal conditions, as this has a very important bearing upon the crystalline structure and chemical constitution of the solidified mass. Solidification naturally takes place wherever the temperature has fallen to the freezing-point of the steel, and may consequently be marked by an isothermal zone, which must progressively travel in an inward direction from all boundary surfaces exposed to the influence of cooling. The rate at which such a zone travels at any moment is, of course, a measure of the speed of actual solidification, and depends upon the temperature of the liquid steel and the rate at which heat is being abstracted.

Should the liquid steel be at a temperature well above its freezing-point and subject to rapid cooling, then the isothermal zone of solidification will, at any moment, be sharply defined and localised at the plane of junction between either the liquid steel and the walls of the mould, or the liquid steel and an already solidified envelope, owing to the abrupt temperature difference between the two in either case. Solidification under these thermal conditions will proceed by the constant deposition of thin films, or, in other words, by the slow and steady inward growth of the solidified envelope when once formed. This mode of solidification is therefore favoured by (*a*) a high initial casting temperature, (*b*) a rapid abstraction of heat.

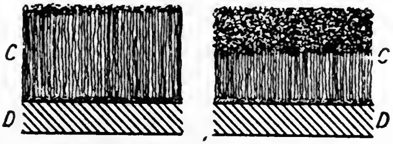
Now, even supposing solidification to have been proceeding in the above manner, the mean temperature of the still liquid steel will have been steadily falling by conduction of heat outwardly to the surrounding walls, and might, in fact, reach a temperature near to its freezing-point before the process of solidification had proceeded very far. In that event there will be little or no temperature difference between the surface of the truly solid and liquid portions, and the isothermal zone, instead of being sharply defined as before, will become obliterated and merge into the liquid steel. Solidification will not then proceed in a distinct and well-defined manner, but will take place more or less irregularly in a zone of much greater depth.

Again, if steel is cast into a mould having a very low conductivity and small thermal capacity, it will not be subject to sudden chilling at its boundary surfaces, and will fall in temperature slowly as a whole, until such a point is reached when a rapid irregular solidification, as above described, will result from any further lowering of temperature; this case applies whether the steel be cast hot or cold. Solidification may also proceed in this manner when steel, at a temperature only slightly above its freezing-point, is cast into a mould of good conductivity and high thermal capacity; this case is analogous to that previously described, where rapid solidification in irregular zones followed well-defined progressive solidification. Those thermal conditions, then, which favour rapid irregular solidification are (*a*) a low initial casting temperature, (*b*) slow abstraction of heat. Either of the above conditions, however, if sufficiently pronounced, will promote this mode of solidification, irrespective of the other.

A careful distinction must be drawn between the total time of solidification and the actual rate at any moment, otherwise those thermal conditions, which have been mentioned as conducive to the two distinct modes of solidification, would appear to be quite erroneous and contrary to fact. For example, suppose equal weights of steel at the same temperature to be poured into an iron mould of large heat capacity and high conductivity, and a sand mould of similar dimensions having a very small heat capacity and low thermal conductivity. Obviously the actual rate of solidification in the iron mould will be rapid at first and gradually slow down until the mean temperature of the still liquid steel has fallen by conduction almost to its freezing-point, after which it becomes more rapid and general, as previously explained. In the case of the sand mould, solidification will be delayed owing to the low thermal conductivity of the material, and will hardly begin to proceed until the temperature of the liquid steel as a whole has more nearly approached its freezing-point; solidification will then proceed rapidly throughout an ill-defined inwardly progressing zone. The total time elapsed between the moments of pouring and complete solidification will, of course, be considerably less in the case of the iron mould, although the bulk of the steel will have changed from the liquid to solid state at a slower rate.

It has been already stated that solidification proceeds by the crystallisation of pure iron from the liquid steel, and that the process of crystallisation is itself influenced according to the manner of solidification at any moment. Having dealt with the various thermal conditions which influence the actual rate of change of state from liquid to solid, and the zone in which it occurs, it is now possible to see how these same conditions affect the manner in which this change occurs or, in other words, the crystalline structure of the solid steel. Referring to Fig. 82, assume a mass of hot liquid steel C to be poured on to a heavy iron chill plate D of indefinite area, and, taking a hypothetical case, assume that there is no heat loss from the upper surface of the liquid steel. The thermal conditions assumed are such as to promote solidification in well-defined zones. The rate at which heat is abstracted from the molten steel is variable; in the first instance, there is a rapid withdrawal of heat by the chill plate D dependent upon its thickness and, therefore, heat capacity, the exchange of heat being rapid by virtue of the high thermal conductivity of the cast-iron, which diffuses the heat throughout its mass. There must be, however, a gradual falling off in the rate of heat withdrawal as the chill plate becomes hotter and finally assumes a temperature of equilibrium, which occurs when the gain of heat from the steel by conduction is equal to the loss of heat by radiation. A thin layer of steel immediately next to the surface of the plate is subject to an intense chilling effect, and solidifies very rapidly with the formation of minute crystals, giving rise to a very close-grained crystalline structure. The thin layer of solid steel thus formed slightly lowers the rate at which heat is abstracted, and then allows solidification to proceed in well-defined isothermal zones, which favours a more regular and perfect growth of crystals. Since the heat travels from the steel to the chill plate in a definite direction, it follows that the crystals must grow regularly in the opposite direction, always presenting their uppermost end to the still liquid steel; this gives rise to a "needle"-like or columnar structure, as shown in Fig. 82. Crystals which grow in this manner will have a well-defined relative orientation, and are known as "chill" crystals. An ingot exhibiting this structure is often said to be "scorched," as it only results when the steel is cast very hot. While solidification proceeds in this way, the liquid steel is losing heat, and, should its mean temperature approach its freezing-point, solidification will then become irregular. If this occurs, crystallisation will not take place by the steady growth of the solidified surface, but may proceed by the formation of individual crystal grains within an ill-defined zone and remote from this surface. The crystallisation will then be irregular and the solidified steel will be built up of crystal grains having no fixed orientation relative to one another. The columnar structure will give place to a granular structure consisting of so-called "equiaxed" crystals; the proportion of "chill" to "equiaxed" crystals will naturally depend upon the relation between the speed of solidification, and the rate at which the still liquid steel loses heat by conduction. Slow heat abstraction will retard solidification and favour a gradual fall of temperature, and the proportion of "chill" to "equiaxed" crystals will become less, as indicated by the two sketches shown.

The above theory may now be applied to demonstrate the crystalline character of a steel ingot cast in a square open-top ingot mould, standing on a heavy cast-iron bottom or chill plate. It has been explained how crystallisation proceeds uniformly in a direction at right angles to a chilling surface, so that, after the initial freezing of the envelope, the steel will solidify in successive layers parallel to one another, provided the rate of heat abstraction is everywhere uniform. If casting and cooling conditions are such as to favour the formation of "chill" crystals, it is evident that they will grow inwards from the four vertical sides and bottom of the mould; the crystals growing from any two adjacent chilling surfaces will be at right angles, and meet obliquely in a plane which lies at an angle of 45° to each surface, if the rate of growth should be uniform from both chill faces. Four such planes will also be formed inclining upwards from the bottom edges of the mould, and their lines of intersection will form a pyramid or truncated pyramid, according to whether the "chill" crystals penetrate to the centre of the ingot or not. In the case of small ingots, the growth of the chill crystals may be so rapid that they will penetrate to the heart of the ingot before the temperature of the steel at any time remaining fluid has fallen sufficiently low to admit of the growth of equiaxed crystals; this only occurs when the steel is cast very hot. The photograph of a broken



Rapid Heat Abstraction. Slow Heat Abstraction

Fig. 82.

ingot shown in Fig. 83 indicates the three distinct forms of crystallisation illustrated in Fig. 82.

If the walls of a mould vary in thickness, the rate of crystallisation will depend upon the heat-absorbing capacity at any section, and, by increasing the thickness of metal, the conditions which favour the growth of chill crystals are intensified. Molten steel, adjacent to the corner of an ingot mould having sides of uniform thickness at any horizontal section, is likewise subjected to more intense chilling than at points intermediate between two corners, so that chill crystals are more developed near the corners of an ingot. It has been well established that the planes marking the junction of chill crystals are planes of weakness, a fact that is demonstrated by the presence of longitudinal corner cracks, which sometimes occur in defective ingots.

The solidification of steel, irrespective of the manner of crystallisation, is accompanied by a shrinkage, which must not be confused with the contraction shrinkage that follows later. The volume of liquid steel filling a mould is greater than the volume occupied by the solid steel, so that, unless solidification is accompanied by a depression of the liquid level to compensate for the difference in the two volumes, cavities will of necessity be formed in the ingot, and, moreover, at that point where solidification is finally completed. The shape of the zone marking the surface of demarcation between liquid and solid during the process of solidification, together with the corresponding shape of the shrinkage cavity, may cause defects in ingots that are, however, capable of being mitigated, if not almost prevented. Contraction cavities and gas cavities may also be formed under certain conditions, and give rise to defects which may only become apparent during the later stages of mechanical and physical treatment. Segregation of impurities, which only occurs to a minor extent in highly refined electric steel, must not be disregarded. The relationship between solidification, or more properly the mode of crystallisation promoted, and the chemical constitution of an ingot is dealt with later under the subject of segregation.

Before describing actual methods of ingot casting, which will differ according to whether the mould is filled from the

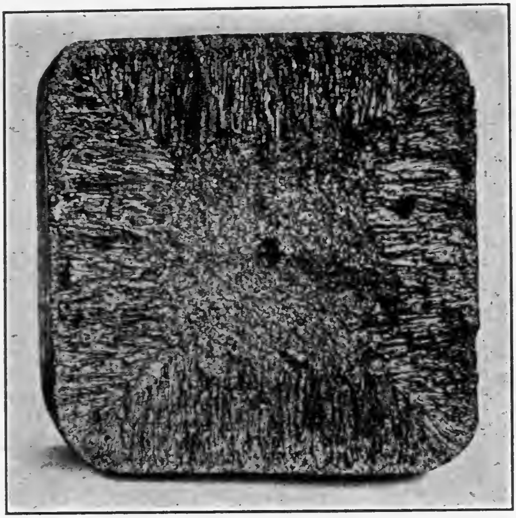


FIG. 83.

[*To face p.* 170.

top or bottom, the design of the moulds and the use of certain special apparatus, it is more convenient to examine the character of the commoner ingot defects and the particular casting conditions responsible for their formation. In this way the merits of the various methods of casting may be better judged in so far as they avoid or minimise these harmful conditions.

Ingot Defects.—Piping.—In the case of steel cast into chill moulds, the change of state from liquid to solid, which is accompanied by shrinkage, has been briefly considered. Without using special precautions, the uppermost layer of molten steel in an ingot mould will become solid before solidification of the interior is complete, and since the crust so formed completes the solid envelope surrounding the still liquid portion, it follows that further shrinkage cannot be followed by a corresponding self-adjustment of the envelope, with the result that a cavity or series of cavities are formed in the interior. The shape of the main cavity depends upon the manner in which solidification has proceeded, and this again is influenced by the taper of the mould walls, the direction of taper of the mould itself, and the position occupied by the last portion of steel entering the mould. Such a cavity is generally called a "pipe," a term which is more literally descriptive of its character when it occurs as an elongated, inverted cone, with its base close to the ingot top. To study the formation of pipes it will be easiest to consider the process of solidification of liquid steel when cast into an open top mould with parallel sides. Fig. 84 represents the solidification of an ingot in successive stages. While the mould is slowly filling, the upper walls are becoming heated by radiation from the stream of steel and from the rising column of steel in the mould; when the mould is full, the bottom of the ingot will have begun to solidify before the top has even begun to chill, so that the isothermal zone, which represents the surface of solidification, will, a few moments after teeming, be somewhat as shown in the sketch A. It is apparent that the body of steel immediately below the top crust formed will, on shrinking from it, be protected from further rapid loss of heat by radiation, and, since it was the last portion teemed, will tend to be the last portion to solidify and thus serve as a reservoir from which liquid steel is constantly drawn off as shrinkage proceeds until finally exhausted. The other sketches shown in Fig. 84 illustrate how the pipe is formed by the thickening of the ingot wall from below the crust downwards, accompanied at the same time by constant depression of the still liquid steel. When solidification is on the point of completion, the ingrowing walls of solid steel at the middle and lower end of the ingot may be almost parallel, and if these walls should meet at certain points between which liquid steel still remains, it is obvious that shrinkage of those isolated portions cannot be met by drawing off from the larger reservoir above. Therefore, at those points there will also be found long, narrow cavities F (see sketch D), which are known as "secondary pipes," and are entirely disconnected from the primary or main pipe E above. Frequently the main pipe is bridged across by one or more crusts of solid steel, which have formed at different levels, allowing the still molten steel below to recede and keep pace with shrinkage.

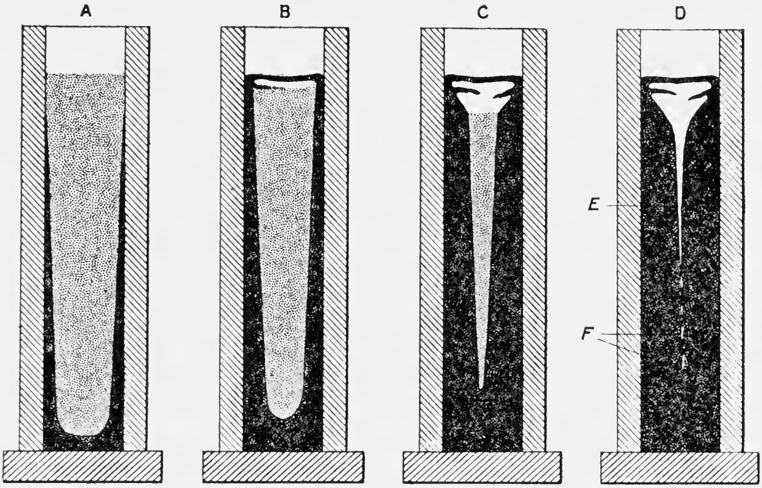


Fig. 84.—Solidification in parallel moulds.

The shape of the pipe formed may clearly be modified by either retarding or hastening the freezing of the steel at the upper end relatively to the lower. By retarding the rate of cooling at the top end, the zone of complete solidification from wall to wall will reach a higher position in the ingot before the reservoir of liquid steel has been finally exhausted; in this way the pipe may be considerably shortened, and will have greater lateral dimensions. It will be mentioned later how this may be effected in practice. By more rapid solidification of the upper ingot walls, the reservoir of liquid steel will rapidly diminish in volume before solidification of the lower portion of the ingot is complete; under such conditions, those portions of the ingot still remaining liquid will, on solidification, produce elongated shrinkage cavities, or "secondary pipes," in a far more marked degree.

In some cases, a solid crust may not be formed, and the pipe caused by the continual shrinkage of the steel will then be exposed to the air and become coated with oxide. Even supposing such an oxide coated pipe were capable of welding, the steel in the immediate vicinity of the weld would be partly decarburised and generally inferior to the rest of the billet. Pipes formed below solid crusts are free from a coating of oxide, but even then piped ingots of certain steels will not weld up perfectly during the forging or rolling operations. The presence and extent of piping are influenced by different methods of casting, to be dealt with later.

In certain heavy engineering work, where safety and reliability is of primary importance, the pipe is either removed by trepanning or by rejection of the upper portion of the ingot in which it is situated. Rejection of the top is only effective in the absence of secondary pipes.

Segregation.—Liquid steel may be regarded as a complex mixture of iron, carbon, silicon, manganese, phosphorus, sulphur, etc., in which the metalloids probably exist dissolved in the iron in a colloidal state. It is now generally accepted that, at the moment of incipient solidification, pure iron begins to crystallise out from the mother liquor in the form of dendrites, which may be regarded as minute, acicular or needle-shaped crystals. According to Stead these crystals shoot out branches at right angles corresponding to the axes of a cube, and the branches themselves undergo growth of crystallisation in like manner. The mother liquor, from which these so-called dendrites or crystallites grow, becomes enriched in sulphur and phosphorus, with a corresponding increase of fusibility. The bulk of this more fusible liquid becomes entrapped in the ever-multiplying crystal branches, which eventually become closely interlocked. If, however, sulphur and phosphorus are present in sufficiently large quantities in the mother liquor, they will form fusible compounds of low specific gravity, and these minute particles, owing to their fluidity, coalesce and are then either entrapped or pushed forward to that zone of the ingot where solidification last takes place. By virtue of their low specific gravity, the sulphur and phosphorus compounds will at the same time tend to rise through the mother liquor, and it is partly for this reason that drillings taken from an ingot in the neighbourhood of a primary pipe contain more sulphur and phosphorus than elsewhere. This local concentration of the impurities is called "segregation," and is favoured by high casting temperatures and rapid cooling. These latter conditions, it has been explained, promote the growth of "chill crystals" which entrap less mother liquor as crystallisation proceeds. The mother liquor, therefore, becomes more and more enriched in impurities, and should any fusible compounds separate out as segregates, they are pushed forward by the slowly advancing chill crystals. For this reason, as Brearley has pointed out, a ring of segregates is often found lying at the boundary of the chill and equiaxed crystal zones. Equiaxed crystals, on the other hand, which result from rapid irregular crystallisation, entrap the mother liquor *in situ*, and so prevent pronounced segregation. Carbon and manganese also segregate, but in both cases the percentage enrichment in the zones of segregation is very small compared with that of sulphur and phosphorus. Sulphur segregates in the form of a fusible manganese sulphide, so that sulphur and manganese enrichments are always coincident.

Segregation is usually regarded as a distinct fault, but in cases where the upper third of a large ingot is rejected on account of a pipe, or where the centre of an ingot is removed by trepanning, it then becomes a virtue to be encouraged.

Ghost Lines.—"Ghost" is the name given to a defect which only becomes apparent on machining. It appears as a whitish streak in the steel, always following the direction of elongation under forging or rolling treatment. The term "ghost" is applied to such flaws, because, owing to their usually extreme thinness, they can be removed with ease on machining; unfortunately, the presence of a ghost line, which may itself be easily removed, indicates the presence of others, which may be situated outside the range of mechanical removal and are a source of danger when the steel is under transverse stress. According to Stead, these white "ghost lines" are carbonless streaks of ferrite in which are usually embedded lenticular or drawn out inclusions of manganese sulphide, the presence of which also indicates the segregation of phosphorus in the same region. The magnitude of the ghost lines is more pronounced in the case of high phosphorus steels, and to this element he attaches the power of expelling carbon from the zone of segregation on slow cooling. Others consider that the primary existence of "ghost lines" is due to slag inclusions, which act as nuclei for the secondary crystallisation of ferrite, or for the deposition of hard cementite (Fe_3C), according to whether the percentage of carbon is below or above ·89 per cent., which is the percentage present in the saturated eutectoid or solid eutectic of iron and carbon. These slag particles are drawn out on forging, and cause repeated deposition of soft ferrite or hard iron-carbide on cooling from a temperature above the critical range. This theory also explains the persistence of "ghost lines" even after repeated annealing, unless the heat treatment is so prolonged as to cause an actual dispersal of the drawn out string of nuclei. Fig. 101 shows slag inclusions imbedded in soft ferrite areas in a sample of an annealed steel casting.

Lapping and Folding.—These terms apply to irregularities produced on the skin of an ingot whilst the level of liquid steel is rising in the mould. When an ingot is being top-poured, it sometimes happens, especially if the steel is teemed cold, that a pasty semi-solidified crust forms on the top of the molten steel and assumes a slightly convex form along the edges of contact with the mould; this convexity increases, and the molten steel is held from the sides of the mould until the pressure becomes great enough for the steel to burst through the pasty crust. The molten steel, being released, immediately flows round a portion of the edge of the crust and fills the gap between its convex surface and the side of the mould. It is improbable that the steel flowing over the convex edges of the oxidised crust ever properly welds to it; in fact, unless the crust has been momentarily remelted by the steel flowing upon it, the surfaces of contact are likely to become subsequently detached on forging or rolling. Even in the event of fusion, the crust will carry into the steel a quantity of oxide, which gives rise to local unsoundness. In top-poured ingots the lap or fold, which follows the line of contact between the edge of the crust and the steel flowing over it, is generally wavy and irregular in form. In the case of bottom-run ingots, the steel rises in the mould with less surface agitation and with uniform pressure, so that if a small annular crust should form, producing only a slight convexity where in contact with the mould, the hot steel on rising will flood the crust uniformly, so that any "laps" or "folds" exhibited on the skin will be close together and parallel. If, however, the steel is very cold and the crust that forms is thick, then the folds will be wavy as in the case of top-poured ingots; the horizontal laps cannot be produced in open top ingots of small section, owing to the commotion caused by the stream striking the steel as it fills the mould. The objection to fusion of the crust by the steel flooding it does not apply so strongly to closed top moulds, where oxidation of the semi-solid crust can only be very slight. Some alloy steels, particularly those containing chromium, are more liable to lap, owing to the readiness with which oxidation crusts form. Folding, then, results either from teeming cold steel or from teeming too slowly, which allows the steel to cool off and form semi-solidified crusts while filling the mould.

"Shell" or "Catch".—If steel on entering an open top mould should either strike the side or splash up against it from the bottom plate, it will be immediately chilled and form a thin, irregular shaped strip. This will either adhere to the side of the mould or be enveloped and remelted by the liquid steel. In the former case, as filling proceeds, the liquid steel will rise and chill against this strip without effecting a proper weld. The irregular shaped strip remains on the surface of the ingot as a "catch," and will sometimes emit a hollow sound when tapped, indicating imperfect adhesion to the body of the ingot. In the latter case the ingot is said to be "shelled," and will usually be rejected as scrap. Even if the "catch" is not very pronounced, it will cause local unsoundness and small cracks owing to unequal contraction of the strip and the liquid steel which is chilled on to it.

A "shell" or "catch" is sometimes caused when bottom-casting, owing to the steel becoming partly chilled in the runner bricks at the commencement of teeming. In such cases, the steel becomes pasty and at first resists the pressure of the liquid steel in the trumpet. As soon as the pressure is sufficient to overcome the resistance offered by the pasty steel, the latter is suddenly pushed forward into the mould and followed by a fountain of liquid steel, which may strike the side of the mould and leave a strip of solid steel adhering to it.

Pulling.—Steel, after solidification and while still at a high temperature, is incapable of resisting any great tensile stress. Cooling and contraction of an ingot must proceed simultaneously, and any tendency to resist this normal function will result in a "tear" or "pull". An ingot that becomes fastened to the top of a mould through flooding or some other cause will shrink away from the bottom and then be forced to carry its own weight; in such a case the tender walls may at some point be unable to carry the weight of the portion below and are so torn apart. Moulds that for any reason have been flooded should be immediately cleared from any "fash," so as to allow free contraction of the ingot from the top downwards. Some types of "dozzles" and such other rigid devices, if too firmly fixed to the mould, will sometimes hold the ingot head and cause pulling, while the same thing applies to "scabbed" moulds or any other cause which prevents free contraction.

Pitting.—A good ingot is often spoiled by an inferior skin, which may give rise to small "rokes" during the later stages of forging. It is not uncommon to find the skin of an ingot studded all over with small pit marks; these small circular depressions are caused by evolution of gas at the moment of contact between the steel and the mould, resulting either from an excessively high casting temperature, or from iron oxide or moisture on the surface of the mould. Moulds reeked with tar, which has not been vapourised before teeming, will also produce the same result. Pitting is not a serious defect, except in very small ingots, and can be avoided by careful attention to casting temperature and the use of clean and properly reeked moulds.

Clinks.—A cold ingot, as it comes from the casting pit, is always under stress which is not equally distributed. In the case of certain steels, especially those belonging to the air-hardening class, the ingots should be re-heated very slowly in order to remove these unequal stresses. If the heating is conducted too rapidly, the ingot may develop deep seated cracks, which is sometimes accompanied by an audible report. Small axial cavities, which may constitute a secondary pipe or be due to internal contraction flaws, may often serve as starting points for such internal cracks.

Contraction Cavities.—Messrs. Brearley, in a paper read before the Iron and Steel Institute in 1916, advanced an ingenious explanation for the formation of contraction cavities. Contraction will follow immediately after solidification, and is therefore irregular in different parts of an ingot; the outer envelope cools first and can contract on to the liquid centre, until finally it becomes sufficiently rigid to resist distortion by forces exerted upon it from within. When solidification is complete in the centre of the ingot, the outer skin will resist the deformation necessary to reduce the volume in accordance with the internal contraction that follows. Since, then, the central portion of an ingot must contract, it can only do so by tearing apart along a central axis, where the steel is least able to resist tensile stress, and this results in the formation of contraction cavities. This process of contraction may actually proceed in those portions that have just become solid before solidification of the entire ingot is complete. From this it follows that contraction cavities will also be situated at other points, particularly where the steel is least able to resist internal stresses, and they are actually found lying in the diagonal planes of weakness which mark the junction of chill crystals growing inwards from two adjacent faces of a mould.

Surface Cracks.—Skin cracks, which are frequently a source of considerable trouble, may be caused in several ways. Transverse cracks or tears, produced by "pulling," are generally very pronounced and always occur at right angles to the direction of pull; there is no difficulty in recognising this type of crack, and the cause, when not at once obvious, is never difficult to find. Surface cracks, which at times can only be discovered by careful examination, may result either from irregular contraction of the ingot just after solidification, or from the internal pressure exerted by the still liquid portion of an ingot on the thin tender wall.

The formation of cracks due to fluid pressure is promoted by a high casting temperature and rapid teeming, as under such conditions the ingot walls towards the bottom may not have attained sufficient thickness to withstand the pressure due to the column of liquid steel above. The thin and tender ingot walls begin to contract immediately after their solidification, so that an exceedingly thin gap will be left between the walls of the mould and the newly forming ingot. Internal pressure may then cause rupture of the ingot skin, as evidenced by irregular surface cracks, which assume a more or less vertical direction.

Cracks caused by irregular contraction of the solidified steel are due in some measure to the un-uniform chilling effect of a mould. In square moulds the chilling effect is far greater in the corners than along the sides, and in many designs of moulds the sides are cast thicker towards their centre to counteract this effect. Contraction cracks will tend to develop along any natural planes of weakness, and it is not infrequent to find longitudinal corner cracks which follow the diagonal plane marking the junction of chill crystals. High casting temperature and rapid teeming promote this latter mode of crystallisation and, under these conditions of ingot pouring, surface cracks are almost certain to be produced as a result both of internal pressure and irregular contraction.

It does not follow that ingots which show no apparent sign of cracks will forge perfectly, and the "dryness" in this case may probably be due to the extension of small internal contraction cracks formed in the ingot while cooling, especially in the neighbourhood of any weak spots, such as contraction or shrinkage cavities. In many cases cracks that are superficial in the ingot will open out and extend inwards on forging. The defective portion can be removed from the cogged bloom by chipping or grinding, which, however, adds considerably to the cost of working down. Frequently a perfectly sound ingot may be rendered defective under the hammer or press by improper reheating, or by forging at an unsuitable temperature, so that before condemning the original ingot it is important to ascertain whether it has been submitted to proper treatment.

Blowholes.—The presence of blowholes, distributed irregularly in the body of an ingot, is usually due to the condition of the steel when teeming rather than to improper methods of casting. Blowholes are usually formed as a result of chemical reaction between dissolved carbon and iron oxide, which may be either present in the steel before casting, or may be formed during transfer from the furnace to the mould by pouring over a damp spout, or by using an imperfectly dried ladle or bottom pouring trumpet. Steels which are low in silicon are more liable to be rendered "wild" by such contact with moisture. The evolution of gas, which occurs internally, is not observed until after solidification has begun. The oxide being in a state of dilution cannot react with the carbon in the molten steel, but, on separation of the ferrite crystals at the moment of crystallisation, the carbon and dissolved oxides are brought into closer contact by their resultant concentration, and then react with generation of carbon monoxide. Other theories are advanced which account for the delayed evolution of gas on the assumption that the reaction between the carbides and oxides only takes place at less elevated temperatures.

In certain cases blowholes may be caused by the evolution of dissolved gases, which follows as a result of their reduced degree of solubility at lower temperatures. Excessive overheating of steel in the furnace will often give rise to wildness which is sometimes impossible to eliminate by an increased addition of silicon and manganese; it is difficult to explain this behaviour, which, however, has its analogy in acid bessemer practice, where hot blows frequently result in "wild heats" unusually high in silicon. For this reason careful control of temperature during the finishing period of a heat is necessary, and this is sometimes by no means an easy matter with small furnaces having a large power capacity.

Occluded Gases.—For many years past it has been thought that the presence of gases, dissolved or occluded in steel, has considerable influence on its physical properties. The actual determination of the quantity of gas dissolved and its chemical composition requires most elaborate methods of sampling and analysis, and even now no definite rules governing the action of occluded gas have been established. At one time the addition of aluminium was favoured, owing to its supposed power of increasing the solubility of gas in steel, but it is more likely that the elimination of wildness is in this case primarily due to the removal of dissolved oxides and consequent prevention of their subsequent reaction with carbon accompanied by generation of gas. There is now little doubt that the mere act of holding molten steel in a tranquil condition prior to teeming does have a beneficial effect, which is generally accounted for by the opportunity afforded of expelling dissolved gas, and of allowing suspended slag particles to rise and so pass out of the steel. The electric furnace serves as an ideal receptacle for applying such treatment, which proceeds simultaneously with the final chemical refining stages of both the acid and basic electric processes.

Ladles, their Use and Manipulation.—Ladles may be divided into two classes: (*a*) Bottom-teeming, (*b*) Lip-pouring.

The former type is universally adopted for ingot casting, and in foundry practice when basic-lined furnaces are used. The second type is only used for foundry casting, and then only when it is an easy matter to remove the bulk of the slag from the ladle and hold back the remainder while pouring, this being only feasible in the case of sticky acid slags.

Bottom-Teeming Ladles.—There are several types of bottom-teeming ladles, which only differ slightly in mechanical design. The tilting ladle (Fig. 85) consists of a slightly conical steel vessel A, built of mild steel plates and mounted on trunnions B, one of which carries a worm wheel C in gear with a worm D. The worm is fastened to one of two heavy suspension rods E connected together by a yoke piece F, and can be driven through bevel gearing by means of a light hand wheel, which enables the ladle to be turned through a complete circle. Two small brackets G and H serve as guides for a sliding bar I, which can be raised or lowered by means of a lever, or held in a fixed position by a set screw. This slide terminates in a head-piece machined over its upper surface to carry a cross-bar J, the head is also provided with a screwed pin or cotter bolt, which

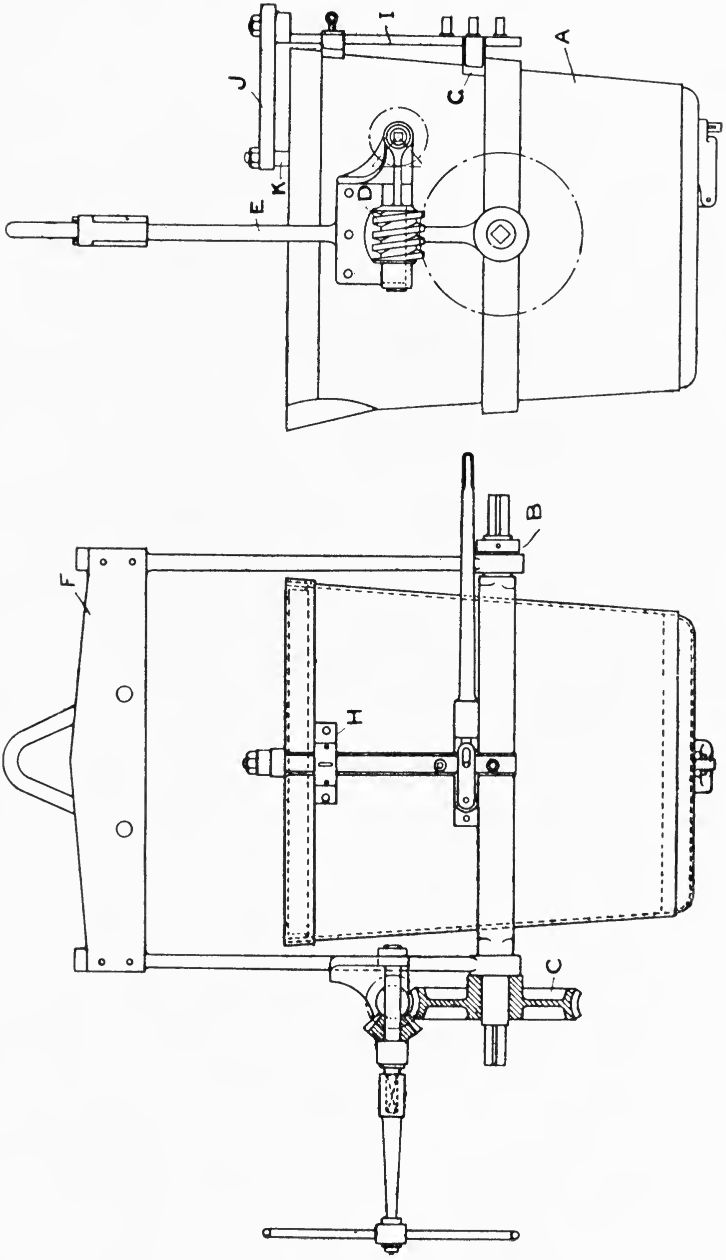


FIG. 85.

enables the cross-bar to be firmly secured to the slide. The outer end of the cross-bar is slotted to receive the upper end of the stopper rod K, which in this case is provided with a collar and screwed pin or cotter bolt; the stopper rod can then be firmly fastened to the cross-bar and so respond to any vertical movement of the slide. This method of attaching the cross-bar to both the slide and the stopper rod allows for lateral adjustment of the latter when being set so as to close the nozzle correctly. The lower end of the stopper rod (Fig. 86), if made from a solid bar, is drilled or tapped to receive the stopper pin; in the former case, the pin is slotted and attached to the stopper rod by a cotter. A hole, through which the nozzle brick passes, is cut in the bottom plate of the ladle, and concentric with it but on the under side is fixed the nozzle box, which supports the nozzle brick and fixes its position.

The walls of the ladle are lined from the bottom upwards with special shaped fire-bricks, ganister or damp fire-clay being rammed to fill the narrow gap caused by rivet and bolt heads. Ladle "compo" or ganister is sometimes rammed to form the entire lining. The bottom plate is covered with a course of fire-brick, or by a bed of ganister rammed solid; in either case a hole is left a little larger in diameter than the nozzle brick to be used. After lining, the ladle should be strongly fired for twenty-four hours until all moisture has been entirely expelled; failure to do this properly may lead to the loss of a cast of steel. The stopper rod is protected by a covering of sleeve bricks which are carefully fitted together, while the stopper end (Fig. 86) is firmly attached to the stopper rod by a stopper pin. Should a slotted pin and cotter be used for this purpose, a rigid connection can only be secured by interposing thin washers between the stopper brick and stopper rod, and by selecting a suitable cotter from an assortment of slightly varying widths. In the case of stopper rods having a shoulder at the upper end, the necessary number of sleeves must be threaded on to it before

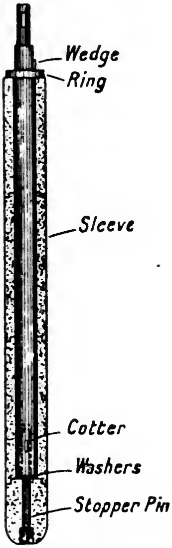


FIG. 86.

fastening the stopper end brick. The stopper end and sleeves are all mortised to prevent steel finding its way through the joints, which are made as close as possible with a little thin fire-clay. As each sleeve is joined to its neighbour below, the space surrounding the iron rod is filled with sand to prevent any lateral movement after building up. The sleeves are usually made fast by driving a small wedge between the rod and a loose washer ring resting on the top sleeve brick. Stopper rods when finished are sometimes given a light wash over with graphite at the lower end and then slowly but very thoroughly dried; any moisture in the sleeve bricks may cause them to burst when suddenly heated.

The correct setting of a stopper rod before pouring is an operation that calls for considerable care and skill, and is usually only entrusted to men who have had considerable experience as helpers, and who have seen the operation performed hundreds of times before attempting it themselves. The nozzle brick, after being placed in position, is firmly fixed by ramming loam or ganister into the space round it, so as to make it one solid piece with the rest of the bottom lining. The ladle is then warmed before setting the stopper rod. The stopper rod is connected to the cross-bar, and adjusted in the manner mentioned until the stopper end finds a perfect seating on the nozzle brick when lowered. Both the nozzle brick and stopper end are convex at their point of contact, but owing to surface irregularity and slight distortion of shape, seldom meet exactly to form a perfect joint. The stopper end is generally ground to fit the nozzle before fastening it to the stopper rod, this being done by grinding the two surfaces together with a little fine sand before fixing them into their respective positions. This practice of grinding in with sand does not find universal favour, as it is argued by some that removal of the hard skin of the fire-brick is liable to cause trouble after beginning to teem. After finally setting and fastening a stopper rod, the seating is tested by throwing a little fine, dry, white sand round the junction of the nozzle and stopper brick; if, on lightly tapping, no sand leaks through, the stopper rod is considered sufficiently well set. Sometimes ladlemen purposely set their stopper rods a fraction of an inch out of centre towards the slide bar, so that the stopper end, on being lowered, just strikes the near side of the nozzle and then slides into the correct position.

Ladles are always heated before receiving a cast of steel, to prevent the steel chilling and stopper troubles that might otherwise result. Sometimes the ladle is warmed by lighting a small coal fire on the bottom after setting the nozzle brick in position, care being taken to cover the nozzle brick with a small bent plate to prevent any ash from adhering to the ground seating; this method is good enough for ordinary purposes, but there is far less risk of skulling and "hard stoppers" if some system of heating by air blast is adopted. Two methods are commonly used; in one, the ladle is inverted over a shallow open top fire-box built in solid masonry (Fig. 87); in the other a small fire is made on the bottom of the ladle, and a blast of air blown downwards upon it from a blast pipe that can be readily removed (Fig. 88). The blast pipe A is constructed in the form of an inverted U, and is suspended from a light jib that swings about the axis of one limb. This limb fits loosely in a fixed blast pipe B in which it is free to slide up and down. The ladle, containing a small coke or coal fire, is placed close to the jib, and the latter is then swung round and the blast pipe lowered, so that the nozzle end is centrally situated in the

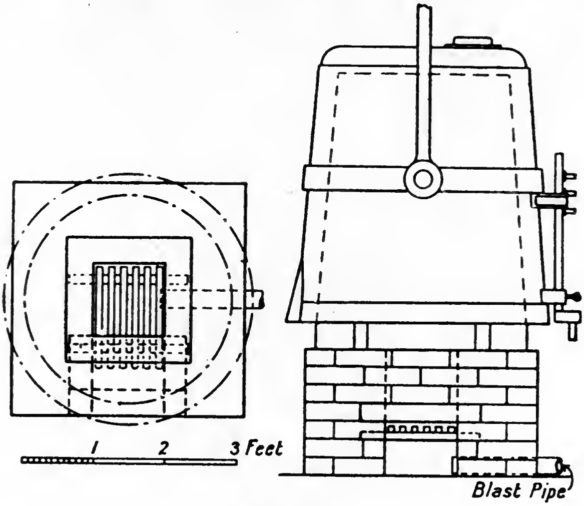


FIG. 87.

ladle. This arrangement is very convenient as no extra handling of the ladle is required, and the heat is produced on the bottom where it is most wanted.

Methods of Ingot Casting.—There are two entirely distinct methods of casting ingots, depending upon whether the steel enters the mould from the top or from the bottom. The former is known as "top-casting," and the latter as "bottom-casting" or "bottom-running". In either case tapered moulds may be used with their larger section at the top or at the bottom, as may be preferred. Before, however, describing these different methods of casting, the various types of moulds used must be considered.

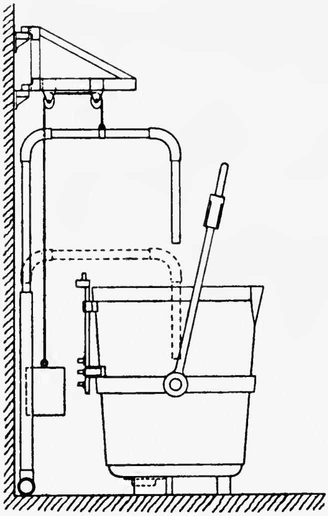


Fig. 88.

Ingot Moulds.—Ingot moulds are of three kinds (*a*) open top and open bottom; (*b*) open top and closed bottom; (*c*) open bottom and closed top.

Moulds should be made of a high grade grey hæmatite iron of coarse open grain and high in silicon and graphitic carbon, as the composition of the iron used has a marked influence on their life. An ingot mould is always subjected to extreme changes of temperature, more especially when teeming, and, since the heat is only applied to the inner walls, the stresses set up by unequal expansion are considerable; rough treatment in handling also demands a high degree of toughness, so that the use of a No. 1 or No. 2 grade pig-iron is essential for the production of a good mould which will withstand the severe conditions of service. Moulds up to 20 inches may be square, but beyond that size are either hexagonal or octagonal in form; the latter shapes favour a more even rate of chilling, which equalises to some extent the internal stresses set up in the outer envelope of the ingot. The inner walls of a hexagonal or octagonal mould are slightly convex, which allows for slight deformation of the ingot on cooling and lessens the possibility of cracks.

The degree of taper and thickness of moulds are subject to variation according to the particular views of different users. The rate of chilling or speed of initial solidification, is dependent upon the thickness of the walls and, therefore, the capacity of the mould for absorbing heat from the liquid steel, so that the dimensions of the mould will influence the manner of crystallisation and also the position of any pipe. The taper of an inverted mould with the larger end uppermost is often increased so as to amplify those conditions which are favourable to the reduction of "pipes" or their localisation at the top of the ingot. Closed and open-top moulds used small end up should, on the other hand, have only sufficient taper for the purpose of stripping, so as to prevent as far as possible the formation of deep-seated pipes.

New moulds often have a thin skin of oxide or dirt on the inside faces, and until they have received a few casts of steel the skin of the ingot will be somewhat impaired by pitting. The interior of the mould should always be carefully cleaned with a wire scratch brush, and all scale or rust removed from those which have been in disuse. Cleaning alone is not sufficient for the production of a perfect ingot skin, it being further necessary to "reek" the inner faces of the mould. The term "reek" was originally used to denote the practice of depositing soot on the inner surfaces of small crucible ingot moulds by exposing them to the smoky flame of burning coal tar; now it is used in the wider sense of covering or painting over the faces with some material which leaves behind a deposit of carbon, either on drying or heating. The older method of smoking is the best whenever it can be used conveniently, but for large moulds it becomes impracticable. Two methods of "reeking" are commonly adopted:—

(*a*) The mould is brushed over with a thin plumbago wash while it is warm enough to drive off the water. This method is open to the objection that, unless the wash is quite thin, clots of plumbago will be left on the mould and produce surface irregularities and local unsoundness of the ingot.

(*b*) The mould is painted over with a thin coating of tar, which on vaporising leaves a thin deposit of carbon adhering to the surfaces so coated. Sometimes the tar is not boiled off before teeming, in which case there is a sudden evolution of gas at the moment the liquid steel comes in contact with it; at the same time the mouth of the mould will be filled with smoke which prevents the teeming operation being watched. When tar is used for reeking it is better to clean the mould while still hot from a previous cast, and then to paint it over so that all the volatile matter may be driven off before it is again required for use.

Ingot Pit.—Moulds are usually set in a specially prepared pit, so that their upper ends are at a convenient height above the shop floor level for teeming. Sometimes the moulds are set on the floor, in which case it is necessary to provide a platform from which the ladleman can manipulate the stopper rod lever. Ingot pits are preferable from every point of view, and should always be provided wherever possible. When moulds are set on the floor level there is a far greater risk of injury to workmen in the event of a running stopper or a break out at the bottom of a mould or trumpet, besides which the manipulation of the ladle in the event of a hard stopper is rendered far more difficult and dangerous. Ingot pits are usually lined with fire-brick, and the walls slightly tapered and crowned with a cast-iron curb plate to facilitate the withdrawal of any steel skull that may have accidentally covered the bottom from side to side.

Top-Casting.—The method of filling an ingot mould from the top is by far the most commonly adopted, and is almost invariably used for casting ingots heavier than about 30 cwts. Sometimes split moulds are used which have no longitudinal taper and form a closed bottom. Moulds that are not split must be tapered longitudinally to ensure ease of stripping, and are used with either the small or big end uppermost. Ingots that have been cast from the same mould but in reversed positions will not have similar internal structures, so that when selecting a method of casting, not only should the relative ease of handling the moulds be considered, but also the probable position and effect of the pipe produced.

As regards handling, it is clear that moulds set small end up will have to be stripped from off the ingots, and then reset in position for a subsequent cast; this does not apply in the case of ingots cast large end up, which can be removed from the moulds without disturbing their position, provided small wrought-iron eyes are fixed in the head of the ingots just before setting. Apart from the question of handling, the direction of taper exerts a considerable influence on the position and magnitude of a pipe, which may be explained by reference to Figs. 89 and 90. Fig. 89, which shows the section through an ingot cast small end up, indicates a pipe extending half way down together with a small secondary pipe. On studying the progress of solidification in such a mould it is clearly apparent that the chilling effect at the head of the mould is greater than at lower sections owing to its contracted area, the rapid loss of heat from the liquid steel exposed, and the sometimes increased thickness of the mould walls towards the top. The top end of an ingot may then solidify almost as rapidly as the bottom, so that the shrinkage, which accompanies solidification in the lower portion of the ingot, cannot be fed sufficiently by liquid steel from above; this results in the formation of an extended cavity or pipe.

FIG. 89.

FIG. 90.

Clearly, any modification of conditions which will delay the solidification of the upper portion of an ingot relatively to the lower, will not only alter the ultimate shape of the cavity, but will also eliminate the extended portion together with the secondary "pipes". Such conditions are better fulfilled by reversing the direction of taper, when the shape of the pipes formed will resemble that shown by Fig. 90. The solidification of the upper portion of an ingot may be further retarded by increasing the taper of an inverted mould, so that in this case taper can be used as a beneficial influence within limits, whereas for moulds used small end up the result would be still more disastrous. Moulds, specially made for casting large end up, are either provided with a solid bottom or must be machined all over the lower open end, so that perfect contact with a flat surfaced chill plate may be ensured. Any gap between the mould and chill plate would immediately be filled with steel when teeming, which would prevent the ingot being stripped without the added labour and expense of removing the "fash". Moulds of this type are often cast with their outside faces parallel, so that the thickness of the walls at the bottom end is greater than at the top; this still further tends to accelerate solidification of the bottom portion. So far, then, as the use of moulds is concerned, those conditions which tend to reduce the extent of piping are best satisfied by top-casting into moulds, large end uppermost, when the effect of wall thickness and degree of taper may also be used to good advantage. A closed-bottom inverted mould is usually dished at the bottom, and, provided care is taken when teeming, its life may be as long as that of an open-bottom mould. An inexperienced teemer can, however, easily ruin a mould by opening out the nozzle too sharply at the start, and so burning the bottom to such an extent that "stickers" will result in subsequent casts; this applies especially when casting high carbon steels.

The bottom or "chill" plate used with open-bottom inverted moulds are cast with a deep concave depression in the centre, so that the splash caused by the stream striking the bottom is prevented from touching the walls of the mould, and from penetrating any open joint between the mould and the bottom plate. The depression in the chill plate is also rapidly filled with steel, which then breaks the force of the stream and so further prevents splashing.

Bottom-Casting.—This method is one that is most usually and conveniently used for casting a large number of ingots of small dimensions. If a large number of small ingots have to be cast separately from a ladle, there is every possibility that, owing to the very slow rate of teeming necessary, the steel will become chilled and too cold before the moulds are all filled. By adopting a system of bottom-casting a large number of moulds may be filled simultaneously, so that the rate of filling each mould is extremely slow compared to the actual rate of teeming; the ladle is thus emptied in a far shorter time and the proper condition of slowly filling each mould is satisfied. Closed-top moulds, having only a small conical gas vent, are frequently used, and have certain advantages over the open-top moulds. A group of such moulds is shown in Fig. 91, where they are mounted on a bottom runner plate suitable for casting four ingots. The bottom plate is usually cast with four or six lateral recesses radiating from a central recess, which is accordingly either square or hexagonal. The central recess contains a centre brick which is simply fitted within it by a packing of dry sand, while the lateral recesses are suitably dimensioned to hold the runner-bricks, which are mortised to fit into each face of the centre brick. If the centre of the mould is far from the centre of the plate an extension runner-brick of suitable length is used. Fig. 92 shows a four-way and six-way centre brick. Both the centre and the runner-bricks must be carefully set in the recessed plate with sand, so as to be flush with the top of it. The mould, or at least one side of it, will rest across the brick, so that any difference in level between the plate and the brick may cause the steel to run out and the ingots to "bleed". The

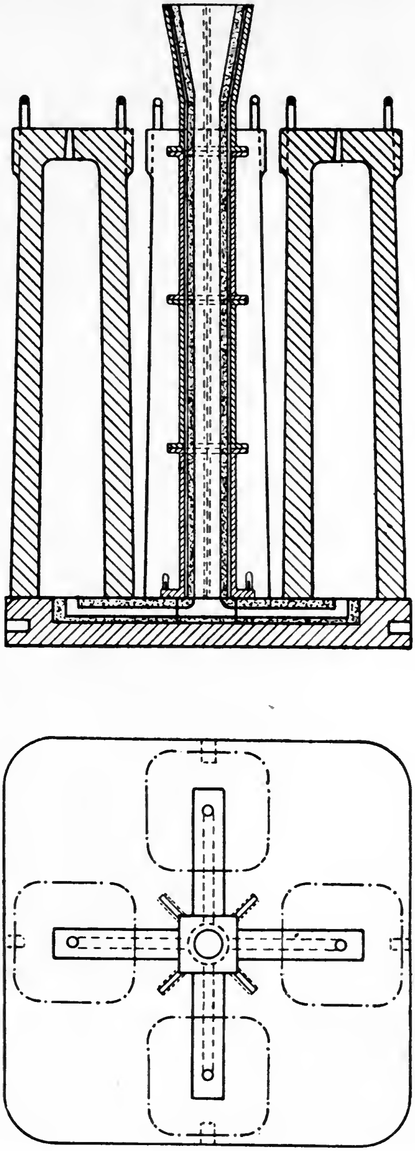


Fig. 91.

runner-brick is closed at one end, but is provided with an orifice that is best surrounded by an annular flange, which helps to deflect the stream of steel vertically upwards. It is most important that the moulds are set on the plate exactly central with this orifice, otherwise the side of the mould nearest the stream of steel may be washed and burnt; careless setting may also cause cracked ingots, due to the unequal distribution of hot steel in the mould (Brearley). The steel is fed into the centre brick from a vertical fire-brick pipe, enclosed in a cast-iron or steel frame called a "trumpet". Fig. 91 shows a section through such a trumpet, each half of which is built up of semicylindrical sections fastened together by bolts or cotter pins; in the figure it is shown in one length, the two halves being cottered together. The pipe bricks, (Fig. 93) are carefully laid and fitted together in one half of the trumpet on a bed of "compo" or ganister, the bottom brick, which is sometimes mortised at its lower end to fit the centre brick, being exactly flush with the end of the trumpet. The bell section is finally set in position, and the entire pipe daubed over with ganister or "compo". The other half of the trumpet is then laid upon it and cottered to the under half, squeezing out any excess of the bedding material. The built-up trumpet is carefully dried before use to prevent the brick pipe bursting through sudden generation of steam. The lowest section or bottom end of the trumpet has a flange, which is either bolted on to the bottom plate or held down by weights, as shown in Fig. 91; this is done to prevent the steel finding its way under

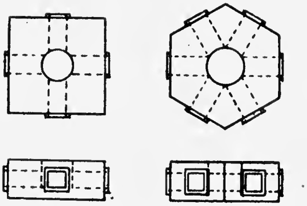


Fig. 92.


Fig. 93.

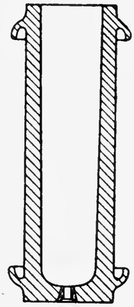


Fig. 94.

the trumpet and lifting it owing to the pressure of the column of steel in the trumpet pipe.

When closed-top moulds are used, the steel rises up until it reaches the base of the vent, where it then normally chills, but if teeming at this point is too rapid and the steel very hot it may sometimes spurt out; any "fash" so formed round the vent hole is immediately removed from the mould top to prevent the steel being held fast in the vent. Should the latter frequently occur the vent becomes worn and enlarged, causing constant trouble with "stickers". A plan commonly adopted is to build a small mound of loam above the vent hole and then prick through with a large nail, in this way the conical plug of steel filling the vent cannot become fastened to the mould top. Care is always taken to dry the runner bricks and moulds thoroughly, as, apart from the possibility of the bricks bursting, there is sometimes a danger of the moulds lifting through any sudden generation of steam. After the steel has become solid in the vent, the ingots may be constantly fed under pressure from the steel in the trumpet, since the bell will be at least 12 inches above the ingot top. In the case of open-top moulds, the bell will be at the same level as the ingot head, and consequently no feeding is possible from the trumpet; "dozzles" and "cheek" bricks may be fitted, however, in the top of the mould to delay solidification of the ingot head, and so reduce the extent of "piping".

The solidification of a bottom-cast ingot takes place in a different manner from one top-cast. In the latter case hot steel is constantly fed from the top, so that, at the moment of filling, the steel in the head of the mould will be hotter than elsewhere; in the former case these conditions are reversed, so that the region of piping in the ingot will be lowered, following upon the slower cooling of the lower portion of the ingot relatively to the top; at the same time, the chilling effect at the head of the mould is greater than at the bottom, which further accentuates the tendency towards the formation of an extended primary pipe and small secondary pipes. This method of bottom-casting small ingots in groups may be safely adopted, provided there is no danger to be anticipated from the presence of pipes, which are always most pronounced.

There is yet one other modification of the method of bottom-casting, which entails the use of a special mould, having a partially closed bottom and used large end uppermost. Such an arrangement is shown by Fig. 94. The bottom of the mould is thick, and has a central conical hole into which fits a fire-brick plug designed to make a morticed connection with the orifice of the runner-brick. The stream of steel is guided vertically upwards without risk of washing the sides of the mould. The bottom of the ingot is exposed to a greater chilling effect, and, since the upper portion of the ingot is wider, the rate of solidification is there retarded. These altered conditions in the process of solidification tend to reduce the length of pipe and eliminate secondary pipes, so that the arrangement presents distinct advantages over other methods of bottom-casting. There is, however, one serious objection, which lies in the preparation and correct setting of the mould before casting. It is obvious that any open joint between the fire-brick plug and the mould will become filled with steel and prevent withdrawal of the ingot; unfortunately, it is not always possible to rely upon this joint being properly filled up with clay, so that the success of the method will depend entirely upon the human element. All methods of bottom-casting demand considerable care from pitmen, as any loose dirt left in the runner-bricks will be washed away and probably become entrapped in the ingot.

Dozzles, Cheek-bricks, and Sinking Head.—It was realised in the early days of crucible ingot casting that the formation of a pipe might be prevented by keeping the head of an ingot molten until solidification was complete at lower levels. This was accomplished by placing a strongly heated fire-brick, with a slightly conical central hole, in the head of the mould while teeming was momentarily stopped; the steel quickly solidifies round its lower edge and holds it fast to the ingot walls. The central cavity in the brick is then filled with steel, which remains molten and serves as a reservoir from which liquid steel is constantly drained as shrinkage proceeds lower down. Such bricks are usually known as "dozzles" or "cores," and are made in a large variety of shapes and sizes (Fig. 95). It is not always possible to pre-heat the largest sizes, and the beneficial effect is then greatly lessened. "Cheek" bricks are used for large moulds in place of the fire-clay "dozzle," and are moulded with a small lug on their upper edge which rests on the top of the mould (Fig. 96). As these bricks may sometimes become firmly attached to the ingot head, the lugs should be cracked after teeming to prevent any tendency to hold the ingot and cause it to "pull".

"Sinking heads" are usually employed for large ingots, and consist of a light cast-iron or steel box, rammed up with loam, "compo," or moulding sand, leaving a central cavity shaped and tapered as desired. The "sinking head" rests on the mould top, and care must be taken, when setting in position, that the joint is well closed to prevent "fashes" being formed which will hold the ingot and cause it to pull. Sometimes the upper part of a large mould is recessed all round, and the space filled up with "compo" or other material; in this way the sinking head becomes part of the mould itself. In many cases the use of such devices is not alone sufficient, so that charcoal is sometimes thrown on the liquid steel to further retard solidification by the heat of its combustion. Sir Robert Hadfield advocated the use of a layer of fusible neutral slag between the steel and the charcoal, and also used an air blast to promote a higher temperature of combustion. With this process the steel sinks quite level in the "compo" or ganister-lined sinking head, leaving only a shell of steel adhering to the sides.

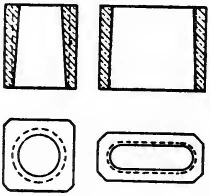


FIG. 95.

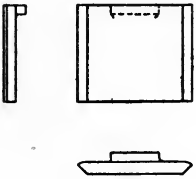


FIG. 96.

Tun-dish Casting.—The importance attached to the speed of teeming cannot be exaggerated, while the difficulty of casting a large number of small ingots under nearly similar conditions from a large ladle has been already mentioned. Group casting certainly provides one solution of the difficulty, but adds to the cost of casting, and does not produce such a reliable ingot as one top-cast large end up.

It is only a few years since it was proposed to interpose an auxiliary receptacle between the ladle and the moulds for the purpose of dividing the stream from the ladle into several smaller streams that feed directly into the moulds. Such a receptacle is known as a tun-dish, an example of which is shown in Fig. 97. The advantages of this system of casting have been well proved, and one instance may be quoted where the quantity of billets regularly scrapped on account of cracks was reduced from over 50 per cent. down to 5 per cent. and under. J. N. Kilby has given some conclusive figures proving the value of the tun-dish, which may be used even in large plants without causing any inconvenience in the shop. A separate tun-dish is usually supported above each group of moulds that it feeds, but sometimes it is attached to the under-side of the ladle and moves with it. They are generally designed to feed two moulds, but are also used for four. The tun-dish must itself be carefully fed, so that the stream of steel from the ladle is equally divided between the several nozzles. Failing to do this, the moulds will not fill at the same speed and ingots of unequal length will result, which may prove serious in cases where the moulds are provided with dozzles or cheek-bricks. The dish is lined with either thin fire-brick splits, or a facing of rammed "compo," or ganister, and is usually deep enough to hold a reservoir of steel at least 8 inches in depth, which helps to equalise the pressure of steel above each nozzle, and, at the same time, prevents the stream of steel from the ladle splashing up from the bottom. Apart from the slow speed of filling, a further advantage is gained, since the streams from the tun-dish are steady and not forced through the nozzles under great pressure; the commotion in the mould is therefore far less, and the skin of the ingot freer from the usual splash markings of top cast ingots.

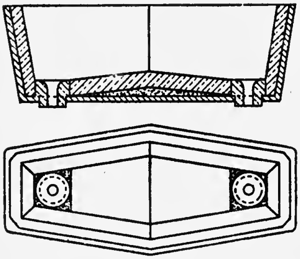


FIG. 97.

CHAPTER X.

APPLICATION OF THE ELECTRIC FURNACE TO FOUNDRY PRACTICE.

Introduction.—The electric furnace has certain features that make it especially suitable, and in many respects superior to other steel-making plant, for the manufacture of light steel castings. It has, of course, disadvantages as well as advantages, and it is only by carefully studying the operating conditions, both technical and economic, that its suitability may be determined for individual cases. Prior to the introduction of electric melting into foundry practice, castings of thin section and high ductility were made by the crucible process, or by employing some form of small converter such as the Tropenas, Bessemer, and later, the Stock. The latter plants are satisfactory, provided that steel with a carbon content of less than ·2 per cent. to ·25 per cent. is not regularly required for castings of very thin section. There is no doubt that thin castings with a carbon content as low as ·1 per cent. to ·15 per cent. can be made, at any rate, from the Stock converter, but such blows cannot be repeated with absolute certainty under the always variable conditions of blowing.

With the rapid development of motor engineering and the introduction of aeroplane construction, there has been a growing demand for steel castings of thin section, but not necessarily of small dimensions. The factor of safety is an element that has to be most carefully considered in these particular branches of the engineering industry, so that the demand for light castings made from steel of a high degree of purity and having the necessary reliability in service has now become general.

The chemical composition of crucible steel is dependent upon the quality of scrap melted, and the degree of carbon absorption from the pot. Clay pots are not so satisfactory for foundry purposes as the tougher and more refractory graphite crucibles, which may be allowed to cool off between melts. Apart from the high cost of the latter type, there is the disadvantage that unless special precaution is taken to add a little ore to the charge, the absorption of carbon by the steel will prevent a high degree of ductility being obtained. The iron ore is added in slight excess of the quantity required to boil out the carbon absorbed, so that the steel, when hot enough for pouring, will be oxidised and require somewhat drastic "killing" with aluminium before casting into moulds. It must not be assumed that castings made in this way from crucible steel are not of excellent quality, provided the raw materials are good, and, for a very small output of uniformly light castings, of which the heaviest can be poured from one or at the most two pots, there is a good deal to be said in favour of this method of steel melting. The production of mild steel from crucibles is almost entirely a matter of steel melting, and can hardly be expressed as "steel-making". When a crucible charge is melted and hot enough to pour, no attempt can be made to control the carbon content, and it is only on examination of a casting itself that the suitability of the steel can be judged. When dead soft steel is melted in graphite crucibles, the final carbon will in most cases reach ·2 per cent. and at times will fall within a wider range, reaching up to ·6 per cent. Such cases as the latter are not, of course, frequent, but are mentioned merely to indicate that the crucible product is not under the same degree of control as converter or electric furnace steel.

Prior to 1914 the converter at least held its own for foundry purposes, but the subsequent shortage of suitable raw material, and the increasing confidence in the economic performance of electric furnaces, caused the latter to gain favour for the manufacture of light soft castings. The converter is restricted to the acid process, and consequently the composition of the final steel as regards sulphur and phosphorus is dependent solely upon the composition of the raw material used. The temperature of casting will likewise depend upon the silicon content of the charge, which must necessarily be high if a high temperature and degree of fluidity is required; this imposes a limitation to the percentage of scrap in a charge, and to the quality of the pig-iron used. The acid process is well suited for foundry purposes, since the character of the slag renders its separation from the steel a matter of great ease, and therefore allows the system of lip-pouring to be adopted. The latter method certainly has advantages over bottom-teeming, which is more likely to cause "scabbing" and dirty castings, but, on the other hand, a higher casting temperature is necessary. Stopper troubles disappear and the steel always enters the mould at a steady, low pressure, due solely to the fall from the ladle lip and not to the pressure exerted by a head of steel behind the stream, as in bottom teeming. On the other hand, unless handshanks are used, lip-pouring is not suitable for very light work which requires a precise and rapid ladle control. The limitation of the converter to the acid process is, so far as the problem of casting is concerned, an advantage rather than otherwise, as both lip-pouring and bottom-teeming are open to choice according to the nature of the work being cast.

The electric furnace offers far more scope to the steel-maker than the converter, and from a purely technical standpoint, the quality of the steel is almost independent of the class of steel scrap used for the furnace charge. It is optional to use either the basic or acid process, and the ultimate choice will depend upon the specification demanded and the quality of scrap available. The temperature of casting is only limited by the fusing point of refractories, and the chemical composition of the steel is always under exact control. The relative advantages of the basic and acid processes, as conducted in the electric furnace, require further consideration.

The basic process enables both phosphorus and sulphur to be reduced to very low limits, so that any chemical specification can be readily met, even if impure scrap high in both these elements is used. The deoxidation of the steel can also be conducted under highly reducing conditions, which certainly presents great advantages over the cruder method of "killing" wild steel by alloy or aluminium additions immediately before casting. In cases where the amount of phosphorus and sulphur present in the raw material exceeds the specification limit of the casting, the basic process becomes obligatory. On the other hand, if the scrap available is sufficiently low in sulphur and phosphorus, and merely requires remelting with adjustment of carbon, silicon, and manganese, then the acid process, so far as the actual steel-making is concerned, will fulfil the purpose. If, then, it is not imperative to use the basic process, the choice of process will rest solely upon economic considerations based upon the prices of what may be called low grade and high grade scrap, and other minor advantages which each may present. The chemistry of steel-making with an acid slag is practically the same for both open hearth and electric furnaces, when applied to foundry practice, only in the latter case the entire or predominant portion of the charge will be steel scrap, which renders the possibility of over-oxidation of the bath more considerable. For this reason, the scrap should be selected and be free from rust, since the process of deoxidation cannot be carried to the same extent as under a basic slag. The acid process certainly shows a small saving in power consumption since the extra time occasioned by the use of a second slag is eliminated. The steel may be finished and poured when the correct casting temperature has been reached, assuming that the bath has been properly boiled down and is thus fairly free from oxides. The convenience of handling steel under an acid slag has been already mentioned, and should not be disregarded when deciding upon the process to be adopted. For cleanliness and convenience of operation the acid process is to be preferred, and will be found rather more economical, if scrap of suitable quality and price can be secured.

The comparative advantages and disadvantages of the crucible, converter, and electric processes for the manufacture of light steel castings are summarised and tabulated in Table I.

Early Development and Statistics.—It was early recognised that the electric furnace offered special advantages in certain departments of foundry practice on account of the high casting temperature attainable. The Stassano furnace was at first favoured for this purpose, but chiefly owing to the severe punishment of the magnesite roof, furnaces of the direct arc type are now more widely used. In the year 1909 the firm of George Fisher in Switzerland had installed a small, single-phase Heroult furnace for the production of very light steel castings, which included the first type of hollow-spoke motor vehicle wheels used in Great Britain. The first electric furnace laid

TABLE I.—Comparison of the Crucible, Converter, and Electric Furnace Processes for Foundry Use.

	Crucible.	Side-blown Converter.	Electric Furnace.
Character of the raw materials used, and proportion of scrap melted.	100 per cent. wrought-iron or steel scrap commonly used. Small proportion of clean foundry scrap also melted in charge. Impossible to re-melt all the foundry scrap made. Value of foundry scrap less than the raw materials used.	It is possible to blow a charge containing 30 per cent. of foundry scrap, provided a high silicon haematite pig is used, but 20 per cent. is a more usual figure. Can use up the bulk of foundry scrap made. Value of the foundry scrap is less than the average value of the metal before blowing.	A charge entirely composed of scrap can be used. All the foundry scrap made can be re-melted, if desired. Usually the foundry scrap made is worth more than the steel scrap bought for melting, in which case it is sold at a figure that has appreciated by treatment in the furnace.
Melting loss.	Very small, and dependent upon the cleanliness of the scrap melted. Loss entirely chemical.	Usually a 14 per cent. to 24 per cent. loss on conversion of the clean raw materials consisting of pig and steel scrap. Loss is both mechanical and chemical.	Loss averages about 7 per cent. to 10 per cent., when using fairly clean scrap in a form that is not liable to excessive oxidation in the furnace. Loss both mechanical and chemical in basic furnaces, and chemical only in acid furnaces.
Fuel cost.	Coke melting expensive and consumption of fuel high under forced draught. Gas, when used, is cheaper. Fuel cost per ton melted is almost independent of the output, when using coke.	Cost of coke or oil fuel used for melting the raw material is very low. Fuel cost per ton steel blown is almost independent of the output.	The cost of melting is high and depends also upon the average load factor or output over a long period.
Size of unit and output possible.	Furnaces can be installed for any desired output, however small.	Minimum capacity limited to ½ ton, being the smallest size commercially practicable. Possible daily output much greater than with an electric furnace of similar capacity.	The smallest satisfactory unit has a capacity of ½ ton, and cannot produce such a large daily output as a corresponding converter plant.
Suitable types of castings and temperature limitation.	Generally very light castings only are made. Possible to pour thin castings since no appreciable heat loss on transfer to mould.	Suitable for medium light and light castings. Very thin castings can be run, but not without risk of short-run wasters. Temperature of the steel depends upon the variable condition of blowing, governed largely by the condition of the lining.	Suitable for medium light and light castings of any description.

TABLE I.—*Continued.*

	Crucible.	Side-blown Converter.	Electric Furnace.
Chemical composition.	Analysis rather irregular, due to varying absorption of carbon from the pot; sulphur and phosphorus dependent upon analysis of scrap.	Chemical composition under good control, giving a consistent carbon percentage. Sulphur and phosphorus in the charge increase slightly during the blow. Loss of Mn and Si to be allowed for on addition of alloys.	Analysis of consecutive heats very consistent owing to the exact chemical control during refining and finishing operations. Removal of P and S possible by basic process. Exact calculated additions of Mn can be made without allowing for any oxidation loss.
General properties of castings and nature of defects.	Crucible steel castings do not case-harden well if the carbon exceeds ·15 per cent. Have a tendency to tear or pull if the steel was highly oxidised just before pouring. Physical tests will only be consistent in so far as the chemical analyses are regular.	Liable to slag inclusions and dissolved oxides. Minute slag inclusions frequently cause test pieces to fail either under the bend or tensile test, which increases the number of wasters. This is, probably, the most serious difficulty with converter steel. The physical tests are not so consistent as might be expected from the chemical uniformity of consecutive casts for the reasons above stated. Castings are liable to tear on contraction owing to the tenderness due to oxides; this is also augmented by high sulphur.	Case-hardens very well up to ·2 per cent. C. Acid steel occasionally shows slag inclusions. Physical tests are most consistent and are not influenced by factors other than ordinary chemical composition and heat treatment. The physical tests show a rather higher ductility than converter or crucible steel having the same resistance to tensile stress. Owing to the unlimited casting temperature possible the risk of short-run castings is reduced to a minimum.

down for the production of castings in England was erected by the Braintree Castings Co., Ltd., in 1911. No considerable progress was made anywhere in the development of the electric furnaces for foundry purposes until 1914, when, owing to the rising price of hematite pig-iron and the availability of an increasing supply of cheap steel scrap, its special merits were more generally utilised for meeting the growing demand for light intricate castings. The early electric steel castings were far superior to the best malleable iron castings then being used in the motor engineering trade; they were of uniform quality throughout, could be case-hardened, and did not suffer from the disadvantage of warping, as so commonly happens during the malleablising process. Drop forgings made in complex dies were at that time in their infancy, but are now used extensively in the place of small castings, when the shape of the article permits. Owing, however, to this latter limitation, the field for drop forgings must always be restricted.

The marked development of the electric furnace for foundry purposes is shown by the following figures, which give the output of castings during the last few years in Great Britain and America. Previous to 1915 the returns for castings and ingots produced in Great Britain were not segregrated:—

	1912.	1913.	1914.	1915.	1916.	1917.	1918.	1919.
Great Britain	—	—	—	2,000	9,288	15,600	44,901	30,000
America	4,162	9,207	8,551	23,064	42,870	64,911	58,000	42,000

The rapidly increasing use of the electric furnace in foundries is not due to any great economy in the production of the steel, but rather to a demand for a higher grade material, and to the considerably reduced loss occasioned by short-run, defective, and rejected castings. The latter include castings returned from machine shops and those rejected on failure to comply with the specified physical tests.

Arrangement of Plant.—The furnace installation and all ladle accessory plant are best situated at one end of a foundry; all supplies may then be handled and the furnace manipulation conducted so as to cause the least interference with work on the foundry floor. The arrangement is also preferable from the furnacemen's point of view, as the working space around the furnace is less likely to be used as a dumping ground for boxes, sand, and other foundry material. In many shops there is one heavy crane for both furnace and casting use, and lighter ones for the manipulation of boxes on the foundry floor; by the above arrangement of plant, the cranes are always situated in a position where they are mostly used, and are therefore less likely to interfere with one another in their operation. A battery of three 2-ton single-phase furnaces installed at the works of Messrs. Thwaites Bros., Bradford, is shown in Fig. 98.

The furnace plant is erected at one end of a long foundry bay and constitutes an entirely self-contained steel-making installation.

Satisfactory means of heating the ladle are always provided, and, in cases where hand shanks are used for the transfer of steel from the ladle to the moulds, special apparatus is installed for their rapid and convenient heating, so as to minimise the production of "skulls". Gas burners are sometimes used for this purpose, but a simpler method consists of inverting each shank over a separate small coke or charcoal fire, built in an unlined steel fire-box provided with a perforated removable bottom and adjustable air blast. With this appliance shanks can be raised to correct temperature in half an hour. The floor plates, which cover the working floor space around the furnace, are sometimes built a few inches above the shop floor level; this more readily enables a general condition of cleanliness to be maintained round the furnace and slag pit.

For the purpose of controlling the melting loss and general efficiency of the furnace, it is necessary to know the actual weight of steel poured into the ladle; this is a difficult figure to arrive at accurately from the weight of finished castings, so that the use of a dial crane hook weighing machine is always to be recommended in foundry practice.

Choice of Furnace Capacity.—At the present time the electric furnace is generally used for the production of light castings of about 3 cwts. or less, and has not yet competed in the heavier trade of open-hearth castings in Great Britain. The number of moulds laid down for every ton of steel cast will usually be considerable, thirty to forty being quite a common figure. Owing also to the light character of the castings, the moulds will be proportionately large, so that the floor space and the number of moulds required for a ton of steel castings will be greater than in most converter and open-hearth plants. When a large number of moulds are laid down for one cast, the majority will be set and remain on the floor long before they are actually filled, so that the boxes are not in constant use. It is obvious that, by laying down and casting a fewer number of moulds which could be promptly knocked out and returned to the moulder for further use, an economy of floor space and moulding box plant would naturally result. Therefore, for a given daily

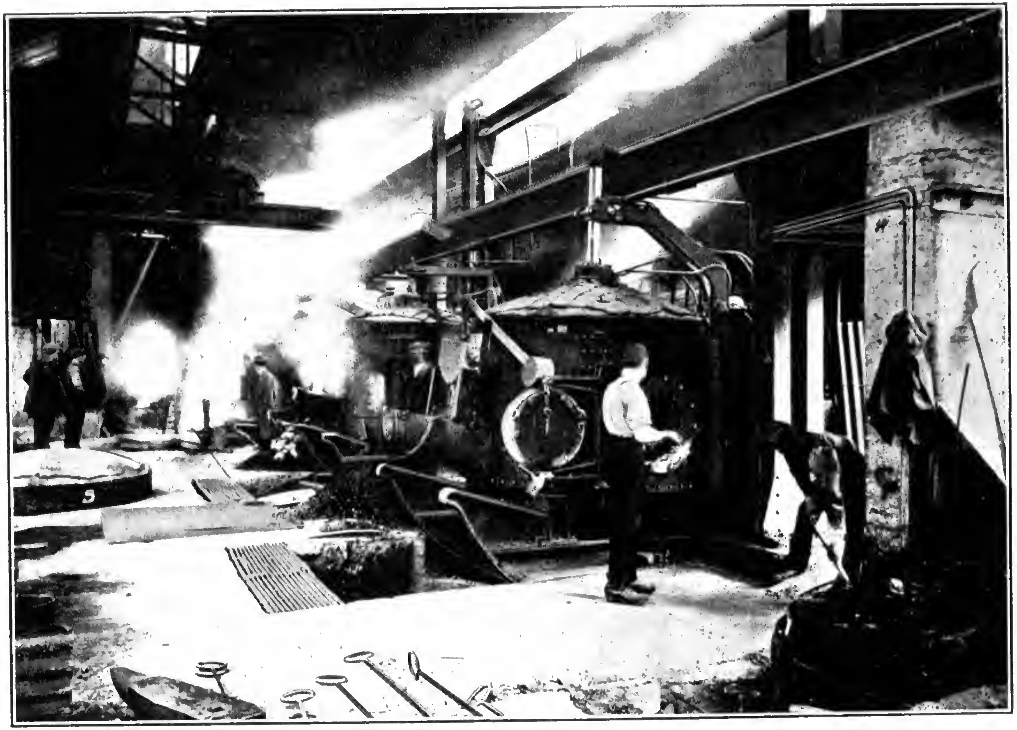


FIG. 98.

[*To face p.* 204.

output it is better to install furnaces of small capacity, which may be worked so as to be ready for casting at regular intervals, and so collectively provide a frequent supply of steel to the foundry in small quantities. The cost of a steel-making plant consisting of several small units will naturally be higher than one large unit having the same daily output, but the numerous advantages otherwise gained will more than compensate for the increased establishment and labour charges. It may well be expected that the electric furnace, if operated in conjunction with a basic open-hearth or converter, will find an extended field of application in the production of heavy castings; in this event, furnaces of large capacity would be installed to meet the demands of the heavy engineering trade.

Specification and Mechanical Tests.—Castings are generally ordered to fulfil a physical test specification, no restrictions being placed on analysis other than for phosphorus and sulphur; this rule allows the steel-maker far more scope in the choice of analysis, which can be varied according to the heat treatment to which the castings are submitted.

The standard specifications for steel castings adopted by the American Society for Testing Materials in 1912 are given below:—

	Hard.	Medium.	Soft.
Tensile stress per sq. inch . .	36·0 tons	31·0 tons	27·0 tons
Yield point	16·0 ,,	14·0 ,,	13·0 ,,
Elongation on 2 inches . . .	15 per cent.	18 per cent.	22 per cent.
Reduction of area	20 ,,	25 ,,	30 ,,
Cold bend 1 inch × ½ inch . .	—	90°	120°

All castings to be annealed and slowly cooled.

Some American Marine specifications are more exacting than the above, as shown by the following example:—

Tensile stress, 36·0 tons; Yield point, 18·0 tons; Elongation on 2 inches, 17 per cent.; Reduction of area, 25 per cent.; Bend, 90° (1 inch × ½ inch). The above specifications are intended for steel castings with phosphorus and sulphur below ·05 per cent. and are easily met by electric steel made either from cold scrap charges or by the duplex process. In America all Government castings are ordered in the annealed condition.

The specifications adopted by the British Engineering Standards Committee for marine castings are very similar to the above, but do not generally apply to the light intricate variety of automobile and other such castings. When specifications are given, it will be seen from the figures given below how easily they can be met and, moreover, outclassed to an extent which places electric steel castings in a special category. The following are typical tests of electric steel castings made by the basic process from cold scrap charges :—

	Max. Stress, tons per sq. inch.	Yield Point, tons per sq. inch.	Per Cent. Elongation on 2 inches.	Per Cent. Reduction of Area.
I. *Analysis*, C ·15 ; Mn ·1 ; Si ·21.				
(*a*) As cast	34·3	23·0	26	—
	31·1	21·8	28·5	—
II. *Analysis*, C ·2-·25 ; Mm ·5-·55 ; Si ·3-·35.				
(*a*) As cast	30·0	—	16	—
(*b*) Annealed at 900° C. to 950° C. and slowly cooled.				
	31·71	22·14	34	52·7
	35·0	23·18	23	34·6
(*c*) Same treatment as (*b*) but from a large casting.				
	31·07	17·73	33·25	52·2
	27·01	18·75	38·5	57·8
(*d*) Annealed, water quenched from 750° C. and tempered at 550° C.				
	39·05	27·93	22·0	38·3
III. *Analysis*, C ·4 per cent. ; Mn ·5 per cent. ; Si ·25 to ·35.				
(*a*) Annealed and cooled in air.				
	40·09	27·01	17·25	21·1
	42·6	24·4	20·0	—

Acid steel made from scrap low in phosphorus and sulphur will give somewhat similar physical test results. At one British foundry, equipped with several acid lined furnaces, the steel is practically standardised and made to the following analysis :—

C	·25 – ·3	per cent.	
Mn	·8	,,	
Si	·3	,,	
P and S	under ·04	,,	each.

The castings, which are very light and intricate, are all annealed at 900° C. for 1½ hours and air cooled ; test pieces submitted to this treatment regularly give the following physical test results :—

Max. stress	30/35 tons.
Yield point	18/21 ,,
Per cent. elongation	17/20 ,,
,, reduction area	30 per cent.

Annealing.—The simplest process of annealing castings consists of:—

1. Slow reheating to a temperature above the highest arrest or Ac point.

2. Soaking at that temperature, so as to break down the coarse crystalline structure.

3. Slow cooling through the recalescence points to about 200° C. to 300° C., after which rapid cooling in air is permissible. The above treatment is sometimes varied by rapidly cooling the casting from the soaking temperature to that of the lowest recalescence point, which means a drop of about 150° C. for mild steel, and then slowly cooling from that temperature; this treatment gives a finer structure than if the steel were allowed to cool slowly from the soaking temperature down to its lowest recalescence point.

If an unannealed casting is examined microscopically, it will be found that the structure is either coarsely granular or else exhibits a large irregular, triangular shaped pattern. Both these structures are due to the prolonged high temperature to which the steel is exposed after solidification, and differ only by reason of slow or rapid cooling respectively from the temperature of the highest to that of the lowest recalescence point. This coarse structure is a source of weakness, as the ferrite areas are more easily torn apart than when irregular in form, more finely disseminated, and more closely interlinked. By soaking such cast steel at a temperature just above the highest arrest or Ac point these coarse crystalline grains are broken down, and complete diffusion of the carbon results. Slow cooling from this temperature allows the excess of ferrite to fall out of solution again, with total elimination of the original structure.

The microphotographs of similar magnification shown in Figs. 99 and 100 illustrate the marked change brought about in the crystalline structure of a mild steel casting by an annealing treatment. The steel contained ·2 per cent. C; ·22 per cent. Si; ·53 per cent. Mn, and it will be seen how the coarse crystalline structure of the steel as cast has been entirely changed to one consisting of finely disseminated and irregular shaped patches of ferrite and pearlite. The marked improvement in the ductile properties of this particular cast of steel resulting from this change of structure is shown by the following test results:—

	As Cast.	*After Annealing.*
Ultimate stress per sq. in.	30·46 tons.	30·9 tons.
Yield point . . .	—	18·8 ,,
Elongation on 2 ins. .	18·0 per cent.	34·0 per cent.
Per cent. reduction of area	20·0 ,,	40·0 ,,

It is obvious from these figures that a steel casting in an annealed condition will be far more resistive to sudden fracture by shock than when in the "green" state.

The grain size of annealed steel will depend upon (1) the soaking temperature to which the steel is raised; (2) the rate of cooling from the soaking temperature to that of the lowest recalescence point; (3) the rate of cooling from the lowest recalescence point to atmospheric temperature.

The temperature of annealing rises from 800° C. to 850° C. for medium carbon steels up to 900° C. to 950° C. for mild steel, while higher soaking temperatures tend to produce a coarser grain. Importance is not usually attached to the effect of rapid cooling from the soaking temperature to that of the lowest recalescence point, but in some cases the practice is followed of opening the annealing stove doors to cause rapid cooling, and then closing them up again until the castings are ready for removal. It is generally acknowledged that the finer the grain of an annealed sample of steel the better will be its physical properties, and to secure this in practice the castings are often withdrawn from the annealing stove and cooled in air. Excellent tests may be obtained in this way, but unfortunately there are risks of setting up internal stresses in castings which vary in thickness, owing to the rapid and irregular rate of cooling. In America this point is considered of sufficient importance to warrant a general stipulation that all annealed castings shall be

Fig. 99.

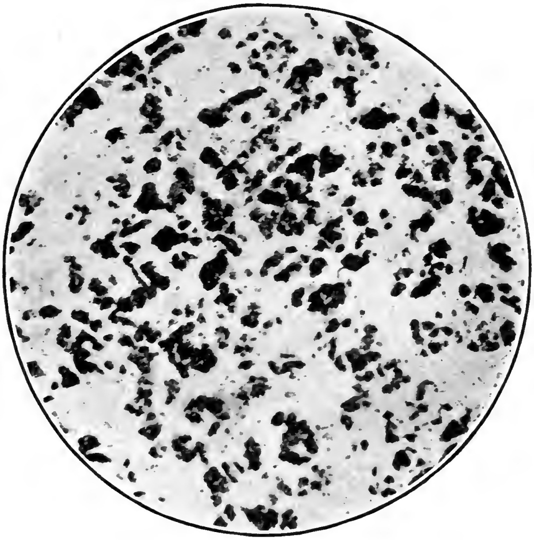


Fig. 100.

[*To face p.* 208.

slowly cooled. Castings cooled slowly will give a poorer test generally, but will be more reliable in service.

The process of annealing also removes the unequal contraction stresses set up in a casting during its initial cooling; this applies particularly to castings that vary considerably in thickness, and which may show "pulls" due to this cause. There is considerable diversity of opinion as regards the advisability of annealing dead soft steel castings containing less than ·15 per cent. carbon. It cannot be denied that annealing will improve the tensile, yield point, and elongation figures of all carbon steel castings, by converting the original coarse structure into one of finer grain which produces a fibrous fracture. However, rough test pieces cast ¼ inch thick from such low carbon steel should bend double when cold, so that, so far as ductility and resistance to shock are concerned, castings made in this low carbon material will, in the "green" state, fulfil the usual requirements. Therefore, unless annealing is actually specified there seems little reason to incur the added expense of improving the quality beyond what is required by the user for the particular purpose intended. The question of annealing dead soft castings must then be left to individual manufacturers to base their decision according to the demand. Unannealed castings, as has been stated, will always be unequally stressed, and their quality in this respect will be improved by heating to a few hundred degrees centigrade.

Defects of Steel Castings.—The production of sound steel castings depends far more upon the art of moulding than of steel-making, and for the purpose of remedying defects a careful distinction must be drawn between their possible causes. Defects in castings may be sometimes due to an improper condition of the steel when cast, but are more generally caused by unsuitable methods of casting, more especially in respect of the nature and construction of the sand mould. Careful examination of a defective casting will usually provide definite evidence as to the cause of the defect, which in certain cases can be remedied by suitable modification of the composition and physical condition of the steel when cast.

Blowholes.—The term "blowhole" is generally applied to cavities formed by the generation of gas, which accompanies the chemical reaction between the carbon constituent of the steel and dissolved iron oxide that takes place on solidification. In the case of ingots such unsoundness is generally due to the imperfect removal of the dissolved oxides from the steel before casting. This does not always apply in the case of castings, since it is quite possible for sufficient oxide to be formed and absorbed by the liquid steel in the moulds, owing to generation of steam from a damp sand skin. The blowholes formed by imperfectly deoxidised steel will be found distributed irregularly throughout the casting from its skin inwards, and will usually be lined with a thin colour film of iron oxide. An unsound casting will have unusually sharp edges, owing to the expansion of the outside skin, which is caused by the internal pressure of the gases generated as solidification proceeds. Steel that may produce a perfectly sound casting in dry sand will sometimes blow when cast into imperfectly skin-dried "green sand" moulds; this is due to the steam being generated from the moisture present, which causes local oxidation unless there is sufficient silicon and aluminium in the steel to neutralise its oxidising influence. Blowholes, which are thus due to the action of water vapour, are generally confined to the region of the skin, and do not persist to the centre of the casting in other than exceptionally bad cases. If the trouble cannot be remedied by ensuring that every mould is properly dried, it may be greatly mitigated by a more liberal addition of aluminium to the steel. Silicon will also assist in the same way as aluminium, but its action is not so certain and rapid. For the above reason alone it is quite a common practice to add one pound of aluminium to a ton of steel in the ladle; this applies equally to electric, crucible and converter steels.

Short-run Castings.—When steel is poured into a sand mould, it may happen that, owing to insufficient temperature or to the thinness of the pattern, it fails to fill all parts of the mould entirely and produces a "short-run" casting. For certain engineering purposes lightness may be of prime importance, so that it is only by increasing the casting temperature, and not by slightly thickening the pattern, that this difficulty can be met. To obtain a very high casting temperature may entail prolonged heating with increased power consumption, so that it is generally more economical to reserve as many as possible of the lightest castings for one heat, which may be cast specially hot for the purpose.

Gas Cavities.—Insufficient venting or porosity of the sand may prevent the rapid and complete displacement of air from the moulds while filling, in which case bubbles of gas may become entrapped, and remain under the skin on the "cope" side of the casting. Large and isolated cavities resulting from this cause cannot be confused with blowholes formed by oxidation of the steel before solidification.

Brittleness.—This defect is entirely independent of the method of moulding and pouring, and is solely due to the condition of the steel when cast. Brittleness of a cold casting may be due to a moderately high carbon content, in which case annealing will often suffice to remedy entirely a defect which is then only apparent in the unannealed condition. Occasionally brittleness will be exhibited in a mild steel casting, and is then characterised by a bright coarsely crystalline fracture. This condition is usually due to a high silicon content, which occurs more especially in acid electric steel owing to the reduction of silicon from the slag under highly reducing conditions. Annealing will generally suffice to remedy this fault, which, however, can be prevented by careful control of the silicon alloy additions and the condition of the slag during the finishing operations.

Slag Inclusions.—Slag inclusions are sometimes present in both acid and basic steel as very minute particles, which under the microscope are found lying along the boundaries of the crystal grains. The slag particles usually appear strung together like a chain of beads, three such chains usually meeting in a common point of intersection. The grains, whose boundary lines they mark, are those which result from the primary formation of pure equiaxed iron crystals during the process of solidification, and which continue to grow outwardly in all directions until they meet one another. During the process of cooling, subsequent to solidification and diffusion of the carbon into the iron crystals, these slag particles exert a selective attraction for the ferrite constituent, so that in low carbon steels, they are found embedded in an area of pure iron from which the carbon bearing constituent "pearlite" is totally absent; Fig. 101 shows their appearance at a magnification of 100 diameters. These chains of slag inclusions cause planes of weakness, since the metal that divides their particles and interlinks the neighbouring grains consists of weak ferrite. Steel containing these inclusions frequently fails under mechanical test in spite of satisfactory chemical analysis and heat treatment.

These minute slag inclusions owe their origin to silicates, which exist in the liquid steel either in a state of solution or more probably in an emulsified form at the time of casting. During the process of solidification the silicates are expelled from the liquid steel, and coalesce to form minute chains of segregates. Highly siliceous compounds have considerable power of coalescence, and were it not for this property, it is doubtful whether these slag inclusions would ever become microscopically visible as such definitely arranged segregates.

It is also possible that part of the silicates are not formed until crystallisation of the ferrite begins, which then allows a reaction between the silicon and any dissolved metallic oxides to proceed. In this way the formation and segregation of the silicates would be simultaneous. Repeated annealing is sometimes sufficient to break down these ferrite areas, but only after actual displacement of the slag particles.

Shrinkage Cavities or "Draws".—Cavities due to shrinkage of the steel during solidification in the moulds are analogous to "pipes" in ingots, and are generally governed by the same laws of formation. The prevention of shrinkage cavities is a question of moulding, but may be assisted to a slight extent by reducing the casting temperature and the percentages of manganese and silicon to a minimum, consistent with other necessary factors.

Tears, Cracks, or Pulls.—This particular form of defect is due to failure of the steel to resist rupture under the tensile stresses developed during either equal or unequal contraction on cooling. Here again, the remedy usually lies in the method of moulding or alteration of the pattern, when possible. Naturally, steel that is tough at a high temperature will best resist fracture under tensile stress, and will then either yield uniformly itself, or compress portions of the sand mould and cores so as to accommodate itself freely to reduced dimensions

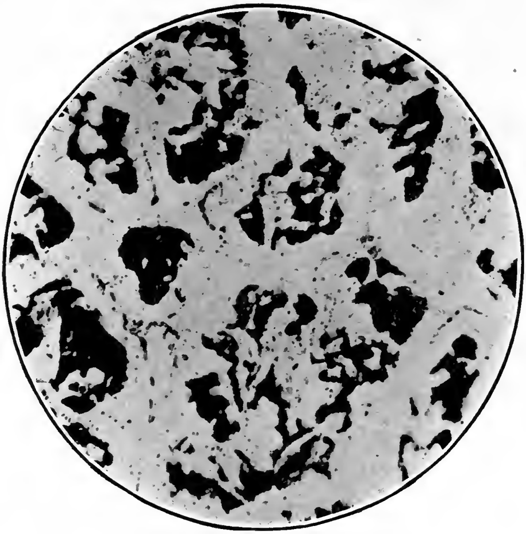


Fig. 101.

[*To face p.* 212.

without distortion. Steel that is most resistant to this type of defect will have a high degree of chemical purity, being as free as possible from dissolved oxides and sulphur, both of which cause "red-shortness" or brittleness at high temperatures. It is, therefore, also in the hands of the steel-maker to obviate these defects, which are least pronounced in basic electric steel.

CHAPTER XI.

CHARACTERISTIC FEATURES AND PRINCIPLES OF FURNACE DESIGN.

THE design of electric arc furnaces for steel-making involves a careful study of the special metallurgical conditions required, and of the electrical and mechanical means by which these conditions may best be fulfilled with simplicity, economy, and regularity of operation.

An electric furnace for steel-making is above all a metallurgical appliance, and as such must be designed as far as possible in accordance with certain conditions imposed by the particular process adopted. It is obviously impossible to embody in any furnace every feature that is desirable from a metallurgical standpoint, as this could only be done by sacrificing certain fundamental principles of the mechanical and electrical design. The result then must invariably be a compromise, in which the mechanical and electrical features are to a certain extent subordinated to and ruled by the metallurgical conditions imposed. For this reason it will be useful to outline the chemical, physical, electrical and mechanical conditions, which are either purposely or unavoidably produced during the basic or acid process of steel-melting and refining, and to point out to what extent these conditions influence the general construction and lining of a furnace installation.

Chemical Conditions.—(*a*) The various chemical reactions between basic slag and liquid steel proceed in most furnaces under the influence of intense local heat, which, especially under reducing conditions, causes slight volatilisation of the slag constituents; the basic fumes so formed have a marked tendency to flux the acid portion of the furnace lining, namely, the silica roof and walls.

(*b*) A direct arc striking downwards on to a slag blanket at an inclined angle frequently causes rotation about the axis of the electrode, and in small furnaces, where the walls are usually close to the arc, this circulation may cause considerable erosion of the hearth at the slag line. This action is, of course, greatly influenced by the temperature and the corrosive power of the slag.

(*c*) Certain operations, notably that of carburising, must be performed when the bath of metal is free from any slag covering, and it is also necessary in many cases to remove one slag prior to the formation of another. The removal of slag by skimming would be an almost impossible operation to perform in a fixed furnace, more especially as the level of the slag line is not always the same in successive heats.

(*d*) Chemical erosion of the bottom is sometimes severe, owing to the close proximity of the arc zones during the melting down stage and the local generation of heat in the hearth of conductive bottom furnaces. The bottom has, then, a tendency to become deeper, so that it would be impossible always to drain a furnace provided with a fixed taphole; this would occasion serious difficulties owing to the possible contaminating influence of a residual quantity of an alloy steel on a subsequent charge, and also to the impossibility of effecting any repair to the bottom, so long as any steel covered the worn or damaged portion.

(*e*) Burnt-lime, which is generally used as a flux in the basic process, always contains a quantity of slaked powder. On charging this flux into a hot furnace, the light dust rises with the natural upward current of air, and so comes into contact with the roof, which, being almost invariably made of silica brick, is liable to be fluxed. To avoid this action, limestone has frequently been substituted for lime, although certain disadvantages that are then introduced hardly justify its use. Experience also shows that the erosion of the silica brick is worst at that point where these dust-laden gases escape from the furnace around the electrodes; here the annular opening is constricted and the increased velocity of the gas escaping under pressure accentuates the fluxing action. Any means by which these gases could be prevented from so escaping would prolong the life of that part of the lining.

(*f*) The various metallurgical processes depend upon definite chemical reactions, which must be allowed to proceed uninfluenced by chemical conditions other than those purposely introduced. Mechanical disintegration of a furnace hearth or roof will invariably exert a contaminating influence upon the metal and slag, and in certain cases will render it impossible to maintain the desired chemical conditions. Such disintegration and failure of refractory materials may also be caused by improper treatment during their preliminary heating, as has been explained elsewhere. Care must therefore be exercised in the choice of suitable materials and in the method of employing them for the hearth and roof construction.

The highly reducing properties of certain slags will be vitiated, if not destroyed, by the oxidising influence of air, when allowed to enter the furnace too freely. This demands attention to door construction and to the restriction, as far as possible, of the annular openings surrounding the electrodes, as the free escape of gases at this point induces natural convection currents of air through the furnace.

(*g*) The chemically corrosive action of acid and basic slags on furnace banks is sometimes considerable, more especially when they are highly charged with iron oxide derived from rusty scrap. Suitable refractory material must be chosen to withstand as far as possible the corrosive action of such slags, and the natural slope of the hearth at the slag line should be such as to permit ease of fettling after every heat.

(*h*) The various reactions between carbon and oxygen, which proceed under the oxidising and reducing slag conditions, are accompanied by liberation of carbon monoxide. This gas is at times generated so rapidly that it is forced to escape through the door crevices under slight pressure. It is also often charged with iron oxide fume during the melting down stage, and therefore causes erosion of the silica brick lining at the various points of escape.

For this reason the door arches and roof openings tend to suffer most, and it is therefore advisable to construct the former in such a way that they can be renewed without damaging the adjoining walls.

Physical Conditions.—(*a*) To satisfy various metallurgical conditions it is necessary to raise a bath of steel to a definite temperature, the maximum being usually dependent upon the casting temperature desired. From the enclosed and confined nature of electric furnaces, and under certain methods by which the arc heating is applied, the temperature of the upper lining may be considerably higher than the bath of steel when ready for casting. Such excessive lining and roof temperatures, when impossible to obviate, are due to electrical and mechanical conditions which produce unshielded or flaming arcs. With an ideal mode of electric arc heating the heat generated would be entirely absorbed by the furnace charge alone, so that the furnace walls and roof would only receive heat by radiation from the heated charge. In practice this is impossible for arc furnaces, and in all cases parts of the upper furnace lining are more or less exposed to the directly radiated heat of the arc itself. Under the best conditions such heating is only local, being confined to the lower portion of the walls above the slag line, and nearest the arc. Heat is also reflected upwards from glassy, thin slags, and may at times be sufficiently intense to cause fusion of the brickwork.

It can be broadly stated that the intensity and distribution of heat in all arc furnaces, other than the largest sizes, is injurious to the roof and wall lining, and for this reason the refractory materials require to be most carefully chosen and utilised.

(*b*) Sudden and considerable variations of temperature are equally important in their effect upon refractory materials, and in this respect the electric furnace, when used for melting cold scrap, is at a disadvantage in comparison with other steel-melting furnaces. The door area is relatively large for the purpose of charging scrap, and the lining is very rapidly chilled during the interval between heats.

Owing to the serious influence of sudden temperature changes, the selection of suitable brands of basic, acid, or neutral bricks should be guided by their power to resist fracture under such variable conditions of service. Bauxite and certain brands of magnesite brick are well able to stand this treatment, whereas others fail.

(*c*) The distribution of heat radiated from the arcs will depend upon their number and position, and in every case the contour of the lining should be so designed that no part of the brickwork is unduly placed under the influence of the intense arc temperature. The shape of the furnace body is therefore dependent upon the number and disposition of the arcs; when the principle of direct arc heating is used, a circular form is favoured for either single, three, or four arcs, and a rectangular shape for two arcs. In the case of indirect arc heating, the heating zones are located between the extremities of two or three electrodes, which in the latter case converge upon a common point; the shape of the body is circular or rectangular according to the particular arrangement of the electrodes. The uniform distribution of heating zones becomes an important factor in the design of large furnaces, owing to the desirability of always melting under the direct influence of the arc, rather than by heat reflected from over-heated portions of the walls and roof, and conducted through the charge; at the same time, uniformity of bath temperature is desirable even during the melting stage, and for these reasons furnaces having only two arc zones are unsuitable for large capacities. The position of the heating zones relative to the furnace lining is also of great importance, and should really serve as a starting point from which all other constructional details are evolved.

(*d*) Heat loss by radiation from the furnace body and electrodes demands careful attention in furnace design.

Metallurgical conditions demand that definite bath and slag temperatures shall be reached, and, so long as such temperature conditions are satisfied without introducing other outside influences, the several processes of steel-making are technically possible. To be economically possible is a different matter, and the rate at which the desired temperature is reached, and not the temperature itself, then becomes of vital importance. Power supplied in the form of electrical energy is converted into heat at restricted zones within the furnace, and, since electrical heating is the most extravagant form used in commerce, it must be used with the strictest economy possible, and every possible attention paid to heat loss. The heat radiated from any hollow body in which heat is internally developed, depends upon:—

(i) The temperature difference between the inner and outer surfaces of the containing walls.

(ii) The thermal conductivity and thickness of the material of which the walls are composed.

(iii) The total area of the external radiating surfaces.

Of the above conditional factors the first is fixed by the internal temperature required and that of the surrounding atmosphere, and is therefore outside the possibility of control; the total radiation loss as affected by the other factors is, however, capable of being to a great extent controlled.

The rate at which heat units pass at all points from inside the furnace to the outer shell is dependent upon the thermal conductivity of the refractory lining and its thickness. Suitable refractories for lining purposes cannot be chosen alone from the point of view of low thermal conductivity, so that the only means of raising the total thermal resistance is by increasing the thickness of the lining, or by inserting a heat insulating backing, consisting of infusorial earth or bricks made of this substance, between the lining and the furnace shell. By adopting such measures the heat insulation of the furnace may be improved, with a consequent lowering of the heat loss. There is, unfortunately, a limiting factor to this simple procedure. The lining of arc furnaces, other than the largest, is always more or less locally exposed to intense heat radiated directly from the arc, and for this reason such parts of the lining will be rapidly fused, unless the heat is capable of being withdrawn fast enough to lower the temperature of exposed surfaces. Therefore, careful heat insulation of small furnaces will reduce the radiation loss, but only at the expense of a shortened life of the lining. Increasing the thickness of furnace walls increases at the same time the area of the furnace body and consequently the surface of heat radiation; this further limits the thickness of linings for furnaces of small capacity and low power input.

From the standpoint then of heat economy the following points in the design of electric furnaces are worthy of note:—

(i) The shape of the furnace body and roof should approximate to a sphere, which, for a given internal capacity, has a smaller external surface than any other geometrical solid. Round or octagonal furnace bodies are fairly close approximations, and are better than a rectangular form.

(ii) The thickness of the lining should be chosen according to the distance between it and the arc zones; in large furnaces the silica lining is sufficiently remote to be uninfluenced by the intense radiated heat, and is only subjected to normal steel melting temperatures slightly above the softening point of the brickwork. In such circumstances the heat need not be rapidly conducted from the inner surface, and the walls may be usefully thickened without in any way impairing the life of the lining, which is then solely dependent upon other chemical and physical influences described above. The chemical influences are those already mentioned as being harmful and destructive to acid refractory materials.

(iii) The dimensions of the furnace body, which to a greater extent affect the radiation loss of small furnaces, will depend upon the necessary hearth capacity, the power input, the distance of the lining from the arc zones, and the thickness of the lining.

For furnaces of small capacity, such as three tons and under, it is customary to make the body as small as possible, and for this purpose to reduce the thickness of walls and their distance from the arc zones to a minimum. This can be understood, if it is realised that a small increase in the outer diameter of a small furnace means a considerable increase in the ratio of the radiating surface to the internal hearth capacity.

The best overall dimensions are therefore based on a compromise between radiation loss and the life of the lining. In some types the life of a lining is sacrificed for the sake of thermal efficiency, and it then becomes very questionable whether the small thermal gain compensates for the cost of constant repairs, especially in cases where the radiation loss under the worst conditions is small compared to the useful energy input.

Electrical Conditions.—The conversion of electrical energy into heat by the formation of an arc takes place in small zones, so that the heat is developed at a very intense temperature, said to exceed 3400° C. It is important that the furnace charge alone is submitted to such intense heat, a condition that is only possible when the arcs are shielded from the lining either by unmelted metal, or by the electrodes themselves. A short low voltage direct arc, shielded by a large electrode, produces the least effect on the lining, whereas long high voltage arcs, striking from small pointed electrodes, lead to maximum erosion. During the greater part of the melting down period, the arc will be partially surrounded by a wall of unmelted scrap and entirely shielded from the lining. Long arcs are permissible under these conditions, and are in fact preferable. For this reason it is now customary to provide means for altering the arc voltage at will, which enables a high voltage long arc to be used during the melting period, and a low voltage short arc at times when the walls and roof are exposed.

The choice of arc voltage is a matter of divided opinion, some designers preferring a much higher voltage than others, owing to the various advantages to be gained. The question will be better understood by enumerating the various advantages and disadvantages of short and long arcs.

Low Voltage Arcs.—Short low voltage arcs are in closer contact with the charge, and the heat is therefore better absorbed by the metal and slag, which favours a longer life of lining and improves the thermal efficiency. Certain slag reactions, notably those produced by a reducing basic slag, are better promoted by a large and well-shielded arc zone, as these reactions are favoured by high arc temperatures. According to modern practice, low arc voltages vary between 60 and 40 volts, the higher figure in this range being used for melting and the lower during refining operations. Low arc voltages involve no risk of serious injury to the furnacemen, as the maximum open circuit line voltage, corresponding to three-phase arcs (star connected) of 60 volts each, will usually amount to only about 110 volts; in the case of two-phase furnaces the open circuit voltage would be still less for similar arc voltages. When working with acid slags, which have a high electrical resistance, the arc voltage should never be less than 60.

Since the heat developed by any arc is proportional to the product of current and voltage, then for the same power input a short low voltage arc will require a proportionate increase of current (assuming the same power factor); this necessitates more expensive electrical plant and equipment, larger electrodes, weaker roof lining construction, and a more expensive and heavier electrode regulating gear. The amount of power dissipated by a short arc is very sensitive to slight variation of its length, so that the load variation is likely to be rather more pronounced when melting scrap.

High Voltage Arcs.—Long high voltage arcs are injurious to furnace linings, and, owing to the rapid fusion of silica bricks, make it difficult to carry out chemical operations which depend upon the maintenance of a highly basic slag. Roof and wall renewals are unavoidably frequent, which constitutes the main objection to the use of long arcs.

High voltage arcs, whether direct or indirect, operate at 110 volts or more at full load, and where considerable reactance is introduced, the arc voltage will increase to 200 volts on open circuit in many cases. Such a high voltage is a source of danger to personal safety, and considerable care has to be exercised when performing any operation in the immediate vicinity of the live electrode circuits.

There are, however, certain advantages to be gained by the use of high voltage arcs. They require less current for the same power, which entails considerable saving in the cost of electrical equipment, as also for the electrode regulating gear. Owing to the length of a 100 volt direct arc, which, when carrying 3000 amperes, is roughly about two and a half to three inches in a hot furnace, a slight lengthening or shortening will have no considerable effect upon the current, as the variation of the resistance thus caused is small. For this reason a steadier load and a better load factor are more easy to maintain after pools of metal and slag have once formed beneath the electrodes. The use of smaller electrodes reduces electrode consumption per ton of steel, and has an added advantage from the point of view of roof construction.

At the present time low arc voltages are generally favoured, but excellent all-round results have also been obtained with furnaces melting on the high voltage principle under favourable conditions of operation.

Conductors.—The design of those portions of the electrical circuits carried on the furnace body demands consideration. Heavy currents up to 10,000 amperes per phase are commonly carried, and the magnetic fields set up by alternating currents of such magnitude are considerable. It has been explained how a reactive voltage is self-induced in such circuits, the effect of which is greatly magnified if iron or steel is introduced into the magnetic field. Apart from increasing the reactance of the circuit, currents will be set up in the steel itself, and result in local heating, which in the case of very heavy currents, becomes a serious source of trouble in constructional design. Conductors, for this reason, are best supported and guided by insulated bronze brackets bolted to the steel framework, and all steel constructional parts are so designed to avoid making any complete steel circuit around any one set of conductors. Furnace conductors are usually built up of heavy flat copper bars, which, when bolted together, are sufficiently rigid to resist the pull of the flexible conductors, either when the furnace is being tilted, or when the electrodes are being raised or lowered. Cables are sometimes used in place of copper bars, but only where they are unexposed to high temperatures, otherwise the copper wires in time become brittle and oxidised and finally break. All copper connections are best made between machined and tinned surfaces bolted together by bronze bolts. The terminal connection between the copper bar conductors and the flexible cables is often a point of weakness. The simplest method of securing a good connection consists in clamping the cables firmly to the conductor bars by rigid bronze plates, which are grooved to prevent lateral slip on movement of the cables; these clamps should be frequently examined, and any slack, caused by spreading of the cables between the plates, taken up. When socketed cable lugs are used great care must be taken to secure a perfectly sweated joint, and it is also advisable to pass a set screw through the wall of each thimble or use some other locking device as a further precautionary measure. Under no circumstances should sweated socket joints be used, unless only a moderate current density through the conductors, cables, and especially their joints, is allowed for. Local heating, when once it occurs, leads to considerable trouble with this type of connection, which, however, is the most convenient and practicable if properly designed and made.

The length of conductors should be the minimum possible, so as to reduce copper resistance losses, besides lowering their initial cost. Lengthy conductors, carrying 6000 amperes and over, introduce considerable reactance into the circuit, particularly when they lie close to steelwork. This may be a good fault in some respects, but it is usually preferred to cut down this uncontrollable reactance as far as possible, and introduce external reactance coils of definite design. The furnace transformer sub-station is best located as near to the furnace as possible, the transformers themselves being arranged at such a height that their low tension terminals are at a convenient level to receive the flexible cables from the furnace. Electrode holders frequently serve as an integral part of the load circuit, and are then usually made of bronze and rigidly connected to the copper conductor bars. Sometimes they are constructed to grip the ends of the copper conductors firmly against the electrodes, and do not themselves carry any current. Furnace conductors are always exposed to radiated heat, and are usually designed for a normal current density not exceeding 1000 amperes per square inch. Should the conductors be very large and their skin effect considerable, the current density should then be reduced proportionately. With this current density the resistance losses are very small, and the circuit connections on the furnace will not be overheated, if properly made. All insulating bushes, washers and plates should be made of a material that remains unchanged at the high temperatures above and around the furnace, as constant trouble is caused by short circuits when the binding composition of the material softens.

Transformer Capacity.—The power capacity of steel furnaces is a matter that very considerably influences the cost of power used per ton of steel and the quantity of steel produced. It has been explained in Chapter VI. how the ratio between the power wasted as radiated heat and the useful power available for melting will affect the power consumption per ton of steel and the total output. Those arguments in favour of a large ratio apply in all cases where the maximum possible daily load factor and output from a given furnace is in no way limited by shop conditions. They are open, however, to modification in cases where a furnace, having a required holding capacity, is installed to operate intermittently, and therefore at a low weekly load factor. Under such conditions, it may be an advantage to reduce this ratio by providing less power than usual: this procedure, of course, increases the power consumption per ton of steel, but, at the same time, lowers the maximum demand or flat rate payment to an extent that will show a small advantage on balance. It has been already shown (Fig. 77) how the cost per unit rapidly rises under a reduced load factor, and for this reason, if the maximum demand or flat rate charges were lowered, with a corresponding rise in monthly load factor, the net result would be a saving in the power bill, notwithstanding a small increased power consumption and lengthened heats. The tendency is, therefore, to provide rather less power for furnaces of a given capacity operating intermittently and at very low daily load factors, as this, besides reducing the power bill when partly charged upon a flat rate or maximum demand rate, also reduces the initial cost of the furnace plant.

Heat Conversion of Electrical Energy.—Direct arc furnaces may be divided into two distinct classes, according to the manner by which the electrical energy is converted into heat.

I. Furnaces in which a conductive hearth, or metallic conductors imbedded in the hearth, become an integral part of one of the load circuits, which is generally a neutral return conductor.

II. Furnaces in which the heat is developed in direct arc circuits entirely independent of the furnace lining.

Opinions are divided upon the relative merits of these two distinct types, and there is no doubt that equally good steel can be produced with either. From purely technical standpoints both designs have their relative advantages and disadvantages.

The several types of conductive hearth furnaces of small capacity, operated by either two-phase or three-phase current, are provided with only two electrode circuits. In comparison, then, with furnaces of the three-phase three-arc type of similar capacity, the two-arc furnaces should show a reduced electrode consumption, and have the advantage of a simplified electrode controlling gear. A more solid roof construction is also possible, the roof being usually arched about one horizontal axis in conformity to the rectangular-shaped body generally adopted. A rectangular form certainly causes a greater radiation loss for a given internal capacity than one approximating to a sphere, but has an advantage in reduced constructional cost and simplicity of design.

It has been repeatedly stated that a certain amount of heat is generated in a conductive furnace hearth by simple resistance, which greatly assists in the manufacture of alloy steels by preventing the formation of a frozen layer of metal on the bottom, due to the chilling effect of the cold alloys added. This bottom heating is usually equivalent to about 5 per cent. to 8 per cent. of the full power input, or 27 kw. to 43 kw. respectively for a furnace operating under a load of 600 K.V.A. at a power factor of ·9. If such a hearth is homogeneous and at a uniform temperature at any horizontal section, the current density and the resistance heating developed will be uniformly distributed. But, although these hearths are generally constructed of layers of material having a progressively lower conductivity towards the top, it is doubtful whether the greater part of the heat is generated in a region near the bath, since the conductivity of the top layers increases very considerably in proportion to that of the bottom layers at high temperatures. It is, therefore, impossible to know exactly in which part of the hearth heat is generated by resistance; it is equally clear that the total heat generated is not entirely absorbed by the bath of steel, but is partially lost as a result of increased radiation from the bottom shell plates. Should, however, the distribution of current through the hearth not be uniform, then there will be more intense heating at certain spots in the bottom, but this is not likely to occur if a well-constructed hearth is at a uniform temperature, as would be the case under a covering bath of steel. J. Bibby in a paper contributed at a joint meeting of the Institution of Electrical Engineers and the Iron and Steel Institute in 1919, has gone further than this, and gives his opinion that the bulk of the heat generated is dissipated by radiation outside the furnace, rather than any being absorbed by the metal. It seems very doubtful, therefore, whether the conductive hearth furnace can offer any advantage from the point of view of bottom heating over the top arc heating furnace. Steels containing 20 per cent. and more of alloyed metals can be regularly made in the latter type without any steel chilling on the bottom, the natural precaution of making small additions at a time followed by vigorous stirring, which is always necessary for mixing alone, being amply sufficient to secure this result.

Bath circulation or auto-mixing is another advantage claimed for these furnaces. Bath circulation can only be due to either heat convection currents or to electro-magnetic effects set up in the bath itself. It is obvious that a bath of steel heated by direct arcs will be hottest at the top, and, unless the bottom can be heated to a still higher temperature, convection currents cannot possibly be set up. Electro-magnetic circulation can in no way be due to the result of magnetic fields dependent upon the high permeability of iron, which becomes non-magnetic above about 750° C. Weak magnetic fields of varying intensity and polarity are, however, set up by fluid conductors carrying heavy alternating currents, and it is then theoretically possible for attraction and repulsion between different parts of a bath of metal to be caused by current traversing it in different directions. The mutually acting magnetic forces induced by solid conductors are visibly displayed by the movement of neighbouring cables of different phases carrying very heavy currents, but it is difficult to say whether sufficient forces are actually developed to cause and maintain movement of heavy masses of molten steel as a result of currents traversing different paths through the bath.

With certain types of conductive hearth furnaces considerable difficulty is experienced in securing a conductive circuit through the bottom, when cold. In these cases it is necessary to use auxiliary gas or oil heating, otherwise the furnace can only be operated under an unbalanced and diminished load. To obviate this difficulty various modifications have been introduced embodying the use of an auxiliary upper electrode, which is connected to the conductive hearth cables and can be used to complete the load circuits, thus enabling a balanced load to be applied. Under continuous operation, however, no difficulty need be anticipated through failure of the hearth to conduct the full circuit current a few minutes after applying load. Experience has proved that conductive hearth furnaces can produce excellent results, but, at the same time, the character of the hearth, owing to its use as an electrical conductor, is not so reliable under all conditions of service as those constructed of similar material but independent of the electrical circuits.

The main advantage of the other class of arc furnaces, apart from the ease with which they may at all times be heated electrically, lies in the durability of the hearth, which, if properly constructed, should never cause any metallurgical or other difficulties by premature and sudden failure under any normal conditions. The disadvantages are those due to the increased number of electrodes and their raising gear, higher electrode consumption, weaker roof construction, and the greater complexity of the load regulation of three-phase three-arc circuits. These disadvantages, of course, only apply when comparison is made with conductive hearth furnaces having not more than two top electrodes. Although this comparison is more especially applicable to direct arc furnaces alone, yet practically the same arguments apply when comparing conductive hearth furnaces with those of the indirect arc type.

Power Factor.—The question of power factor is of great importance in electric furnace design, and must be considered in its relation both to the reactance of the load circuits and to the inherent reactance of the transformers or generating plant. According to most power contracts the flat rate or maximum demand rate charge is based on a K.V.A. and not on a K.W. input, so that payment is made on a figure which does not represent the true maximum rate of power absorbed. Since the ratio of K.W. to K.V.A. is proportional to the power factor, the nearer the latter approaches unity, the better it will be for the consumer. At the same time, there is a clause in most contracts by which the consumer guarantees that the average power factor shall not be less than ·8 or ·85, so that careful attention must be given to the design of the transformers and the load circuits to the furnace electrodes. The relation of power factor to the capacity of transformers for doing useful work has already been mentioned, and determines the initial cost per K.W. capacity of plant installed in contradistinction to K.V.A. capacity, which does not truly indicate capacity for doing useful work.

The objectionable features of heavy load fluctuations, and the extent by which they may be reduced by reactance coils, has been fully dealt with in Chapter IV. Where, however, such reactance coils are introduced into the load circuits, they should be designed so as not to reduce the power factor seriously at or below normal full-load current, but only on heavy current overloads. Reactance coils designed to fulfil these conditions will considerably reduce heavy power or K.W. overloads.

Certain furnaces have been designed in which the amount of reactive resistance introduced into the load circuits is such that, at normal load, the power factor is about ·7. Under these conditions the reactive effect is so great that on dead short circuit the power in K.W. is considerably reduced below normal full load, and the normal full load current only increased by 41 per cent. In this way a practically automatic load control, resulting in excellent load factors, can be obtained at the expense of power factor.

Furnace design, from the point of view of power factor, will then depend to a great extent upon the limiting figure allowed by the power companies, the increased initial outlay for larger transformers, and the cost of electric energy as purchased on the basis of either K.V.A. or K.W. demand.

The power factor of furnaces will be influenced by the nature and relative disposition of the circuits between the transformer terminals and the electrodes. Any individual circuit carrying an alternating current is influenced by the magnetic field set up, but if two or more such circuits carrying currents that are out of phase are brought close to one another, the effect of the magnetic field set up by one will be partly counteracted by the magnetic field due to the others; for this reason the resultant reactive voltage induced in each circuit will be very much reduced, and the power factor considerably less affected. Therefore, when high power factors are desired in furnace construction, it is preferable to keep the several conductors close together, to avoid a closed iron circuit around any one set, and to support them as far as possible away from all steel parts. The length of a circuit also affects power factor by increasing the self-induction.

Mechanical Features.—*Furnace Mounting.*—It has been previously indicated that provision must be made for tilting electric furnaces owing to the necessity of skimming a bath and completely draining the furnace hearth. The earliest types of both arc and induction furnaces were fixed, but the necessity for tilting and emptying furnaces for steel melting soon became evident.

There are two usual methods of mounting furnace bodies for tilting:—

1. The furnace body is carried on rocker castings, which either roll forward on a flat base plate, or are supported on sets of rollers which allow the furnace to roll about a horizontal axis.

2. The furnace body is provided with trunnions mounted on trunnion bearings, and can be tilted by hand or mechanically driven gearing; this method is generally confined to furnaces of small capacity.

Tilting Gear.—When hydraulic power is available, tilting may be effected by one or two rams situated under the rear side of the furnace; this is undoubtedly the most reliable and least complicated method of tilting.

Various mechanical methods have been used, none of which can be said to give entire satisfaction. The tilting bar may consist of a heavy screw, fed forward or backwards by a rotating nut, or of a straight or curved rack engaged by a pinion; in the latter case the rack is fixed to the furnace body concentric with the rocker castings supported on roller mountings. The screw and nut method, which is almost universal in Great Britain, is likely to give trouble through failure of the screw thread or ball races, unless the gearing is carefully cleaned and greased at frequent intervals; this, however, is often neglected as the tilting gear is usually in a position that is not easy of access. A rocking arm or connecting link is also used for tilting; in this case a plain tilting bar is attached by a swivel joint to the furnace body, and connected at the other end to a crank, which is made to slowly revolve; this method is very simple, and has proved most satisfactory for small furnaces of 2 tons capacity and under.

In the case of any electrically driven tilting gear, limit switches should be provided to prevent the furnace being tilted too far either way, which might cause disengagement of the tilting bar or strain on the tilting mechanism.

Electrode Regulating Gear.—The gearing used for adjusting the electrodes is operated either electrically or by hand, provision being usually made for operating by either means at will, by introducing a simple clutch device. Various methods of gearing have been employed, which utilise either a rack and pinion drive, a nut and screw feed, or a simple rope and winch hoist.

The downward movement of any electrode should only be possible so long as the full weight of the electrode and its mounting is carried by the lifting gear, and, as soon as resistance is offered by the charge or furnace bottom to further movement of the electrode, the mechanical gearing should be thrown out of action; in this way no excessive strain can be thrown on to the gearing. The same applies to the limit of upward travel, and for this purpose limit switches and clutches are also used. Power driven gearing is frequently operated in conjunction with automatic regulators, and is then designed with a sufficient braking action to prevent any tendency of the motors to over-run, which would cause incessant hunting. The tendency to hunt is more pronounced where a rack and pinion lifting gear is used, but, on the other hand, a screw and nut feed is more liable to failure through troubles arising from wear of the screw threads. Electrode raising mechanisms are always exposed to heat and dirt, and should, therefore, be heavily constructed and enclosed as far as possible. Electrically driven gearing is necessarily heavy and cumbersome, and difficult to operate manually owing to its low mechanical efficiency, especially when using heavy electrodes. Air pressure has also been employed for adjusting light electrodes, but has not been developed to any extent. Hydraulic control is now being introduced in place of electrically driven gearing, and is being satisfactorily developed for automatic regulation.

The raising gear for all electrodes may be mounted together on one side of a furnace shell, or may be divided and attached to two or more sides. When this latter arrangement is adopted the raising gear is always set to one side of the plane of tilting, so that the furnace may be provided with pouring spouts and charging doors both back and front; this is certainly an advantage as all skimming operations can be performed over one spout, while the pouring spout is reserved for casting.

When the several raising gears are set side by side, they are either attached to the back of the furnace or to one side of the plane of tilting so as to utilise the double-spout construction.

Electrode Holders.—There is considerable variation in the construction and design of electrode holders, which may be roughly classified according to whether the holder (i) is itself the conductor, (ii) clamps the conductor to the electrode and carries its weight, or (iii) acts as a support for some independent clamping device which is really an integral part of the whole.

A holder belonging to the first class should be made of metal of high conductivity, be water-cooled or specially designed for air cooling, and possess sufficient flexibility to permit of rapid opening or closing with minimum risk of fracture. A maximum degree of flexibility has been obtained by hinging two portions of a holder together; the rigid portion is firmly fixed to the electrode arm, the hinged portion being thus alone free to move. Such holders are generally made of bronze, but steel has also been used successfully.

The same conditions apply to holders of the second class, only in this case there is no necessity for using a gun metal or bronze of high conductivity, since the holder itself is not called upon to carry current to the electrode. In the case of the third class the copper plate conductors are gripped to the electrode by an independent flexible clamp, which is supported by a fixed annular collar fastened to the movable carriage or gallows arm. The flexible clamp in this case is only under lateral tension and vertical compression, and is not subjected to any bending or twisting forces. The supporting collar, which actually carries the weight of the electrode, is not required to open and close, so that its construction can be greatly simplified. This division of a holder of the first class into three distinct parts certainly simplifies the construction of each part individually, but renders the whole less compact.

A perfect holder has yet to be designed which will combine flexibility, strength, electric conductivity, and rapid means of clamping, and at the same time preserve these characteristics under all normal conditions of working. The use of graphite electrodes greatly simplifies the construction of holders, and the required degree of flexibility is not sacrificed to the same extent as for large diameter amorphous electrodes where it is necessary to have great strength and rigidity.

Furnace Doors.—A furnace door of good design should combine as far as possible the following features: (*a*) It should be tight-fitting but allow the free escape of gases under pressure from the furnace interior; (*b*) it should be capable of easy and rapid movement; (*c*) it should be possible to open it slightly for inspection purposes; (*d*) it should prevent undue loss of heat through the door opening in the furnace lining.

The simple lift-up door which fulfils all these conditions has been severely criticised, but, if proper care is taken to keep the furnace door jambs in good condition, the electrode consumption due to in-draught of air is not materially increased. The various designs of close-fitting swinging doors are, from a mechanical point of view, quite satisfactory, but are not suitable as inspection and working doors, since they cannot be slightly opened for spoon sampling and other manipulative operations.

At least one door opening should be large enough for the removal of a full-diameter piece of electrode, and all others large enough for the purpose of charging uniformly and fettling.

CHAPTER XII.

MODERN TYPES OF ELECTRIC STEEL FURNACES.

The several types of electric steel furnaces now in use may best be studied in the order in which they have been successively introduced. In this way the introduction of novel features peculiar to any particular design will be more readily understood. It is also necessary to divide electric arc furnaces into two distinct classes:—

(*a*) Indirect arc furnaces.
(*b*) Direct arc furnaces.

Indirect Arc Furnaces.—This class includes all arc furnaces in which the arc strikes between electrodes, so that the furnace charge is entirely independent of the arc circuits and receives heat by radiation and reflection alone. This type was originated by Siemens, whose furnace is illustrated in Fig. 2.

Stassano Furnace.—Stassano was the first to use a single indirect arc for metallurgical purposes conducted on a commercial scale, and in 1898 built his first furnace, which was intended for the direct production of steel from iron ore. This furnace did not meet with economic success, so that Stassano ultimately modified its construction for melting steel scrap.

The outstanding feature of the modern Stassano furnace lies in the mechanical method of mixing the molten or semi-molten charge in order to utilize the heat radiated by the arc to the best possible advantage. There are also several less important features embodied in this design, which are nevertheless characteristic:—

(i) Fixed orientation and inclination of the electrodes.

(ii) Special hydraulic electrode regulating mechanism, operated by low pressure water circulating in cooling jackets, which carry the electrode holders.

(iii) The melting chamber is lined with magnesite bricks, and assumes the form of a hollow segment of a sphere, or of an ellipsoid for large furnaces (Figs. 102 and 103).

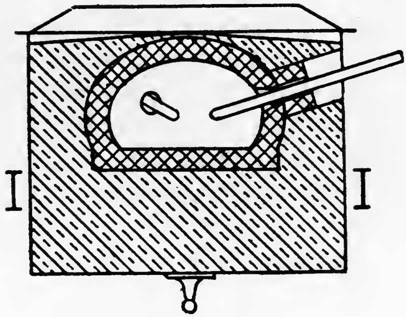


Fig. 102.

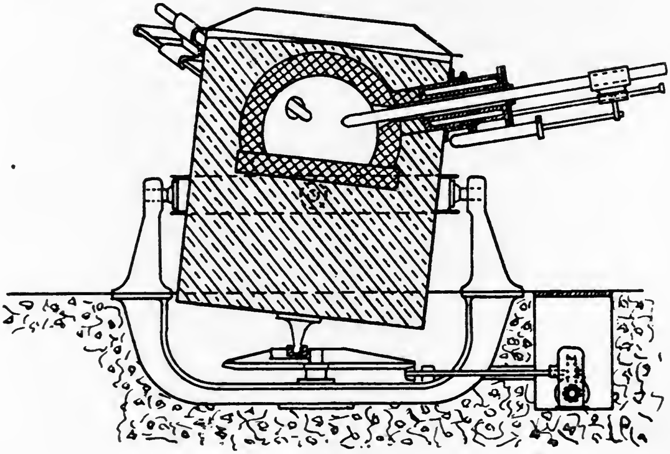


Fig. 103.

A vertical section of the furnace as shown in Fig. 103 clearly illustrates these special features of the construction. The furnace body is provided with trunnions, which rest on bearings carried by a ring encircling the furnace body. This ring also carries trunnions, which are supported on fixed pedestal bearings. The axes of the two sets of trunnions are set at 90° to each other, so that the furnace is free to swing in every direction just like a compass mounted in a gimbal. A pivot is fixed centrally to the underside of the bottom plate, and is displaced to one side of the normal vertical axis of the furnace by a large bevel wheel, which engages the pivot through an adjustable ball and socket bearing. The bevel wheel is axial to the trunnion ring and the normal axis of the furnace when vertical, so that rotation causes the pivot to describe a circle about the normal vertical axis, and imparts an oscillating movement to the furnace body; the degree of oscillation can be easily adjusted by altering the eccentricity of the pivot.

This construction is a considerable departure from the earlier types, in which the furnace body slowly rotated about a slightly inclined axis, a design which necessitated the supply of power to the electrodes through rubbing contacts.

The furnace lining is composed of magnesite brick, which is surrounded by a heat insulating backing of either brick or special refractory earths, the shape of the melting chamber being designed to reflect the heat downwards on to the charge or bath. The lining is naturally exposed to a very intense heat, and for this reason only magnesite brick can be successfully used, and then only when certain brands are available, which not only stand up to the intense temperature, but resist "spalling" to a most marked degree. A single charging door is provided, together with a small inspection hole, both of which are closed when highly reducing conditions are required. A closed taphole is used and any slag skimming has to be done through the charging door.

Two furnaces of the above described type were installed in the north of England for the manufacture of light intricate steel castings. They were of 1 ton capacity and designed for three-phase operation, each being equipped with a 300 K.V.A. transformer supplying three-phase current at a line voltage of either 150 or 100 volts. The high voltage was used more especially for melting, and the low after fusion of the charge was complete. Such high voltage arcs are always considerably drawn out, especially in a hot furnace, and care has therefore to be exercised when charging in fresh scrap, so as not to break an arc and thus interrupt the proper electrical conditions. The method of electrode mounting used on these particular furnaces, as shown in Fig. 103, enabled the electrodes to be rapidly removed or adjusted in their holders. Three cylindrical water jackets, fixed to the furnace shell at an inclined angle and converging to a common centre at angles of 120°, served as guide boxes for the electrodes. Each jacket also carried two projecting guide rods upon which the base of the holder was free to slide, the rods being so spaced that the axis of the holder was central to the cooling jacket. The holder was indirectly connected to the end of a piston rod, operating in a small cylinder fastened to the underside of the cooling jacket, and was thus capable of axial movement and rapid removal. These furnaces of 1 ton capacity are reputed to have made 80 to 85 heats before requiring to be relined, and used about 1100 units for each heat.

The furnace load is controlled with the aid of ammeters which indicate the current flowing through each electrode circuit, the electrodes being moved either inwards or outwards until the current flowing through each is at the desired value. If the arc between one pair of electrodes is shorter than either of the other arcs, then the current flowing through either electrode of that pair will be greater than the current flowing through the third. Balance of current, therefore, is only possible when the arc lengths are equal, and when the electrode tips form the apices of an equilateral triangle. The arcs themselves are mesh connected, so that the current flowing through each arc equals the line current ÷ 1·73.

If A is the current flowing through each electrode circuit and V equals the line voltage, then the power can be calculated from the equation—

$$\text{K.W.} = \frac{A \times V \times 3 \times \text{power factor}}{1{\cdot}73 \times 1000}.$$

Rennerfelt Furnace.—The outstanding feature of this furnace lies in a special arrangement of the electrodes, whereby the arcs are forced to take the shape of a flame that is strongly deviated downwards in the form of an arrow head. The heat is in this way more concentrated on those zones where it is required for melting and refining purposes, and, at the same time, the roof and upper walls of the lining are not exposed to the same intense heat of uncontrolled indirect arcs that always have a natural tendency to flame upwards. Unlike other indirect arc furnaces there is also a shading effect from a vertical electrode, which is to some extent comparable to that of a direct arc furnace. The arc zones can also be moved in a vertical plane, so that their distance from the charge can be kept constant as melting proceeds.

Electrical Design.—The furnace as generally constructed operates on a low tension two-phase system, the current being conveyed to the melting chamber by three circuits connected to adjustable electrodes. One circuit serves as a neutral return for the current flowing through the two phases, and is connected to a vertical electrode passing centrally through the roof. The outer terminals of each phase are connected to horizontal electrodes, the axes of which, together with that of the neutral electrode, lie in the same vertical plane. The arcs strike between the tips of the horizontal and vertical electrodes, and are deflected downwards by the resultant magnetic effect of the fields set up by each arc.

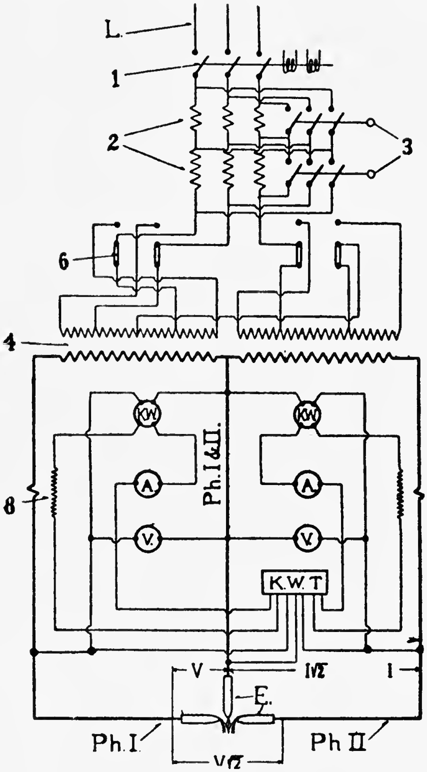


Fig. 104.

A complete diagram of the power supply, furnace, and instrument connections is shown in Fig. 104. The power supply is three-phase, as indicated by the three line wires L, which are brought into an automatic tripping oil switch 1. The high tension cables then pass to a set of choking coils 2, each of which is divided into two unequal parts in the ratio of 1 to 2; circuit breakers enable either portion to be short-circuited, so that three different values of choking effect may be obtained. These choking coils are frequently introduced into the low tension circuits. The high tension current is transformed down to a suitable voltage by a Scott-connected group of transformers 4, primary tappings and selector switches 6 being provided for secondary voltage variation. Auto-transformers installed on the secondary side of the power transformers have also been used for this purpose. The low tension circuits Ph. I. and Ph. II. are shown, together with the neutral return conductor Ph. I. and Ph. II. The current transformers 8 operate the various controlling instruments and the automatic regulators, if used.

The usual voltages available between the horizontal outer electrodes and the vertical neutral are 80 and 100. When choking coils are used on the secondary side, the voltage across each phase is about 150, which allows for a considerable arc voltage drop on normal full load. The load is regulated by moving the two side electrodes either towards or away from the vertical neutral electrode, which is always so adjusted that the tips of all three are in line. Electrode adjustment is effected either by hand or automatic control. When the load is equally balanced between the two arcs, the current through the neutral is 1·41 times the current flowing through each phase. The side electrodes are capable of being tilted downwards, so that it is also possible to strike two entirely distinct direct arcs on to a molten charge, provided the neutral electrode is likewise in contact with the slag or dips into it.

Three sets of bus bars are brought out horizontally from the transformer house at a point well above the furnace, flexible cables being then employed for connecting these bus bars to the three electrode holders. The transformer ratings for various furnace capacities are given in the following table :—

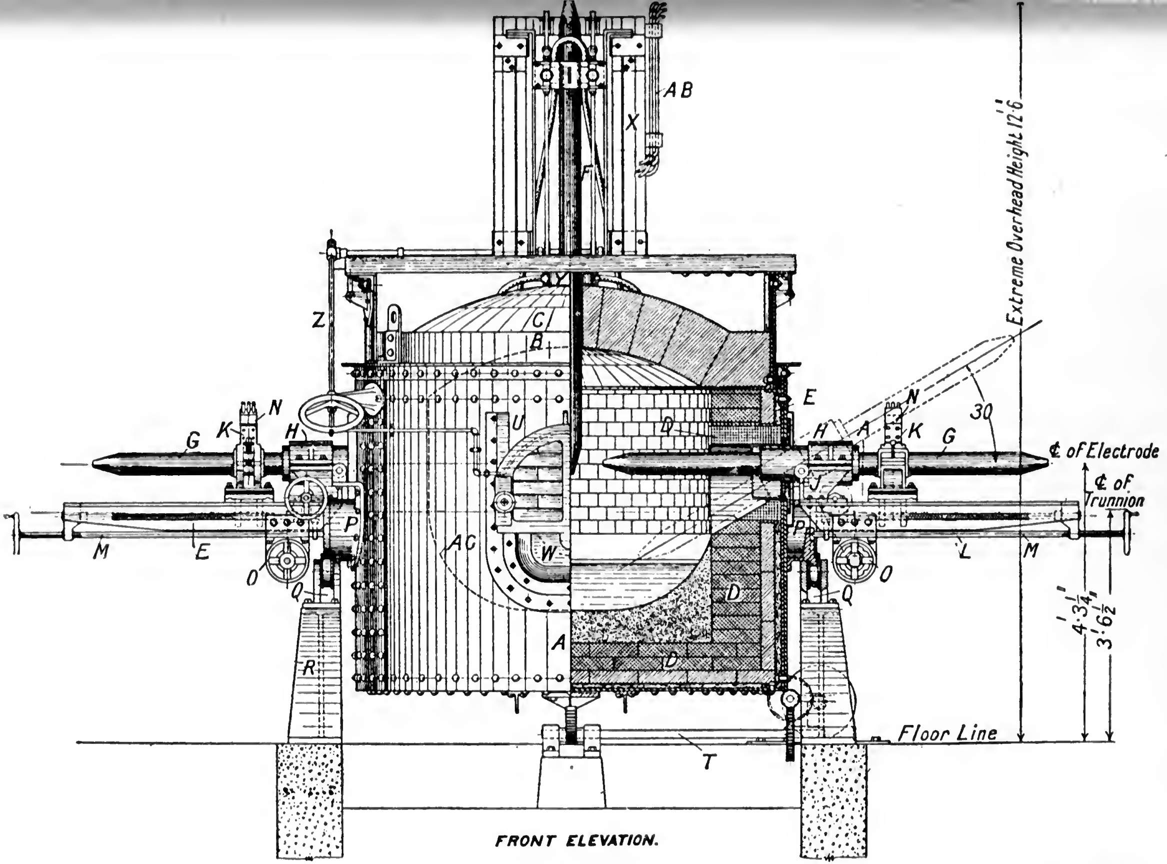
Furnace Capacity.	*Transformer Rating.*
4 cwts.	75 K.V.A.
7 ,,	125 ,,
15 ,,	250 ,,
1½ tons.	400 ,,
3-3½ ,,	800 ,,
4-4½ ,,	1000 ,,

The power factor is normally about ·90 at normal full load which allows for sufficient circuit reactance to prevent very heavy fluctuations or short-circuit currents.

Structural Features.—The furnace body of the most modern type is built in the form of a vertical cylinder with a flat bottom, and is covered by a detachable circular roof. The furnace is mounted either on trunnions or on rockers to permit tilting, which is done by hand in the case of the smaller sizes up to about 1½ tons capacity. A half section front elevation of a trunnion mounted furnace is shown in Fig. 105. The electrodes pass through cylindrical cooling jackets, which are pivoted on brackets fastened to the furnace shell. These jackets are also rigidly connected to the electrode carrying frame, which can be tilted by means of the hand wheels shown on the extreme sides of the drawing. The electrode holders are adjusted by a nut and screw feed, driven either by hand or motor as shown. The motors are either fixed under the carriers or on brackets bolted to the furnace shell.

The rectangular furnace (Fig. 106) has been designed for capacities of 5 tons and over, and embodies a complete duplication of the low tension furnace circuits operating in parallel. A furnace of this type of 4 to 5 tons capacity with a power input of 750 K.V.A. has been in use for two years at the works of Stridsberg and Biorck, at Tröllhatten, for making high class carbon steels.

Furnace Lining.—Acid and basic linings are both employed, the walls being in either case 14 inches thick, which includes a 4½ inch backing of fire-brick. A section through a rectangular basic lined furnace is shown in Fig. 107. The bottom is covered with two courses of fire-brick, above which is laid a single course of magnesite bricks placed on edge: the fire-brick is shown



FRONT ELEVATION.

Fig. 105.

[To face p. 240.

stepped up towards the sides and carried to the top as a backing to the magnesite and silica wall bricks. The magnesite bricks

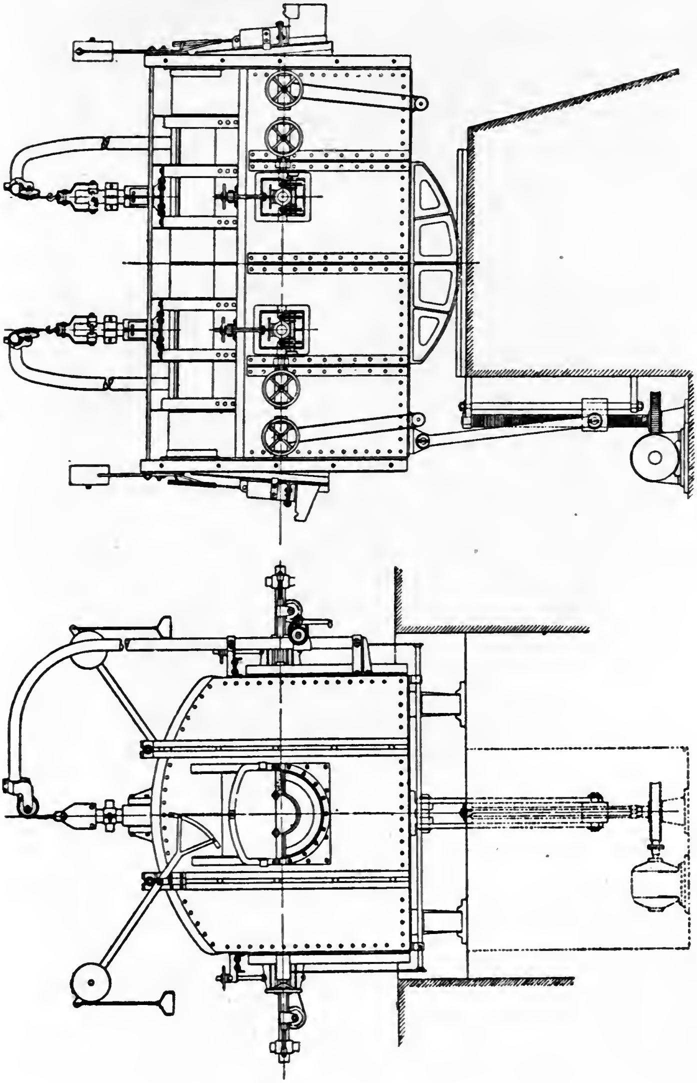


Fig. 106.

are also stepped up and carried to a point a few inches above the slag line, and from there the walls are built of silica bricks. The hearth is built of a mixture of calcined magnesite and basic slag, which can be sintered in layers by the heat of the arcs. In the modern circular form of body the walls are built up to the top of the steel shell, and form a seating for a circular domed roof, which is lined with special 9-inch silica bricks; the sintered hearth is also about double the thickness of that shown in Fig. 107. In the case of acid lined furnaces, silica brick and ganister are used in place of magnesite brick and the sintered basic hearth mixture.

Electrodes.—Graphite electrodes are preferred, and are generally loaded up to 150 to 220 amps. per sq. in. Small sized electrodes have, however, been loaded as high as 400 amps. per sq. in. The vertical neutral electrode is larger in diameter than the side electrodes, owing to the heavier current carried. The electrode consumption has been carefully ascertained in terms of lbs. consumed during each hour of operation under definite conditions. This is certainly a convenient and accurate way of expressing electrode consumption, which enables the consumption per ton of steel to be approximately estimated for intermittent or continuous operation.

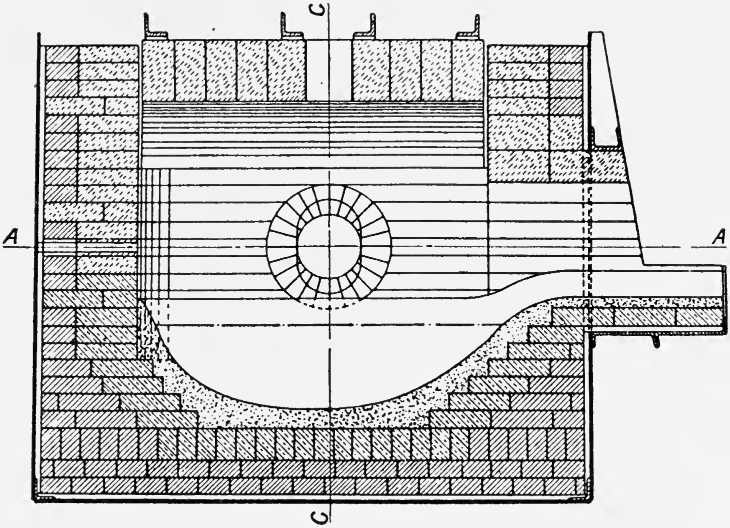


Fig. 107.

The following table gives actual figures of electrode consumption for furnaces operating with graphite electrodes of various diameters for the manufacture of tool steel and castings :—

Diameter of the Side Electrodes.	Diameter of Vertical Electrode.	Lb. per Hour.	Period Averaged.
1½ in. graphite	1½ in. graphite	·5	—
2 ,, ,,	3 ,, ,,	·66	1224 hours
3 ,, ,,	3 ,, ,,	1·5	several months
3½ ,, ,,	4 ,, ,,	2·36	—
5 ,, ,,	5 ,, ,,	2·2	480 hours
4 ,, ,,	5 ,, ,,	1·65	130 ,,
2 sets of 5 ,, ,,	2 sets of 6 ,, ,,	6·4	130 ,,

The Rennerfelt furnace, apart from its use for making tool steel and castings has been employed for melting ferro-alloys and special grades of pig-iron. It has found considerable favour in the United States of America and Scandinavia, but is at present not widely known in Great Britain.

Furnace Operation.—The furnace may be preheated simply by means of the free burning arcs, which can be kept at any desired distance from the bottom. The usual practice is to start melting a cold charge of scrap by means of the free burning arcs, which are gradually lowered as the charge melts. When the charge has been completely melted, and it is desired to obtain strongly reducing slag conditions, the side electrodes are tilted downwards until direct arcs strike on to the slag, the neutral electrode being at the same time lowered to make contact with the bath. Under these conditions the ordinary deoxidising and desulphurising carbide slag can be maintained.

Since the charge is quite independent of the arc circuits, it follows that the load is not so subject to fluctuation during the melting period as in furnaces of the direct arc type. The circuit reactance prevents heavy overloads when striking the arc, and tends to steady the current, especially when starting to heat up a cold furnace. The chief difficulty of manipulation lies in charging the scrap. Great care must obviously be taken to prevent the side electrodes arcing on to the charge when they are being tilted downwards as melting proceeds; the same difficulty applies when charging scrap into the bath. Heavy small scrap will give less trouble than bulky light scrap, which requires more constant feeding.

The Rennerfelt furnace has, so far, only been used for melting cold scrap charges up to 5 tons in weight. Larger units are being developed, which will be more suitable for refining liquid steel.

DIRECT ARC FURNACES.

Heroult Furnace.—In this furnace the principle of direct arc heating was first commercially applied to the metallurgy of steel. Direct arc furnaces had been used for many years before the introduction of Heroult's modified form, and were always provided with a carbon-lined bottom, which was connected to one terminal of a single-phase power circuit. It was essential to eliminate this carbon-conducting bottom to prevent carbon absorption by the steel, and this Heroult accomplished by splitting the single direct arc, as hitherto used, into two direct arcs in series, using the metallic charge to complete the circuit between the two arcs. This enabled any suitable refractory material to be used for the hearth lining, and burnt dolomite was chosen for that purpose.

Heroult's chief aim was to produce a furnace of simple design, in which the basic open hearth process of steel-making could be practised by merely substituting electric heating for gas. This aim was actually realised, but it was found too expensive to use the electric furnace for boiling out carbon from pig-iron for conversion to steel. For this reason its application was later confined to melting and refining mixed charges of scrap iron and steel, in which the carbon was not sufficiently high to prolong the refining operations. It was also found that internal electric heating enabled a highly reducing atmosphere, and consequently reducing slag conditions, to be maintained within the furnace, and this resulted in the discovery of further refining powers in the nature of sulphur and oxygen removal.

A bottom metallic electrode, imbedded in a refractory hearth, was also tried as a substitute for the carbon bottom and, although made the subject of a Belgian patent application in 1902, was abandoned in favour of keeping the arc circuits entirely independent of the furnace lining. This early decision of Heroult has been firmly upheld to the present day, so that the out-standing feature of the original Heroult steel furnace still remains.

The single-phase furnace design was not suitable for operation on polyphase systems, owing to the high cost of motor-generator sets coupled with their poor electrical efficiency. For this reason the furnace was redesigned to operate on a three-phase supply, being provided with three electrodes for striking star-connected arcs on to the metallic charge or bath, which serves as a star point.

Features of the Electrical Equipment.—Three single-phase transformers are connected in either delta-delta or star-delta fashion, and with this simple method of grouping, the transformers are identically the same, so that it is only necessary to keep one spare in case of emergency. The connections of the primary windings can also be made readily interchangeable from star to delta or vice versa, when considerable variation of the secondary voltage is desired. Flexible cables are taken direct from the secondary terminals, where the mesh connection is made, to the cable clamps attached to the furnace conductor bars, so that the least possible length of cable is used. The primary windings are always provided with one or two tappings for effecting voltage variation across the secondary circuits, this being done by means of special switches, as already described in Chapter IV.

The usual line voltages employed for basic working are 84 and 72 volts at normal full load, the open circuit voltage being somewhat higher according to the reactance of the circuits.

For working the acid process, a considerably higher line voltage is necessary, owing to the high electrical resistance of the siliceous slag, and for this purpose a line voltage of 110 at normal full load is usually provided, corresponding to an arc voltage of approximately 63 volts. The transformers suitable for various furnace capacities are generally rated as follows:—

Furnace Capacity.	*Transformer Capacity in K.V.A.*
10 cwts. . . .	200-400
1½ tons . . .	450-600
2 ,, . . .	600
3 ,, . . .	600-900
6-7 ,, . . .	1200-1800
10 ,, . . .	1800-2400 for melting cold scrap.

The power factor of the furnace load is invariably higher than the guaranteed figure of ·85, which some power companies demand, and installations are working for which the average monthly K.V.A. maximum demand is calculated on a carefully recorded average power factor of ·90 to ·92.

The power circuits are always designed with sufficient reactance to prevent very heavy current overloads, and when reactance coils are introduced into the low tension circuits they are designed to produce only a small reactance drop at normal full load, which, however, rises very rapidly on overloads. In this way the power factor is hardly affected at normal full load current.

The amount of current flowing through each electrode is indicated by an ammeter, three of which are usually mounted on a panel fixed to the back framework, just above the electrode raising gear. The panel also carries three lamps, each of which is connected between one set of cables and a common point connected with the furnace hearth. When these lamps are of equal brilliancy it is an indication of balance, since the arc voltages, and therefore the current through each electrode, must be equal to produce this effect. The luminosity of each lamp, being dependent upon the voltage between each electrode and the furnace charge, is bright unless the electrode touches the charge, when the lamp is extinguished. In this connection it should be noted that one electrode can be forcibly lowered on to a charge of scrap without causing any current to flow until one of the other two completes the circuit, and for this reason a lamp which indicates contact is an exceedingly useful accessory to an ammeter, and prevents breakage of fragile graphite electrodes. With a constantly breaking load they are also most useful, as they enable the electrodes to be rapidly adjusted with less risk of causing heavy overloads on again striking arcs.

An indicating wattmeter and voltmeter are usually mounted on a separate panel, which is hinged to a wall bracket and can be swung outwards into a prominent position. The wattmeter connections are made to current transformers placed in each electrode circuit, and to the three furnace conductor bars. The voltmeter is arranged to indicate by suitable plug connections the line voltage and any of the three arc voltages.

Besides the foregoing instruments, used for regulating the furnace load, a graphic recording wattmeter and an integrating watt-hour meter are generally installed, both operating off the low tension power transformer circuits.

Furnace Design.—The modern furnaces are three-phase and, except for the largest sizes, the same general design (Fig. 108) is adopted for all.

The furnace body is octagonal to conform as far as possible to the circular form of the melting chamber; at the same time this shape simplifies the construction of the doors and the attachment of the electrode columns and raising gear. The bottom plate is bent to a slight curve and is also more easily constructed than would be the case for a cylindrical furnace shell. The octagonal form of the shell for a given holding capacity reduces the surface of radiation to a minimum. The shell is bolted on to two rocker castings, which roll forward on a cast-iron bed plate on tilting, these rockers being rigidly braced together by two cast-iron separators and one steel casting to which the tilting screw is connected. Three door openings are provided, one at each side and one in the front wall immediately above the pouring spout, so that every part of the furnace hearth is readily accessible both for charging and fettling operations. The shell plates are strengthened at each door opening by a cast-steel stiffener through which the furnace doors are raised and lowered, the doors being suspended by chains from a rocking arm, pivoted on an angle support and balanced by counter-weights.

The steel framework which carries the electrode carriages and gallows arms consists of three pairs of channels, each pair being set with their flanges facing so as to form a long rectangular guide-box for two pairs of rollers attached, one at each end, to the steel electrode carriage. The latter is free, then, to move up and down between these channels with only sufficient lateral movement to ensure ease of working. Two projecting lugs are cast on the back of each carriage, between which a rack, guided by means of straps, is free to slide. The racks are meshed in with pinions which are driven through reduction gearing and strongly mounted on brackets attached to the back framework. In the event of the downward movement of the electrode being

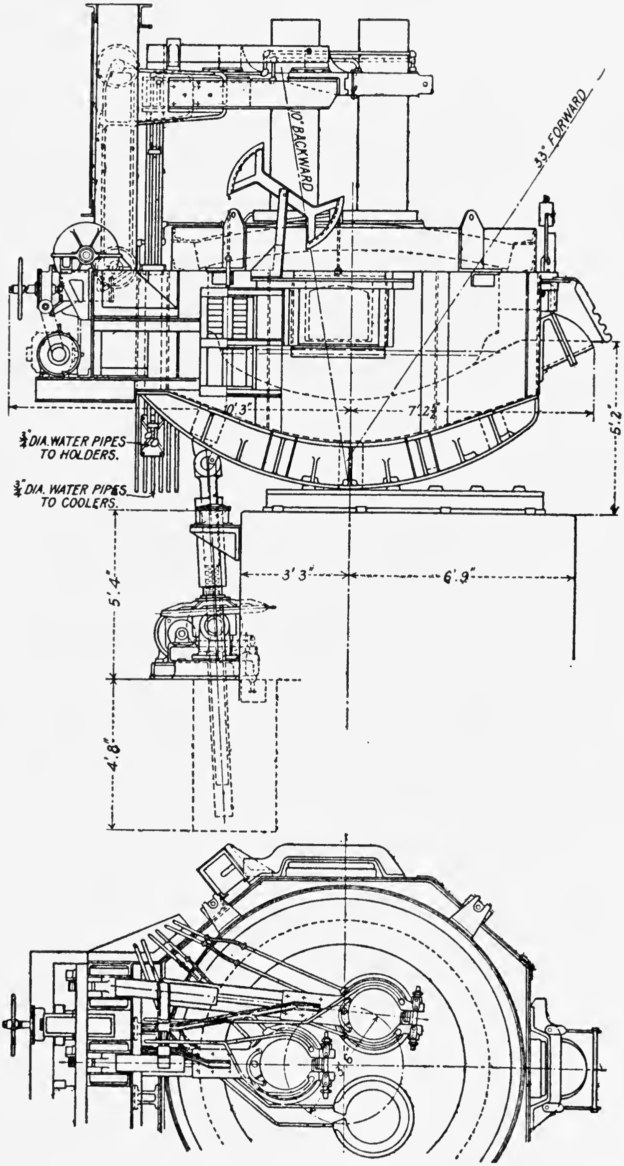


Fig. 108.—6-Ton Heroult Furnace.

resisted, the rack will no longer carry the weight of the arm, and, by moving away from the top projecting lug, opens a switch which automatically stops the motors. This device is a safeguard against damage to the electrodes or the raising mechanism. The steel electrode carriages are cast with short arms, to which are fixed extension pieces of steel channel carrying the holders, from which they are carefully insulated. The arms of the two outer carriages are slightly set inwards, so that the centre lines of the channel extensions pass through the centres fixed for the electrode axes; in this way the three holders can be made identical and therefore interchangeable.

A space is provided between the back shell plate and the channel-guide framework, which are rigidly connected together by steel plates at each end so as to form a narrow rectangular chamber. The conductor bars pass down through this chamber, and are guided by insulated gun-metal boxes, which bridge across both the top and bottom. With this construction there is no complete iron circuit surrounding any one phase, and the heating effect of eddy currents and a reduced power factor are avoided.

The electrode holder is built in two halves hinged together and water cooled, the water connection from one half to the other being made by means of a short loop of copper tube. A lug of ample dimensions is cast on to the rigid half of the holder, and is machine-faced for connection to the conductor bars. These bars are firmly held in place by insulated gun-metal brackets attached to the upper side of the electrode carriage. Water circulating pipes to and from the holders are clipped to the conductor bars, and terminate alongside the cable clamps.

The roof frame is circular, and when bricked up rests upon the body lining, four lugs being riveted to the framework for purposes of bolting down to angle plates fixed to the shell. Cast-iron coolers, split so as to break magnetic circuits, rest upon the roof brickwork, and are connected to water circulation pipes which are grouped close together and terminate in flexible hose pipes.

The tilting gear is of the screw feed type, which is clearly shown in Fig. 108. A heavy tilting screw is connected by a pin joint to the steel casting which separates the two rockers, and works in a heavy phosphor-bronze nut, journalled in a cast steel trunnion box. This nut is bolted to a large bevel wheel, which is rotated by a small bevel pinion driven by motor through reduction gearing. The end thrust on the nut is taken on a heavy ball race, another ball race being provided on the under side to prevent any possible axial movement in an upward direction. A telescopic dust guard covers the screw. This type of tilting gear should be cleaned, oiled, and greased at regular intervals to prevent excessive wear of the screw threads. This method of tilting is widely adopted for other types of furnaces in Great Britain.

The 1½-ton furnace, shown in Fig. 109, embodies the same general principles of construction as above described, but a special feature is introduced by the provision of two swivel arms for carrying the ladle. By this means an overhead casting crane in the furnace bay can be dispensed with. This arrangement is used in conjunction with a special bogie, which serves two purposes, according to whether ingots or castings are being made :—

(*a*) for the purpose of teeming ingots, the transfer bogie is mounted on rails, which are supported above and on either side of the ingot pit. The ingot moulds are set carefully in line with the teeming nozzle, and can be filled successively by carefully controlling the travelling movement of the bogie, which is effected by a spur wheel and pinion drive ;

(*b*) for foundry purposes the bogie is merely used for transferring the ladle from the furnace to the casting bay, where the ladle is then handled by a casting crane.

The method of using this ladle carriage for the transfer of the ladle to and from the furnace is as follows :—

The removable bogie rails which span the ladle pit are placed in position in readiness for pouring. The ladle, which is provided with extended double trunnions, is slung on the bogie and run up to the position shown in the figure. The ladle arms are swung inwards, and the furnace is slightly tilted backwards, so that the arms lift the ladle clear of the bogie trunnion bearings. The bogie is then moved backwards clear of the ladle pit, and the detachable pieces of bogie rails removed. The furnace can then be tilted forwards and poured, and again brought back to its original position. The rails are replaced,

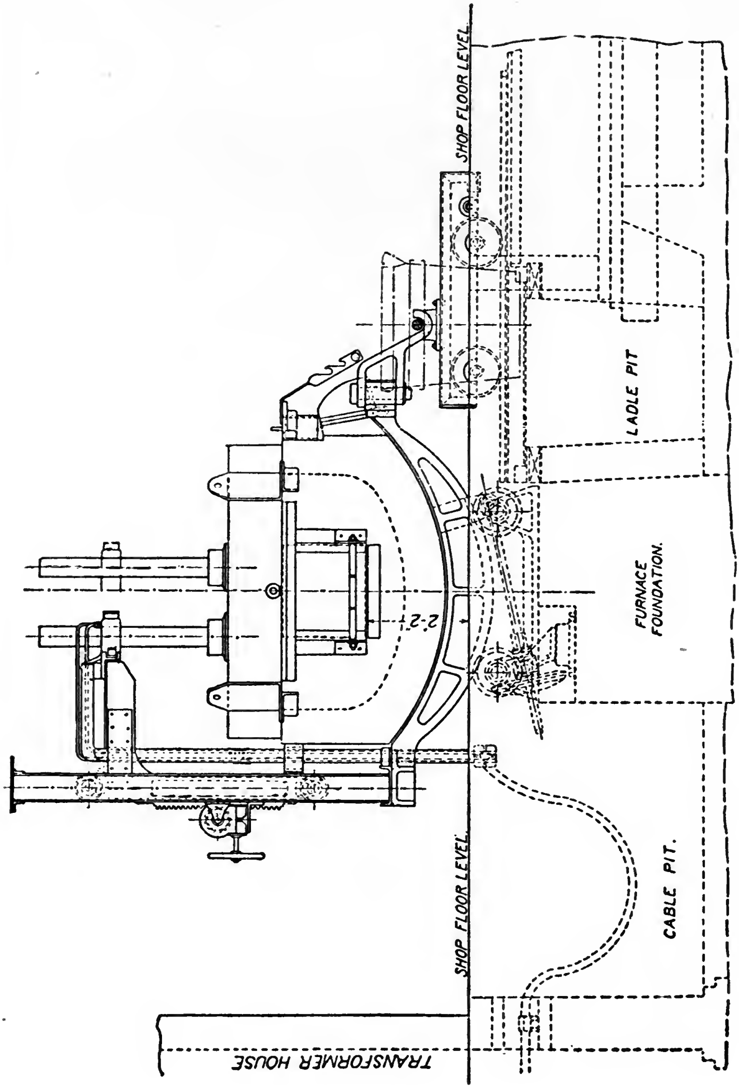


Fig. 109.—1½-Ton Heroult Furnace.

the bogie again brought up, and the previous cycle of operations reversed. The bogie then carries the ladle of steel and is free to serve either of the above purposes.

Furnace Lining.—The furnace is equally suited for either the acid or basic process, as the hearth is not called upon to carry any current. The methods adopted for lining with either acid or basic material are those which are fully described in Chapter XIV.

Electrodes.—Both amorphous and graphite electrodes are used, the latter being the more suitable for furnaces of 2 ton capacity and under. Amorphous electrodes are, at present, almost exclusively used for the larger furnaces, the diameters varying from 14 to 20 inches. Economisers of special design are described in Chapter XV.

Girod Furnace.—The original design of the Girod furnace was characterised by metallic electrodes, which penetrated the hearth and electrically connected the furnace charge to one of the line conductors.

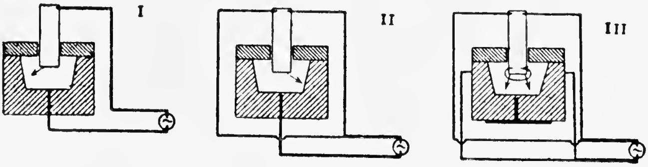


FIG. 110.

Electrical Features.—The furnaces are designed to operate on either single or three-phase low tension systems. The diagrams in Fig. 110 show three methods that have been used for supplying single-phase current to the furnace electrodes. In the first two instances the bottom electrodes were insulated from the furnace body, whereas according to the latest method they are electrically connected to the steel shell plates, to which is directly attached one set of the conductors. The small arrows indicate the direction in which the arc is deflected in each case, the deflection being due to magnetic fields set up in the steel shell by the heavy alternating current traversing the bus bars in close proximity to it. At the Gutehoffnungshütte the local destruction of the furnace walls was so considerable that it was eventually found cheaper to adopt the third and more symmetrical method of bus bar arrangement, which entailed the use of extra copper and resulted in a rather lower power factor.

The actual arrangement of the conductor bars is more clearly shown in Fig. 111. The bars are brought interleaved from the generator to a point "US" underneath the furnace, whence they are split into two separate sets; each set consists of conductors of opposite polarity similarly interleaved, which are carried up to a point level with the rolling axis of the furnace. Here, either cables or flexible strips are used for making the short connections both to the electrode bus bars and to the furnace body. The steel shell, below the point where the flexible connections are made, is under the influence of alternating currents of similar magnitude and opposite phase, so that the magnetic effects are neutralised. In this way, only very slight, rotating magnetic fields are set up around the

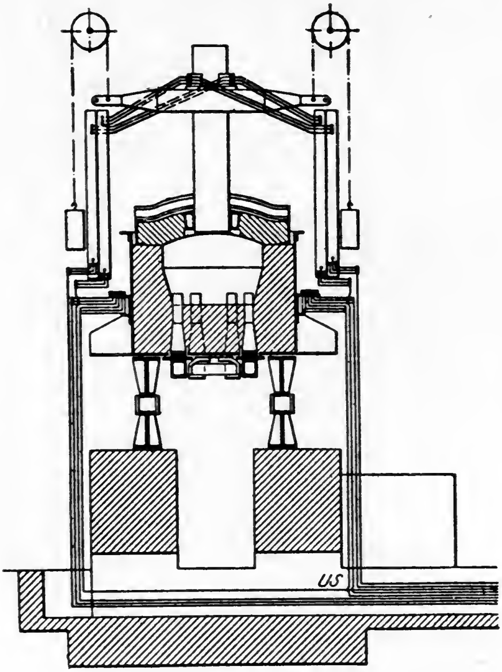


Fig. 111.

carbon electrode, which results in more uniform heating of the furnace charge and lining.

The system of connections used for the three-phase furnace is exceedingly simple. The three low tension phases are star connected, the outer terminal of each phase being connected to an upper adjustable carbon electrode, while the star point is connected to a series of metallic pole pieces fixed to the bottom plate and embedded in a conductive hearth. With this arrangement, when the load is equally balanced between the three arc circuits, no current will flow through the bottom electrodes and the return conductor; the phase or open circuit arc voltage is generally about 65 volts.

Furnace Design.—The 3-ton single-phase furnace, as used at the Gutehoffnungshütte, is shown in section in Fig. 111, the body in this case being square. The six bottom electrodes are electrically connected by means of a copper ring and plate with each other and with the furnace body. The furnace shell is mounted on rockers resting upon roller mountings.

The ratio of the cross-section of the steel electrodes to the rest of the bottom area is as 1 to 16. These electrodes are 4 inches in diameter at their upper, and 6¼ inches at their lower extremities, which project about 8 inches below the furnace bottom. The projecting portion has a cylindrical cavity 5½ inches long, through which water circulates to prevent excessive melting of the electrode at its upper exposed end.

[1] A 10-ton furnace, for melting and refining cold scrap charges, was put into commission at the works of the Bethlehem Steel Company, U.S.A., in 1916. Furnaces of this capacity are designed for three-phase operation and are constructed circular in shape. The furnace shell is 5 feet in depth and 15 feet in diameter. A large, single charging door, sliding in a water-cooled frame, is provided on one side of the furnace and immediately opposite the pouring spout. The furnace can be tilted either forwards or backwards, so that slag can be poured off through a notch in the charging door sill. Fourteen soft steel electrodes, about 3½ inches in diameter, are electrically connected to the furnace bottom plate, the lower ends being

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[1] American Electro Chemical Society.

water-cooled as usual. The electrode carriers are mounted in structural columns, which are fixed on opposite sides of the furnace and are raised and lowered by a screw and nut feed. Special care is taken to insulate the electrode bus bars, and to prevent induced currents in the shell and roof frame. The furnace is supplied with power from a group of three single-phase transformers having a total capacity of 700 K.V.A., each of which is protected by a reactance coil of 106 K.V.A. capacity.

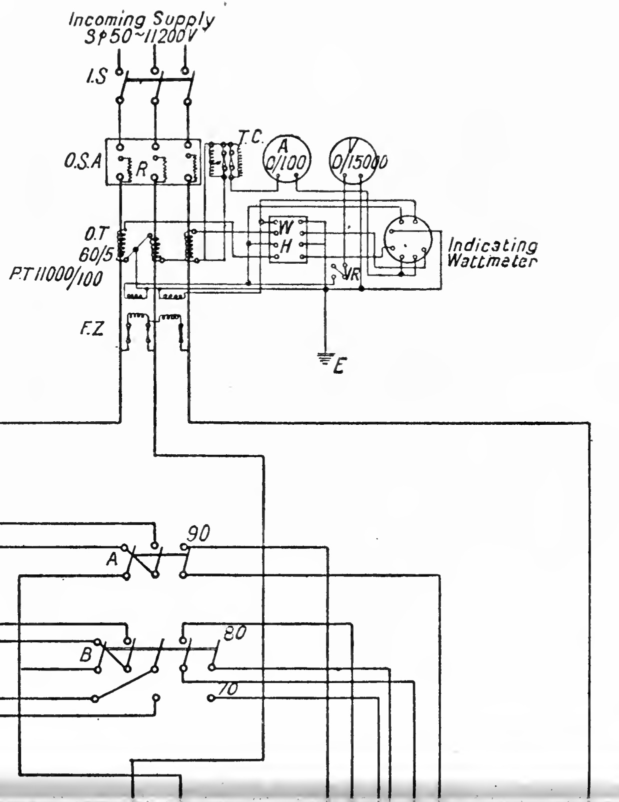
Furnace Lining.—The single-phase furnace referred to above was originally lined with magnesite, which was later given up in favour of dolomite for both the hearth and wall construction. When the furnace was used for liquid refining, the hearth and walls would generally last about 120 heats, the hearth, originally 18 inches thick, dropping about two inches during this period. The roof was lined with silica bricks springing from magnesite skewbacks to admit of its easy detachment from the dolomite walls, which was generally found necessary after 60 or 70 heats. The heat loss, due to water cooling the bottom electrodes, was carefully determined by measuring the quantity of water flowing and its temperature before and after passage through them. The loss was equivalent to an energy consumption of only 2·9 K.W. hours per ton of steel, and is small compared to the loss of 10·5 K.W. hours per ton of steel measured at the electrode cooling jacket. The amount of water required for cooling the bottom electrodes was only ·20 cubic meters per ton of steel as compared with ·65 cubic meters required for the top electrode cooling ring. With the proper degree of water-cooling the steel electrodes should only melt to a depth of about one or two inches below the hearth level, so that no serious erosion of the dolomite results.

In the larger three-phase basic furnace the wall lining is built of magnesite brick up to the roof, from which it is separated by asbestos plates; in other respects the lining does not differ from standard practice.

Electro-Metals Furnace.—The original feature of the Electro-Metals furnace was the application of a three-wire two-phase system of low tension connections. This design enables either two or three-phase high tension current supplies to be used without the aid of motor generators, which were formerly necessary for single-phase furnace operation. A conductive hearth is still an essential characteristic of this furnace.

Electrical Features.—The method by which two-phase current is supplied to this furnace has been fully described in Chapter III., and does not require further explanation. A diagram of the entire electrical equipment is shown in Fig. 112. Here, the two-phase low tension current is transformed down from a three-phase high tension supply by Scott-connected transformers. Tappings are taken out from the primary windings to give either 90, 80, or 70 volts across each of the low tension phases on open circuit, and the various connections can be made by two selector switches A and B, which are interlocked with one another, and with the main oil switch O.S.A. In the 7½-ton furnace rather higher open circuit voltages are used, namely, 100, 85, and 75. The neutral conductor cables are connected to copper bars placed side by side on a course of bricks laid on the bottom plate; details of this method of conveying current to the hearth are given in Chapter XIV. (Fig. 126). A furnace operating on a four-phase low tension system has been recently designed for large capacities. The special method of transformer grouping adopted for supplying such four-phase low tension current from a three-phase supply has also been dealt with in Chapter III.

The large power inputs required for furnaces exceeding 10 tons capacity cannot be satisfactorily carried by only two electrodes, to which number the two-phase pattern is limited, and it is chiefly for this reason that four-phase current requiring four upper electrodes is employed. This system requires five separate sets of conductors from the transformer group, four being connected to the upper electrode bus bars, and the other to copper bars imbedded in the furnace hearth in a manner similar to the two-phase furnace. The hearth is only called upon to carry rather more than the current flowing through any one electrode when all are equally balanced, and the current density is one-third of that of the two-phase three wire type. The load is controlled and balanced by ammeters, which register the current in each arc circuit. Voltage variation is also provided for by tappings taken from the high tension transformer windings in the usual way.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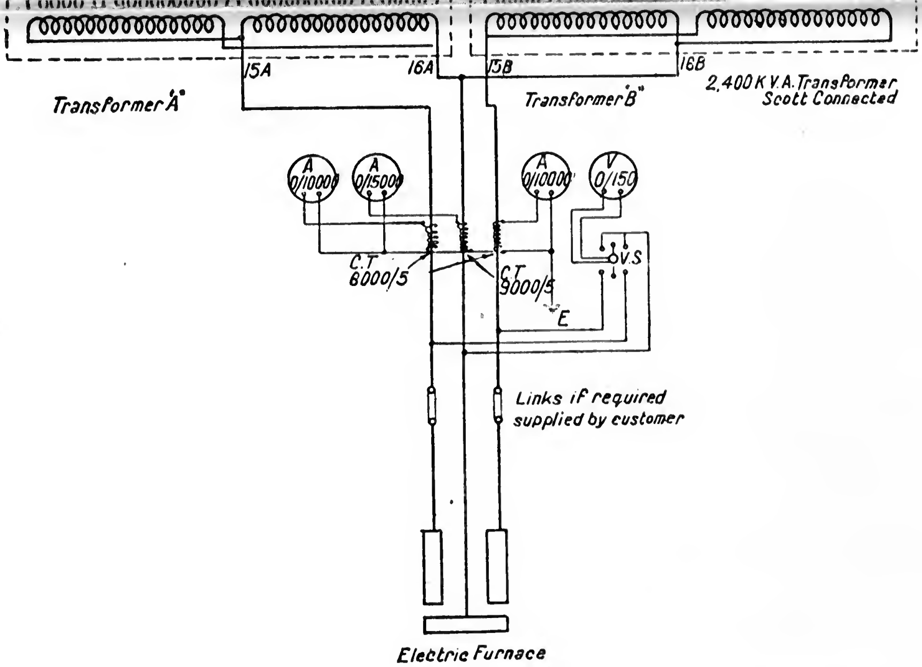

Fig. 112.

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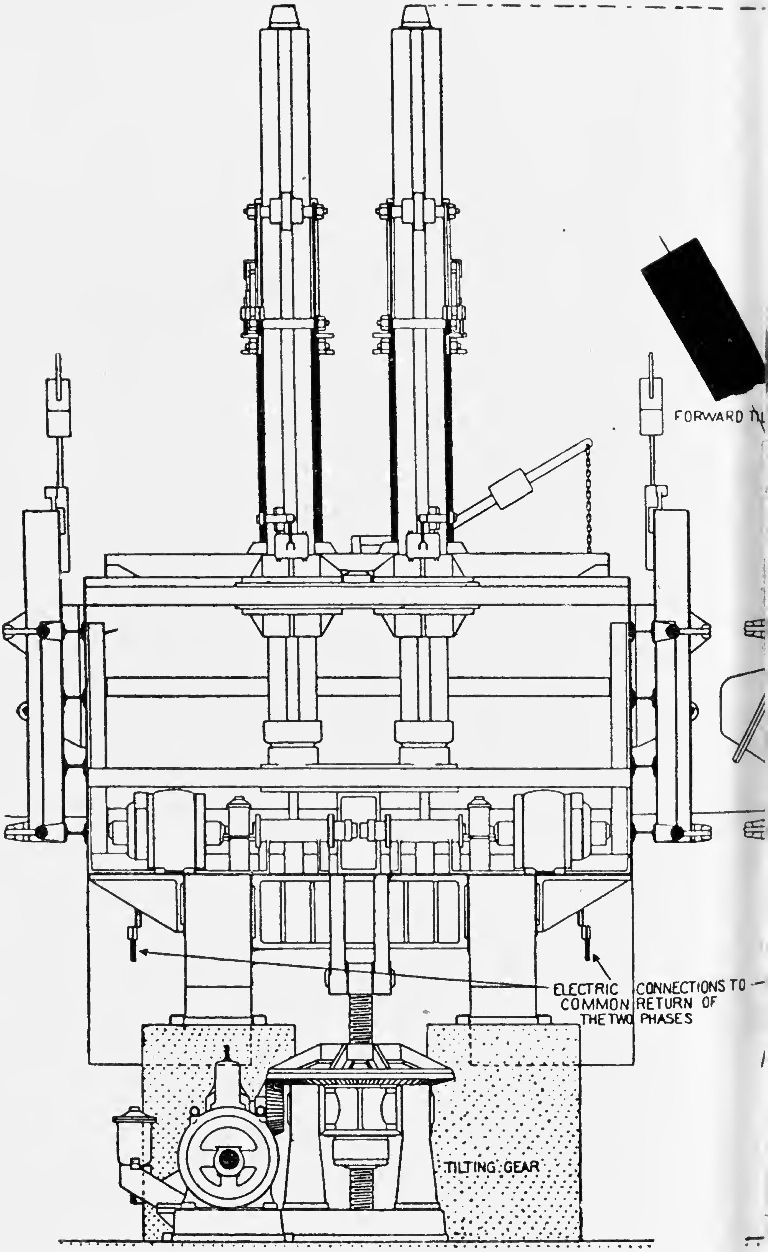
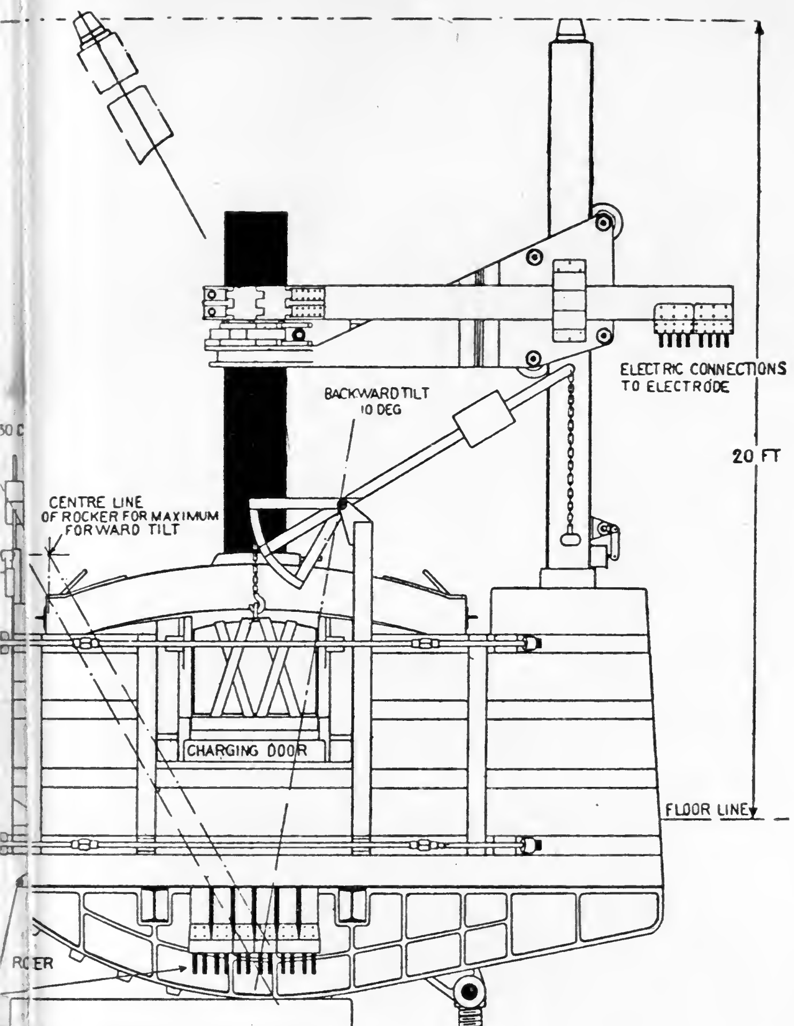

FORWARD TI
ELECTRIC CONNECTIONS TO COMMON RETURN OF THE TWO PHASES
TILTING GEAR



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Furnace Design.—The furnace body of the two-phase type is built in the form of a rectangular tank, which is supported on rockers and carries the necessary structure for guiding the electrode carriages. The construction of the 7½-ton furnace, shown in Fig. 113, is generally similar to that of the smaller capacities. Each of the columns, which serve as guides for the electrode carriages, consists of a pair of channels arranged so as to leave narrow openings between their inwardly facing flanges. The electrode carriages encircle these guide columns, and each one consists of two steel plates rigidly bolted together at four corners as shown; guide rollers are mounted on two of these bolts in such a manner that the weight of the arm always causes it to grip the column and allow vertical adjustment without lateral or other movement. A long vertical screw is mounted centrally within each column, and passes through a bronze nut fastened to the electrode carriage. Rotation of this screw, which is effected by hand or motor driven gearing, raises or lowers the electrode. A protective device is introduced to prevent damage to the raising gear in the event of an electrode being forcibly driven against a resisting obstacle. The electrode arm terminates in a collar, which supports the electrode clamping device and carries the weight of the electrode so held. Horizontal conductor bars are fastened to each electrode arm by insulated clamps, and carry suitably bent copper strips at one end and cable sockets at the other. The electrode clamp encircles the copper conducting strips, which are bent to the approximate diameter of the electrode, and consists of a small number of steel links hinged together and capable of being tightened up by means of a right and left hand screw. The connections from the horizontal conductors to the transformer terminals are made with flexible cables. The tilting gear and electrode regulating mechanism are shown in the back elevation drawing of Fig. 113. The rectangular roof frame is of very simple construction, being arched about the longitudinal axis only.

The four-phase furnace is illustrated in Fig. 114, which shows the disposition of the four electrode columns and motor platforms. The principle adopted for guiding the electrode carriages is very similar to that already described. The furnace shell is circular, but the relative position of the charging and slagging doors is unaltered. The bottom hearth connection is similar to that of the two-phase pattern.

Furnace Lining.—The Electro-metals furnace, being dependent upon a low hearth resistance, is always lined with basic material from the bottom to above the slag line. The method of lining furnaces of this type is fully described in Chapter XIV. A chrome brick parting between the magnesite bricks and silica walls is sometimes used. Furnaces from 2 to 7 ton capacity are provided with thick linings, the

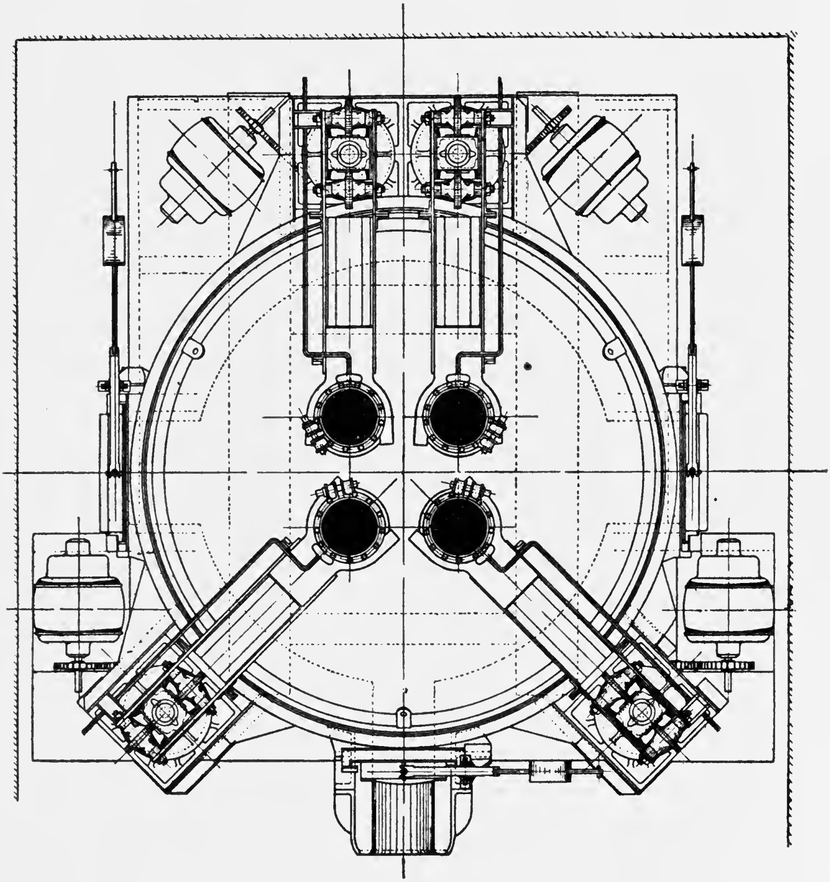

Fig. 114.

walls being 14 inches and the hearth 20 inches thick; these dimensions are also retained for the largest size yet designed.

Electrodes.—Both amorphous and graphite electrodes may be used, but the latter variety is generally more convenient owing to the high input which has to be carried by only two electrodes in the two-phase type. Apart also from the necessity of limiting the diameter of the electrode to conform to the internal furnace dimensions, there are, no doubt, other factors which have led to the general adoption of graphite electrodes. Economisers of special design are also used (see Chapter XV.).

Furnace Operation.—Melting operations can only be conducted under satisfactory electrical conditions, when the conductive hearth is capable of carrying practically the full return current of the two individual phase circuits. This is only possible when the refractory hearth is at a high temperature, and its conductivity thereby materially improved.

The method of heating used for baking the hearth of a newly lined furnace is fully described in Chapter XIV., and may be adopted for heating up an old lining after a prolonged stoppage. Gas or oil heating may be equally well employed when circumstances permit.

When cold charges are being melted, the scrap is charged in the ordinary way mixed with lime and ore, or mill scale. Arcs are struck by lowering the electrodes, and, if the bottom is sufficiently hot, the current flowing through the neutral return conductor, as shown by an ammeter, will be practically 1·4 times the current flowing through either of the electrode circuits when the latter are balanced. It will usually be found that just at first the hearth is not capable of carrying this proportion of current, and some operators then prefer to work at a slightly reduced load until the proper electrical conditions are fulfilled. When successive heats are being melted the hearth will generally carry the full return current after a few minutes, and it is only after stoppages that care requires to be exercised. Even though the current flowing through each electrode circuit may be the same, the primary supply circuit will not properly balance unless the neutral conductor is carrying the correct proportion of current; by reducing the load, however, to enable the hearth to satisfy this condition, correct balance on the power supply system is obtained. The indicating instruments, consisting of three ammeters, a voltmeter, and an indicating wattmeter, are mounted on a panel rather behind and to one side of the furnace. When the transformer substation is situated above floor level and close behind the furnace, the instrument panel is generally mounted on the dividing wall, and then also carries the selector and oil switches. Two tramway type controllers for operating the electrode motors are mounted at the foot of this panel, and can be conveniently handled by a furnaceman seated in front of them and facing the ammeters.

Automatic load regulation is generally adopted, each electrode being controlled by a regulator actuated by a current transformer in each of the separate electrode circuits.

Stobie Furnace.—The original Stobie furnace conforms in general principles to the Electro-metals design. In place, however, of the three-wire two-phase system of connections adopted in the latter, Stobie keeps the two low tension circuits distinct, thereby introducing a fourth conductor. The use of a conductive hearth and bottom electrodes has since been abandoned for furnaces of 6 ton capacity and over, the power being introduced by four upper electrodes, connected to a three-phase, and more recently to a two-phase system.

Electrical Features.—The arrangement of the low tension circuits of both the above-mentioned types has been explained in Chapter III. (Figs. 45, 49). In the four top electrode two-phase system, there are two arcs in series in each phase, so that the method of automatic regulation has to be based upon both voltage and current control. The reactance of the transformers and conductors is such that the power factor varies from ·84 to ·93 at different periods of melting and refining a charge of scrap. The average power factor is stated to be ·88 in the case of the large furnaces operating with four top electrodes.

Furnace Design.—The general design of the small two-phase bottom electrode furnace is indicated in Fig. 115, which does not, however, show the cable connections. The body is here shown mounted on trunnions, but is also constructed to be tilted on rollers by a motor-driven winch and wire cables. There is only one charging door, which is opposite the pouring spout. The electrode raising gear is a distinct departure from the more

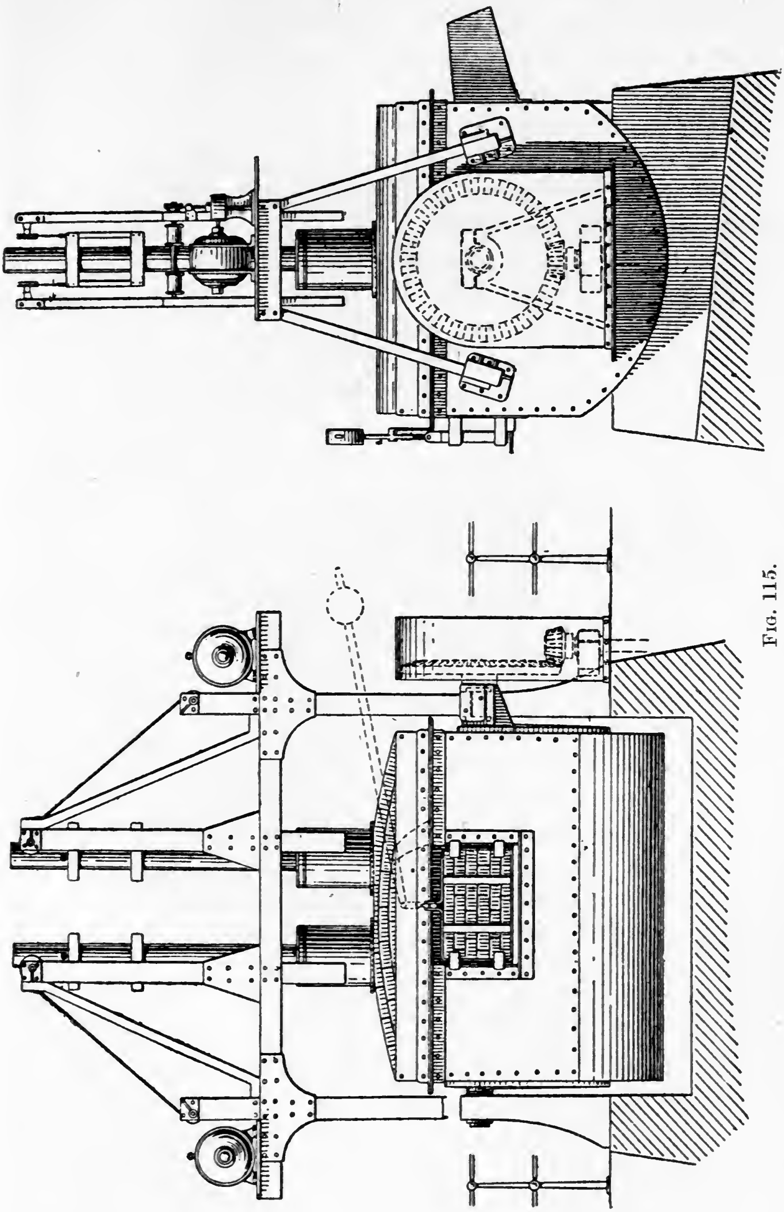

FIG. 115.

usual types, which embody guide columns, electrode carriages and arms ; in this case the electrodes are suspended from ropes, which are wound on to light winches mounted on small extended platforms, as shown. The point of suspension is considerably elevated to allow the necessary range of electrode travel above the economisers, which are about 2 feet high. The roof is arched about the shorter axis of the body, which allows more headroom for the door opening.

The larger furnace with four top electrodes is octagonal in shape, and is provided with three charging doors and a tapping spout. The electrodes are grouped in the form of a square, and are so arranged that each door opening faces the gap between any pair of adjacent electrodes ; this facilitates charging operations and general manipulation.

The same method of rope suspension is adopted for the electrode regulating gear, which, being symmetrically situated above the furnace shell, allows the above-mentioned arrangement of door openings to be adopted. Electrode economisers as described in Chapter XV. are always used, and are generally regarded as the most characteristic feature of the Stobie furnace.

Furnace Lining.—Furnaces of 5 tons capacity or less operate on the conductive hearth principle, and are consequently only suitable for basic linings. It has been explained that the two-phase low tension circuits are kept distinct as far as possible, and to effect this purpose a bottom electrode is embedded at each end of the dolomite hearth. The large octagonal furnaces with four upper electrodes may be lined with either acid or basic materials, since the hearth is independent of all electrical circuits and does not carry current.

Electrodes.—Stobie furnaces are designed for use with graphite electrodes, which considerably simplify the problem of gas sealing at the points where they enter the roof. It is also pointed out by Stobie that the electrical resistance due to skin effect increases with the diameter of the electrode, and that, although the relative resistance of amorphous carbon and graphite is as 4 to 1, the cross section of the amorphous carbon electrodes must bear a still greater ratio for similar carrying capacities. This theory is advanced as a reason for using graphite electrodes, but owing to the comparatively high resistance of both varieties the skin effect is not very pronounced.

Snyder Furnace.—The Snyder steel furnace, as first introduced, embodied certain electrical and mechanical features, which constituted a distinct departure from other then existing types. A high arc voltage and considerable circuit reactance were utilised to enable a steady load, and therefore a high operating load factor, to be maintained. The shape of the furnace body was designed to approximate as far as possible to a sphere, so as to reduce the radiation surface to a minimum for a given hearth capacity. Thick linings and close fitting charging doors of a special type were also used to obtain a high thermal efficiency. In the modern pattern even the side door openings are dispensed with, provision being made for charging scrap directly into the furnace body by introducing special means for lifting the roof after each heat.

Electrical Features.—Synder furnaces were originally designed for single-phase current, but are now also constructed for three-phase operation. The arrangement of the single-phase circuit is diagrammatically shown in Fig. 30. One terminal of a single-phase transformer or alternator is connected to an upper adjustable electrode, the other, which is earthed, being connected to a metallic electrode which penetrates the hearth. A choking or reactance coil is also incorporated in the circuit, and is so designed that, when the normal full load current is flowing, the reactive voltage is equal to the voltage drop due to the non-inductive resistance of the circuit. Under these circumstances the voltage between the top and bottom electrode will be reduced to ·71 times the voltage across the transformer terminals, and, as has already been pointed out in Chapter IV., it is impossible for the current to exceed 1·41 times the normal full load value, even on dead short circuit. If the current flowing in a circuit containing such a reactance coil is gradually increased from zero, it will be found that the voltage across the practically non-inductive portion, which in this case lies between the electrodes, will progressively fall, and, assuming the power factor of the circuit through the furnace to be known, it will then be possible to calculate the kilowatts delivered to the furnace itself for various current values. The curve A in Fig. 116

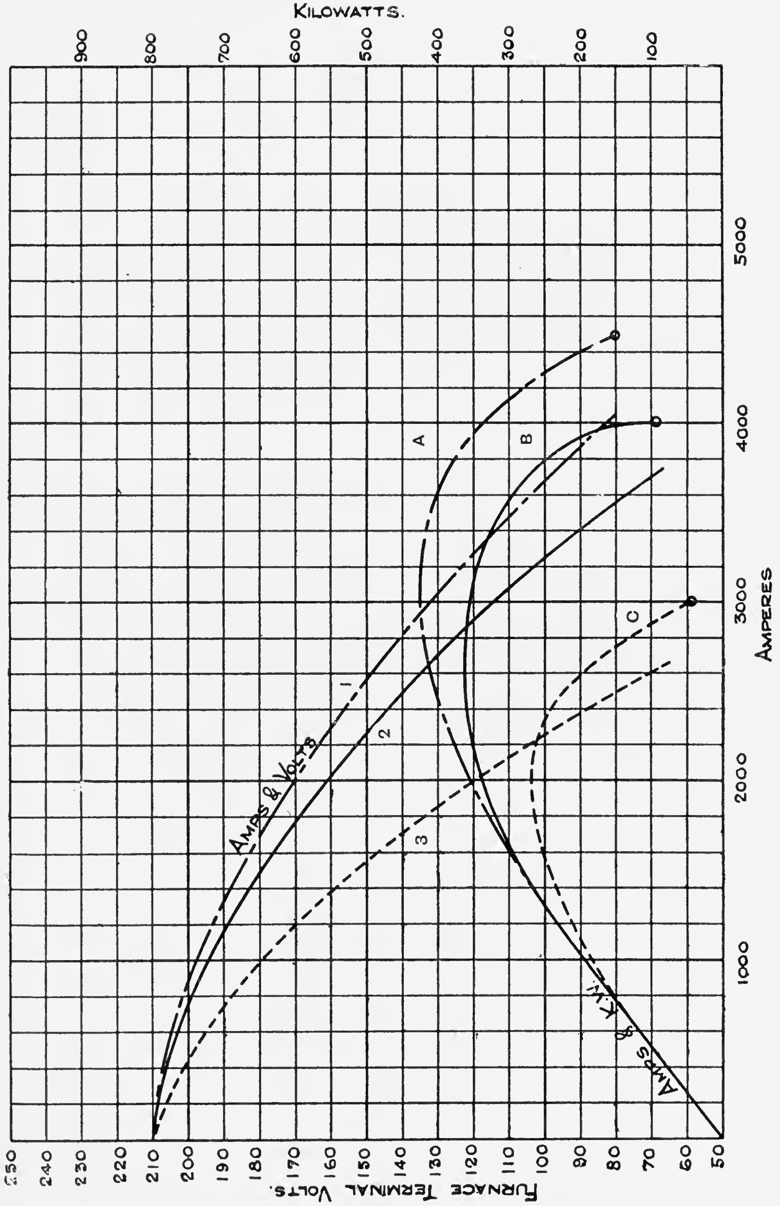
KILOWATTS.
900
800
700
600
500
400
300
200
100
AMPERES
1000
2000
3000
4000
5000
FURNACE TERMINAL VOLTS.
250
240
230
220
210
200
190
180
170
160
150
140
130
120
110
100
90
80
70
60
50
A
B
C
1
2
3
AMPS & VOLTS
AMPS & K.W.

FIG. 116.

has been plotted from actually measured kilowatt inputs corresponding to current values, ranging from zero to the maximum obtainable on short circuit. In this case the external reactance was introduced in the primary circuit of the furnace transformer in the form of a choking coil, which was provided with tappings, so that by means of a selector switch various degrees of reactance could be introduced. The curves B and C show the variation of power input for different current values after further increasing the amount of reactance. Curves 1, 2, and 3 were plotted from simultaneous readings of the furnace terminal voltage and current, and it will be seen that the product of two such readings gives a K.V.A. value about equal to the corresponding K.W., as actually indicated by a wattmeter. Further, the K.V.A. values as calculated, or the K.W. actually measured, are fairly constant within a certain range of current variation, and it is this special characteristic of such high reactance circuits that renders the power input capable of self-adjustment within certain limits of current variation. If during the melting operation the arc length is once adjusted so that the normal full load current flows, then, notwithstanding a slight change in the arc resistance due to alteration of the relative position of the electrode and the charge, the power input will remain practically unaltered. Partial short circuits caused by scrap falling against the electrode will be accompanied by a diminished power input, which is contrary to the effect produced in furnace circuits provided with only 12 per cent. to 30 per cent. transformer circuit reactance. A typical load chart is illustrated in Fig. 117, and shows the remarkable steadiness of the load, and high load factor rendered possible during the entire melting and refining operation. A further feature of the electrical design, conducive to steadiness of load, is the high arc voltage, which, at the maximum working load, varies according to the reactance of the circuit. In the case of the installation for which the curves in Fig. 116 were plotted, the open circuit secondary voltage across the furnace terminals was 220 volts; this voltage dropped to 135, 122, and 103 volts at maximum load according to the reactance introduced by a reactance coil provided with three interchangeable tappings.

On account of the high reactance used to stabilise the load, the power factor is considerably lowered and must be ·7, or less, at normal full load, if the special characteristics of load self-adjustment are to be fully realised. Automatic regulation, although by no means essential, is sometimes employed, and in that case relieves the furnace operator from even the occasional attention otherwise required.

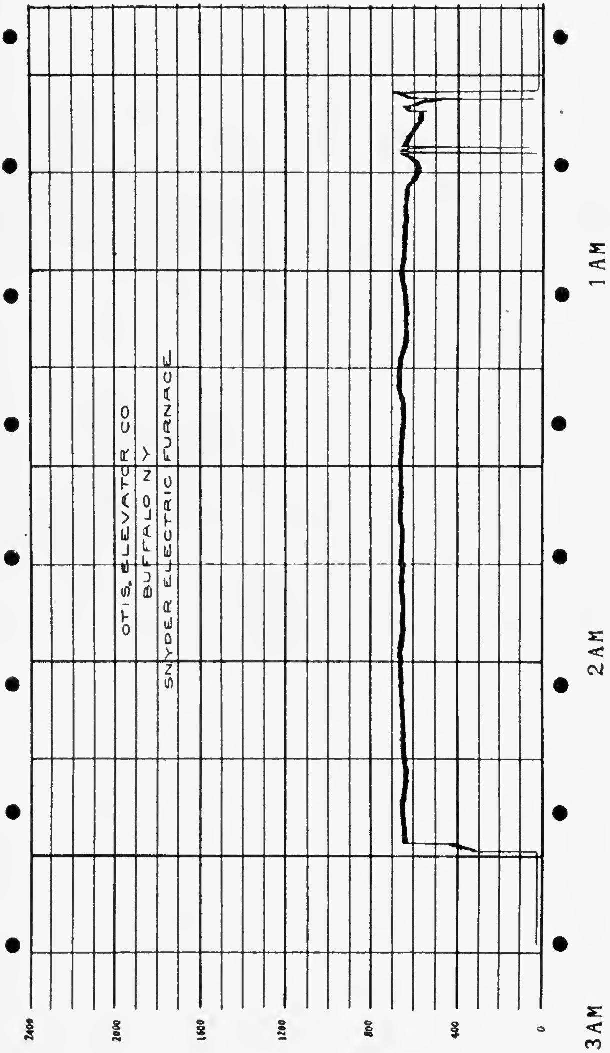

Fig. 117.

Three-phase current has also been used for operating Snyder furnaces. Two line conductors connect two terminals of the three-phase system to two upper carbon electrodes, a third conductor being used to make the connection between a bottom metallic electrode and the third terminal. With a simple star or delta grouping it is obvious that proper distribution of the load between the three-phases is impossible, unless the resistance in each circuit is equal, and for this reason some form of compensating device is no doubt introduced. In the Greaves-Etchells furnace a similar problem is encountered, and is overcome by adopting some special form of transformer grouping, whereby the suitable line voltage between different pairs of the line conductors is obtained.

Load variation over wide ranges is not such a simple problem as in the case of furnaces designed with only limited circuit reactance. In the latter case the circuit resistance may be increased by simply lengthening the arc, which causes a lowering of the current without any material rise of voltage, and thus reduces the load. This simple expedient does not suffice where high circuit reactance is introduced, since on lengthening the arc the current fall is accompanied, up to a certain point, by a corresponding voltage rise. Therefore, in order to effect any considerable reduction of load, it is necessary to lengthen the arc to such an extent that it becomes far less stable and has a more intense heating effect on the furnace lining. According to one method that has been employed for effecting variation of load, the furnace terminal voltage is reduced by introducing extra reactance into the circuit, and the three curves A, B, and C in Fig. 116 show the different loads at which it is possible to work for the particular case mentioned. With this arrangement the furnace may be operated under proper electrical conditions at either 420, 360, or 260 K.W., but a still lower load can only be obtained by lengthening the arc; other methods of voltage variation may also be adopted for effecting load adjustment.

Furnace Design.—From a purely constructional standpoint, there are two distinct designs of Snyder furnaces. In the more recent enclosed body type there is only one small front opening in the furnace lining for the purpose of pouring, inspection, bath manipulation and sampling; also, the entire charge of scrap can be introduced into the well of the furnace from above, owing to the special mechanical means provided for lifting the roof. A side elevation of a 5-ton furnace constructed on this principle is shown in Fig. 118. For furnaces of ½-ton capacity, and under, the same roof-lifting principle is adopted, but the

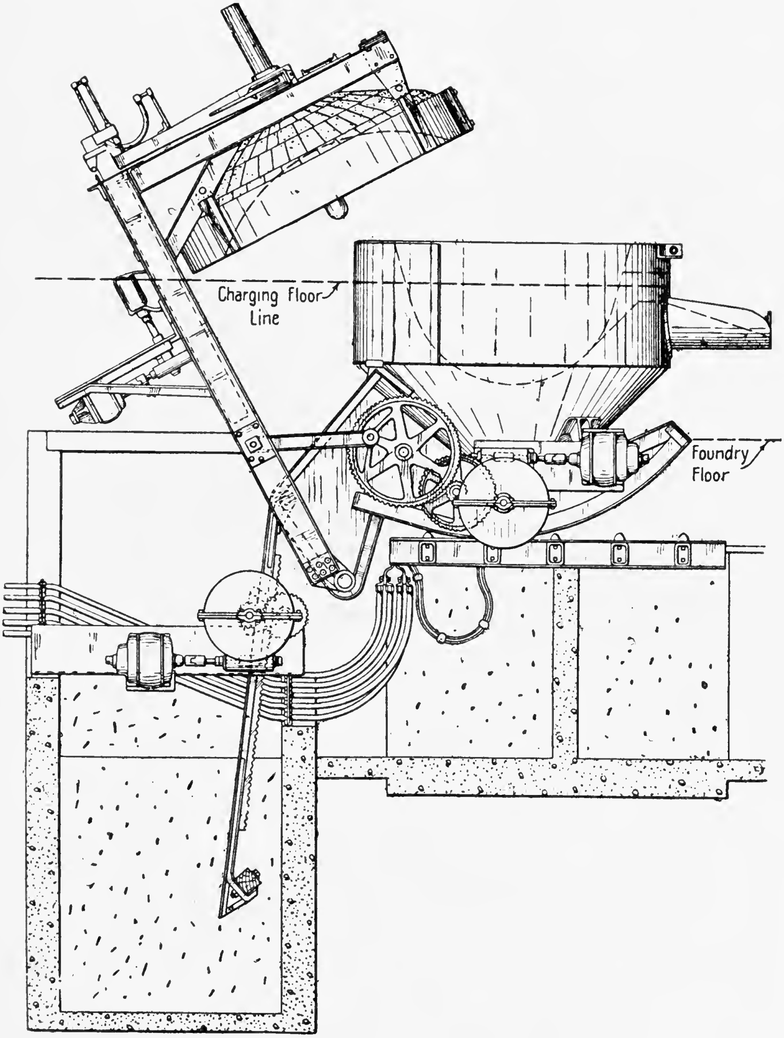

Fig. 118.

entire furnace is mounted on trunnions and designed for hand tilting.

The earlier type, in which two charging doors are provided, does not permit of trunnion tilting. The doors were originally introduced as one of the special features of the Snyder design; they are constructed in the form of thick, slightly conical brick plugs, held in a strong framework which fits tightly to the furnace shell. The door frame is pivoted about a vertical axis between two short horizontal arms, which can be swung outwards from the furnace body. This construction is not suitable for sampling and inspection purposes, as the plug is generally luted at its inner end after charging to effect a close seal; at the same time, it does not admit of being partially opened, and for this reason the spout opening is alone used for the general manipulation subsequent to charging.

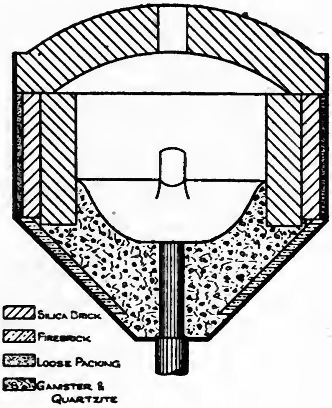

Fig. 119.

Furnace Lining.—Snyder furnaces, owing to the high arc voltage used, are generally more suitable for acid operation, although not precluded from the use of basic linings.

A section through an acid lined furnace is shown in Fig. 119; the working hearth consists of a suitable ganister mixture, which is made and rammed to shape according to the method described later in Chapter XIV.

When a basic lining is required, the circular wall of magnesite bricks is built up to a level slightly above the slag line, from which point the wall may be constructed by ramming "black basic" between the shell plates and a strong template; the working hearth is likewise rammed in the usual way, using prepared dolomite in place of ganister. In either case silica bricks are used for the roof construction.

Furnace Operation.—A method which may be employed for heating a newly lined furnace is described in Chapter XIV., although, more especially in the case of acid linings, such treatment preliminary to melting scrap is frequently dispensed with. It is customary to introduce the entire charge of scrap, either by lifting the roof or through the charging doors, before starting to melt, so that during the entire process of steel-making the furnace remains closed, with the exception of the one small opening above the pouring spout. After striking the arc the electrode is adjusted until either the correct arc voltage or current is obtained, either of which figures is chosen to correspond to the crest of the power current curves A, B, and C of Fig. 116. Melting will proceed up to a point without materially affecting the load, and only occasional adjustment of the electrode will be required before the charge is completely melted.

Special precautions are taken after pouring each heat to ensure a perfect circuit from the bottom electrode through the scrap to the upper electrode for the next heat. Any slag must be carefully scraped away from the small pool of steel, which marks the upper end of the metallic electrode, and then a cast steel spigot about 9 inches long is stood upright in the pool of steel until the latter solidifies. The whole operation must be done most expeditiously before the pool sets.

The wear on the roof and lining is naturally severe owing to the exposed nature of the high voltage arc, but is not so pronounced with acid linings owing to the shorter duration of each heat and the absence of basic dust-laden gases.

The excellent operating load factor made possible by introducing a high circuit reactance is a feature that naturally conduces to the maximum possible output of steel for a given available power input. Thirty heats have been regularly obtained during five days constant operation with a 2-ton furnace supplied with a maximum power of 420 K.W.; in this case the furnace was acid lined and used for making mild steel castings from good turnings scrap, while the power consumption averaged about 710 units per ton of steel in the ladle.

Greaves-Etchells Furnace.—*Electrical Features.*—It has been explained in Chapter III. that the characteristic feature of the electrical design lies in a special application of three-phase low tension current, whereby two-phases are connected to two upper electrodes and one-phase to a conductive hearth. If the three-phase system comprised three similar generating circuits, either simple star or mesh connected, then it would be impossible to obtain a balanced load, since the resistance in each of the line circuits would not be equal. The means adopted for overcoming this difficulty were also briefly indicated, and need not be further dwelt on, as full explanation could only be given by introducing a complex mathematical treatment of the problems involved. The current flowing through either electrode always traverses two of the low tension transformer windings in series. It is pointed out that, under such conditions, any sudden current overload will cause a momentary lowering of the power factor, which thus acts as a limiting factor to load fluctuation. Voltage variation is generally provided for in the usual way by transformer tappings, and load regulation is effected either by automatic or hand control.

The special three-phase system, embodying two top electrodes and a hearth connection, is adopted when the load capacity does not exceed 1300 K.V.A. When transformers of larger capacity are used, the above system is duplicated, and four top electrodes and one bottom connection are then required. In this way the power input can be greatly increasd without overloading the electrodes. The electrical power plant consists of two distinct transformer groups, each group being star connected on the secondary, and mesh connected on the primary side, the transformer ratios of each pair of windings being designed according to the same manner adopted for the smaller furnaces. Two such transformer groups may be combined in three ways, so that the current flowing through the hearth is either ·7 or ·35 times the sum of the currents in the four electrodes, or is reduced to nil. Without entering into a full explanation of the methods of combining the two transformer groups, it is sufficient to state that the two secondary windings connected to the conductive hearth are arranged to be either in phase, 120° out of phase, or 180° out of phase with one another to produce respectively the above proportions of hearth current. In Fig. 120 is shown a cross section through a 12-ton furnace, together with a diagram of the furnace connections from one group of transformers. The short leg of the other secondary group is similarly connected to the conductive hearth, and its phase will bear any one of the above given relationships to the corresponding leg of the former group. By the provision of tappings in the primary transformer windings it is also possible to work at three different voltages, one being used for melting, one for refining, and one when holding the metal at a constant temperature.

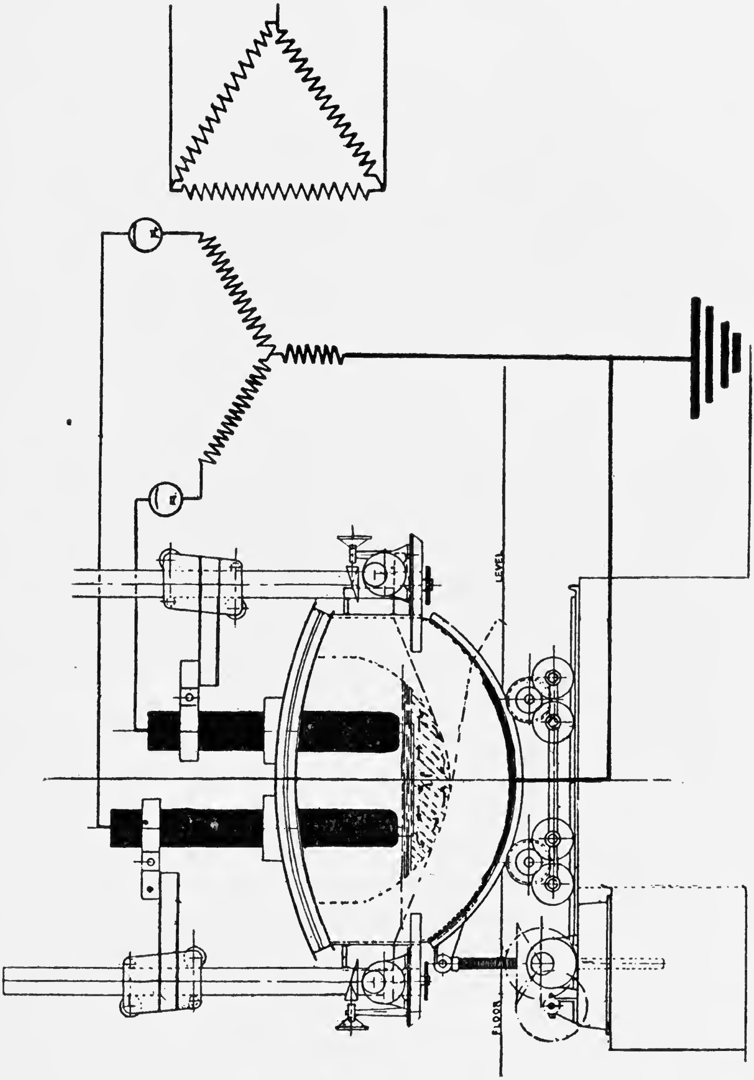

FIG. 120.

The conductive hearth is specially constructed to offer resistance to the passage of current, and from 8 per cent. to 10 per cent. of the total energy supplied to the furnace can be converted into heat in this way. By this means of applying heat to the bottom of the bath, it is claimed that convection currents are induced, which produce circulation of the metal. The effective result of such convection currents will obviously depend upon the difference in temperature between the bottom, middle, and upper layers. The latter is under the direct influence of arc heating and must be at the highest temperature; the middle layers derive heat by conduction, and would normally be above the temperature of the bottom layer. The degree of bottom heating must then be sufficient to raise the temperature of the lowest layer not only up to that of the supernatant metal, but actually above it, before any circulating convection current is possible.

Furnace Design.—For the smaller two-electrode furnaces the body is rectangular in shape, and provided with a curved bottom on which it rolls about its shorter axis. By this arrangement the pouring spout is fixed at one end of the body and a charging door at the other. A further door opening is provided in the long side of the body facing the electrode raising columns, which are mounted side by side. The method of guiding and moving the electrode carriages is substantially the same as that adopted for the Electro-Metals furnace; the current is also conveyed to the electrode in a similar manner by bent conductors and a clamping device, which, however, is constructed in the form of a flexible link belting, instead of using a small number of large links.

The large four top electrode furnace is circular, the electrode columns being equally spaced around the body. Both the large and small furnaces are supported on two pairs of rollers, and are tilted by a tilting screw, fed forwards or backwards by the usual swinging trunnion block and nut device, as shown in Fig. 121. The rollers upon which the furnace body rests are themselves supported on roller carriages, to which they impart a travelling movement when the furnace is tilted. The diameters of the supporting roller spindles, and of the tyres of the roller carriage wheels upon which they rest, are so proportioned that the travelling movement retains the lip of the spout in the same vertical plane during pouring.

Furnace Lining.—Greaves-Etchells furnaces have been invariably lined with basic material to obtain the required degree of hearth conductivity. The hearth is not, however, homogeneous, but is rammed in layers consisting of a basic mixture of varying composition, so as to obtain as far as possible an increasingly higher electrical resistance towards the top. In this way the greater part of the heat is generated by resistance in close vicinity to the molten metal. Experiments have recently been made with a conductive acid lined hearth and have given promise of ultimate satisfaction; no details of its construction are as yet available.

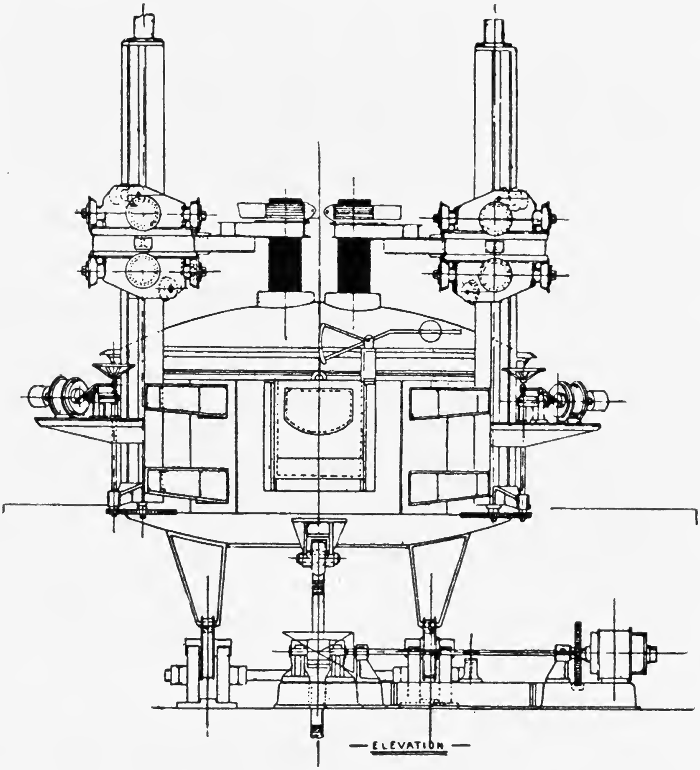

Fig. 121.

There are no special features in the method of lining apart from the construction of the hearth, which is at least 20 inches thick.

Electrodes.—Either amorphous or graphite electrodes can be used, but the latter variety is more generally favoured. The same considerations which govern the selection of either variety for the Electro-metals furnace are equally applicable in this case. A special gland sealing type of economiser is also used.

Booth-Hall Furnace.—*Special Characteristics.*—This furnace is designed to operate with a two-phase current, which is generally transformed from a three-phase high tension system.

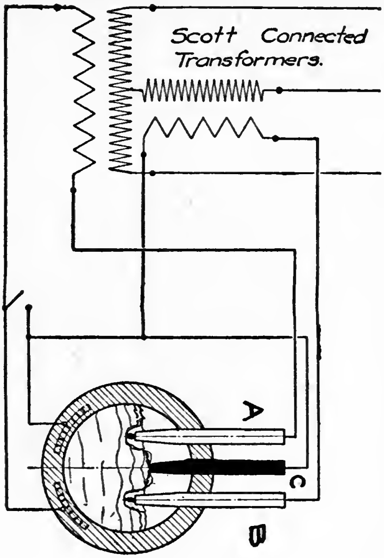

FIG. 122.

The low tension power connections are so arranged that a conductive hearth, in the case of the basic lined furnace, may be made an integral part of the power circuits, which are otherwise independent of the furnace lining and convey current to the charge through three carbon electrodes.

Electrical Design. — The two secondary phase circuits of a Scott-connected transformer group are independently connected to the furnace conductors. The outer terminal of each phase (Fig. 122) is connected to an upper vertical electrode, the inner terminals being connected to conducting grids embedded in a basic hearth. It will be noticed that the grid in each circuit is placed diagonally opposite to its corresponding upper electrode, instead of vertically beneath it. This is done with the object of lengthening the path of current through the furnace charge. A switch is provided for short-circuiting the two grids, so that the two phases may be connected to a common neutral point. A third auxiliary electrode is permanently connected to one of the grids, and is intended to carry the return current from the two main electrodes when the bottom is non-conductive, the grids being short-circuited for this purpose. With this system of two-phase connections it is possible to utilise the full power of the transformers at a time when the bottom conductive hearth is cold and inoperative. The grids are not used in the hearth construction of acid lined furnaces; the two main electrodes are then connected direct to the outer terminals of each phase, and the auxiliary electrode connected through the neutral return to the common neutral point of the two phases.

The auxiliary electrode, when in operation, either rests upon the charge of scrap or almost touches the slag covering the bath. The main phase electrodes strike direct arcs on to the charge and are quite independent of one another in their operation. A fluctuation of current in one phase will not, therefore, affect the other. This somewhat simplifies automatic regulation and eliminates the tendency to "hunt".

The voltage across each phase at the transformer terminals on open circuit varies from 75 to 150 volts, depending upon the conditions of the power service. Where a low power factor is permissible it may be an advantage to transform to the higher voltage, which will be reduced to a desirable figure at the furnace terminals by interposing powerful reactance coils in each low phase circuit. Such reactance coils, which cause a reduced power factor, would, of course, be installed solely for the purpose of steadying the load and thereby improving the load factor. The usual voltage variation is obtained by means of transformer tappings and selector switches.

The load rating of the Booth-Hall furnaces is exceptionally high, as the following figures show:—

¾ ton	300 K.V.A.
1½ tons	700 ,,
3 ,,	1400 ,,
6 ,,	2100 ,,
15 ,,	4200 ,,

Furnace Design.—The latest type of furnace has an elongated body with semicylindrical ends, so that the two main vertical electrodes can be set equidistant from the end walls at all points; this construction is followed so as to obtain uniform heating and wear of refractories. The furnace body, which has a dished bottom plate, is mounted on rockers and tilts about its shorter axis. The raising gear and gallows arm mountings for the two main electrodes are attached to one side of the furnace body, the auxiliary electrode being carried in a holder mounted on an inclined bracket fixed centrally on the opposite side. The auxiliary electrode is inclined downwards towards the furnace interior at an angle of about 45°, and passes through the furnace shell plate at a point about midway between the top of the walls and the slag line. The opening in the brickwork through which this electrode passes encloses a water-cooled guide box, which, in the 6-ton furnace, is 1 foot long and penetrates to within 6 inches of the inner face of an 18 inch lining. The furnace is designed to tilt either forwards or backwards from a vertical position, which enables slag to be skimmed over one door sill, and steel to be poured over a spout situated at the opposite end; this arrangement is used both in the new and in the older cylindrical body types.

In the earlier types, except for the smaller sizes, the auxiliary electrode was mounted in the same way as the main electrodes, and passed through the roof at a point midway between the centre line of the furnace and either one of the ends. The main electrodes are always symmetrically placed in relation to the furnace interior, as shown in Fig. 123. The transverse section shows two sets of cast steel grids, which are embedded in a basic hearth and insulated from one another; the bus bar connections are also shown. The electrodes are gripped by three equally spaced contact plates; one plate is fixed to the end of the gallows arm, the others being pivoted on the ends of horizontal levers, which have their hinged fulcrums a few inches behind the rigid contact plate; the levers extend to the back of the gallows arm, and can be thrust apart or brought together by a double threaded screw, turned by an insulated hand wheel for the purpose of either gripping or slackening the electrode. All three contact plates are water-cooled besides the back and front door frames; water jackets are also provided to reduce the annular openings between the electrodes and the roof brickwork. The furnace rolls on rockers, and is tilted by a heavy tilting bar carrying a rack, which is held in mesh with a driving pinion mounted in a rigid bearing. The vertical electrodes are controlled by motor or by hand through a rack and pinion drive. The roof consists of a steel frame lined with silica brick, and rests upon the furnace walls.

Furnace Lining.—There is no special feature in the method of building either a basic or acid lining. The walls of the larger furnace of 6 ton capacity are 18 inches thick, which is rather greater than in other types of furnaces. The basic hearth is rammed in with a hot mixture of dead burnt magnesite or dolomite, 15 per cent. of basic slag, and sufficient roofing pitch to make the mixture bind. When the lining is complete, the

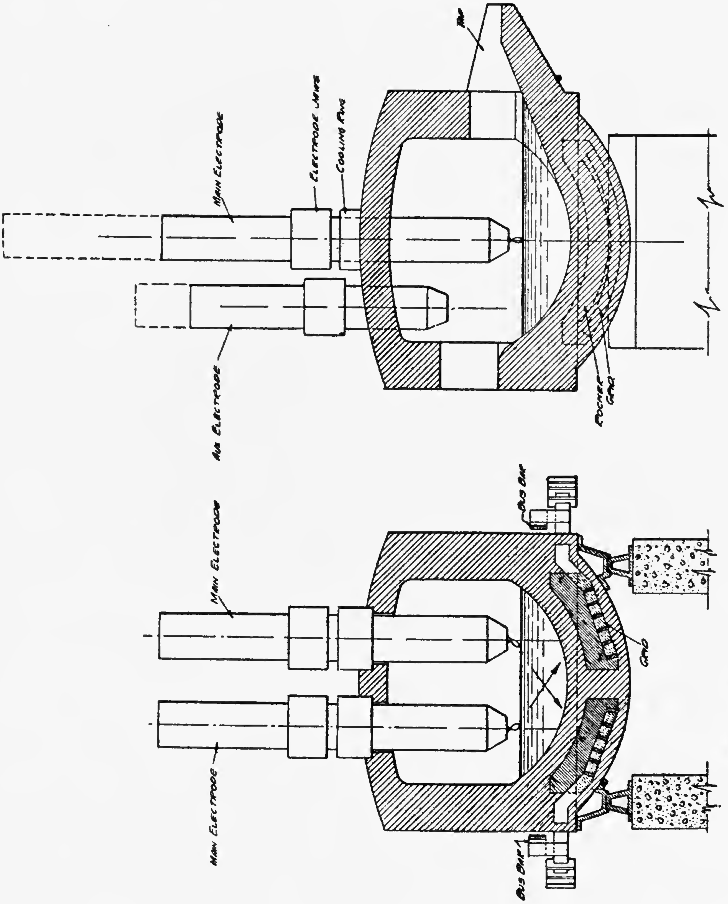

FIG. 123.

furnace is gradually heated by means of an arc to such a high temperature that the hearth becomes fritted in place. The hearth, which is never less than 24 inches thick, is frequently built up of basic material fritted in layers to form a solid homogeneous mass. There is nothing distinctive in the acid lining, which is composed of the usual silica brick and ganister materials; the hearth, however, can also be fritted on in layers by using the main electrodes in conjunction with the auxiliary.

Electrodes.—Either amorphous or graphite electrodes are used, the holder being so designed that interchange from one to the other type is easily accomplished by substituting contact plates of different curvature.

Furnace Operation.—In an acid lined furnace there is no conductive bottom, and the auxiliary electrode is always used as a common return for the main electrodes; for this purpose it is kept in touch with, or just rests upon the charge until melting is completed, after which it is adjusted to remain in contact with the slag. No refining other than that of carbon control is attempted with an acid bottom, and the process of steel-making is followed in the ordinary manner.

When starting to melt a cold charge, the auxiliary electrode mounting is released by a clutch attachment, permitting it to rest upon the scrap. Direct arcs are then struck by the two main electrodes, and melting begins in each arc zone. There is no definite melting zone under the auxiliary electrode, which remains supported by the unmelted charge. When melting has proceeded to such an extent that the unmelted portion begins to sink into the bath and can no longer carry the weight of the auxiliary electrode, the latter is again connected to the lifting gear, and then kept just in contact with the semi-fused scrap or bath. This procedure may be followed both in acid or basic lined furnaces, only in the latter case, the hearth usually becomes sufficiently conductive to carry the full return current before the charge is melted, when the auxiliary electrode is raised clear of the charge. The furnace then operates electrically in the same manner as the original two-phase Stobie furnace, in which the two-phase circuits are independent. The Booth-Hall furnaces have been used solely for melting scrap charges up to the present time, the high voltage arcs, thick linings and the heavy transformer rating being especially designed for this purpose. The radiation loss under the two latter conditions is only a small fraction of the very considerable power input, as will be seen from the ratings of the 1½ and 3 ton furnaces, and the thermal efficiency is probably about 85 per cent. at full load. The power consumption per ton of steel should be rather lower than for furnaces less heavily rated. A small 1½ ton furnace, working about eleven hours a day, has averaged about 670 units per ton of steel over several months, which is a very good performance for a basic furnace operating intermittently.

Ludlum Furnace.—The chief features of this furnace are purely constructional, and concern more especially the disposition of the electrodes and the shape of the melting chamber.

Electrical Design.—The furnace operates off a simple three-phase low tension supply, the line conductors being connected to three electrodes which enter the furnace through the roof. The open circuit line voltages, which are interchangeable, are 110, 95, and 80.

Furnace Design.—The three electrodes are placed in a line and the shape of the body so designed that the heat radiated from the arc zones is, as far as possible, of uniform intensity at all points of the lining. The body has therefore an ellipsoidal shape, and is provided with one door at each end of the major axis. The roof, which is of varying width between the side walls, has a constantly changing curvature, but is said to be very simple to build and provides a uniform distribution of reflected heat. The furnace tilts on rockers set parallel to the major axis, the pouring spout being incorporated in one of the end charging doors. The electrode columns are mounted on one side of the furnace shell parallel to the major axis.

The furnace is suitable for working either the acid or basic process, as the hearth, which is quite independent of the electrical circuits, can be readily lined with acid material. It is also interesting to note that graphite electrodes have been definitely found to be more economical than amorphous by users of this type of furnace.

CHAPTER XIII.

REFRACTORY MATERIALS AND THEIR APPLICATION TO ELECTRIC FURNACE CONSTRUCTION.

THE economical production of electric steel is, above all other considerations, dependent upon regularity and speed of operation. Frequent stoppages for furnace repairs, metallurgical difficulties caused by the chemical action of failing brickwork, and excessive loss of heat through prematurely worn linings, may all be credited to the failure of the refractory materials used in furnace construction. It would not, however, be correct in all cases to assign failure of the lining to the inferior quality of the material used, since heat developed by the electric arc is so intense that by ill-considered application it may easily cause complete breakdown of the best refractory materials procurable. Of recent years special refractories, such as fused alumina, carborundum and zirconia, have been suggested for use in the electric furnace in the form of bricks, as it is admitted that both silica and magnesite do not meet the very exacting demands occasioned by the peculiar conditions of electric furnace operation.

Classification.—Refractory materials are broadly classified according to their powers of resisting the corrosive action of siliceous and basic slags. Materials in which silica predominates are acid, while those composed essentially of such powerful bases as lime and magnesia are basic; others, composed of oxides or compounds that are attacked equally by both acid and basic fluxes alike, or remain unattacked by either, are called "neutral". Considerable care has to be exercised in the selection of refractories and their treatment both before and during service, and in this chapter such information as is given bears more especially upon their use in electric furnaces.

Conditions of Service.—Before considering the properties, composition, and behaviour of the various refractories commonly used, it is advisable to consider the nature of the chemical, physical, and mechanical conditions to which they are exposed.

Chemical corrosion by the action of slag is not so confined to the banks as in the open-hearth furnace, but is promoted also in the region of the walls and roof by the action of fluxes, especially powdered lime, and by slag which may be violently thrown upwards from the bath on the addition of damp scrap, or for other reasons. The high temperature of the slag also exerts a powerful influence upon its corrosive action on the lining, which may be further intensified by only slight deviation from its correct composition as regards degree of basicity or the amount of iron oxide present.

The requirements imposed by the physical conditions are essentially those which demand the highest power of resistance to change of state under the influence of an intense and sometimes abruptly changing temperature. In the case of indirect arc furnaces, the roof and walls are more exposed to the direct radiation of heat from the arc than is the case with direct arc furnaces, but even in the latter certain portions of the lining, under the influence of drawn-out or laterally deflected arcs, will be exposed to intense local heating.

The internal mechanical stresses set up by temperature variation are, in the case of electric furnace linings, exceedingly severe. Almost all refractory materials expand on heating, so that frequent changes of temperature accompanied by alternate expansion and contraction are liable to cause rupture. It is the usual practice in electric furnace roof construction to build the silica brickwork in a rigid steel frame, and, although special precautions are taken to allow for expansion, the risk of distortion, which may lead to crushing of the brickwork, will be greater than in the case of large gas furnace roofs which are freer to rise and fall under less frequent variations of temperature. Rapid heating causes unequal expansion, and internal mechanical stresses are set up which cause rupture unless the tensile strength of the agglomerated material is sufficiently high.

The charging and working doors of electric furnaces are generally made as small as is consistent with the necessary ease of charging, fettling, and withdrawal of broken electrodes, so that the door jambs and arches are always liable to rough usage and injury from the various tools used for manipulation. Unfortunately, it is just the brickwork round the door openings that suffers most from temperature changes, so that it is difficult to find a satisfactory material that will resist such a combination of severe conditions.

After indicating the severity of the working conditions that have to be faced, it is now possible to consider the properties and suitability of various refractory materials, classified according to their character as previously defined.

Basic Materials.—Dolomite and magnesite are the only basic materials used either for the manufacture of bricks or for the construction of furnace hearths.

Dolomite.—The mineral dolomite may be regarded as a crystalline limestone in which part of the calcium carbonate is replaced by magnesium carbonate. It becomes exceedingly hard and tough after calcination at a high temperature, and in this state has only a moderate tendency to slake when exposed to a moist atmosphere. Before it is suitable as a refractory material, it requires to be burnt or calcined until all the carbon dioxide has been expelled. During this calcination the raw material shrinks and, if properly burnt, will show signs of incipient fusion.

The composition of dolomite before and after burning may be given by the following typical analyses:—

Constituents.	Per Cent. before Burning.	Per Cent. after Burning.
Carbon dioxide	44-46	1·0
FeO and Al_2O_3	1·2	3·1
CaO	32·0	57·3
MgO	20·0	37·4
SiO_2	1·0	2·0-3·5

Burnt dolomite is best purchased and stored in the lump condition, and only crushed to the required size shortly before use. Sometimes it is bought already crushed and barrelled, but this is not to be recommended, as it is more liable to perish or slake during transit and on prolonged standing. It should be remembered that the fine, white, slaked powder present in crushed dolomite, that has been stored for any length of time, contains water of hydration, and if such material were used for forming a furnace bottom the water would subsequently be driven off and destroy the solidity of the rammed material. Power of heat resistance will depend on the amount of silica and other oxides present, which tend to increase the fusibility by the formation of multi-base silicates.

Dolomite is usually reserved for the construction of the working hearth, but can be more extensively used in the lining of small furnaces; in the latter case entire walls composed of rammed dolomite have given perfect satisfaction.

Magnesite.—Magnesite is the only material that can be satisfactorily used for the manufacture of basic bricks. The mineral, which in its raw state consists of magnesium carbonate containing small and varying amounts of SiO_2, Al_2O_3, and Fe_2O_3, is extensively mined in Styria, Greece, Asia Minor, and India. Like dolomite it requires to be burnt at a very high temperature, not only to expel the carbon dioxide from the carbonates, but also to convert the resulting amorphous magnesia (MgO) into a crystalline state. The proportion of ferric oxide present in the raw material is considered of far greater importance than silica, as it gives a much greater binding strength to the particles of a burnt brick without impairing its power of heat resistance.

There is no doubt that bricks made from Austrian magnesite have given better results than others, notwithstanding the fact that the burnt material contains a larger percentage of impurities than varieties of inferior physical quality. The following are typical analyses of burnt Grecian and Austrian magnesite[1]:—

Constituent.	Grecian.	Austrian.
MgO	88·0 per cent.	83·0 per cent.
CaO	5·0 ,,	4·0 ,,
Al_2O_3	1·0 ,,	3·0 ,,
Fe_2O_3	·5 ,,	8·0 ,,
SiO_2	5·5 ,,	2·0 ,,

Amorphous magnesium oxide becomes crystalline at the temperature normally attained in electric furnaces, and in so doing undergoes considerable shrinkage. This is the chief cause of "spalling," which leads to the rapid destruction of magnesite walls, and for this reason raw magnesite should be burnt at the highest possible temperature until it becomes at

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[1] Transactions of the Faraday Society, June, 1917.

least partially crystalline. The Austrian "Spaeter" brick has a dark, finely crystalline structure, and is somewhat heavier than English varieties; the greater density, although no doubt partly due to the higher percentage of iron oxide, results from the more perfect conversion of the amorphous magnesia to the crystalline state.

Magnesite, which is burnt either in cupolas or in gas furnaces, is generally crushed rather smaller than dolomite before calcination. Cupola-burnt magnesite requires to be carefully hand picked, and all semi-burnt pieces are returned for re-calcination. Burnt magnesite may be used in place of dolomite for lining furnace hearths, but it is doubtful whether any advantage can be gained in cases where it is not possible to build up the hearth by fritting on thin successive layers of the burnt material, mixed with a suitable chemical binder such as basic slag. For electric steel furnace use there is no question that dolomite alone can be employed with perfect success, provided the correct methods of mixing and ramming are followed.

So as to judge the suitability of magnesite bricks for electric furnace use, it is well to consider their physical properties and the temperature and mechanical conditions they are called upon to resist. Magnesite bricks have a high coefficient of thermal conductivity, and in order to prevent excessive heat loss, they should be backed wherever possible by a material having a much higher power of heat insulation. In most basic lined electric furnaces the basic brick lining is, however, only continued a few courses above the slag line, so that this precaution need hardly be followed where the walls are only built 9 inches thick. Magnesite bricks behave admirably so far as they resist fusion at very high temperatures and corrosion by basic slags, and the only serious objection to their use is their friability when hot and their tendency to burst or "spall" after exposure to very high temperatures. This serious drawback has been studied by numerous investigators and, according to the general opinion formed, is largely due to imperfect conversion of the amorphous magnesite to its crystalline oxide during the burning operation. Magnesite bricks when used in electric furnaces are exposed to temperatures which rapidly promote this physical change, accompanied by shrinkage, so that fracture generally occurs along a plane dividing the unaltered from the altered material. Those indirect arc furnaces, in which it is imperative to use magnesite brick instead of silica for the roof construction, are placed at a serious disadvantage in this respect.

Neutral Materials.—The only neutral refractories commonly used are bauxite and chromite, but considerable work has recently been done on the study of fused alumina or "alundum" and zirconia for use in electric furnace linings.

Bauxite.—Bauxite is found as a mineral containing 45 per cent. to 75 per cent. Al_2O_3 in the form of a hydrated oxide together with varying amounts of iron oxide and silica. For use as a refractory material bauxite should contain as much alumina as possible while the impurities should be correspondingly low. The analysis below is that of a high grade sample of French bauxite :—

Al_2O_3	78·1	per cent.
Fe_2O_3	1·02	,,
SiO_2	5·78	,,
Water	15·10	,,

The raw material is first calcined to dehydrate the oxides present, then mixed with high-grade clay to form a plastic mixture, and finally pressed into the form of bricks. The refractory quality of bauxite is considerably influenced by the nature and the amount of the impurities present, which may form double and triple silicates of lower melting-point than pure aluminium silicate. The best quality bricks have a coarse-grained fracture, similar to a fire-brick but nearly white. Iron oxide colours the brick, from a pale yellow to a light red, according to the amount present. The inferior grades behave best in an oxidising atmosphere, and when exposed to reducing gases become less refractory owing to the reduction of the ferric oxide to ferrous oxide, which is not only more fusible but readily combines to form double silicates.

Bauxite bricks of the highest quality provide an excellent material for furnace linings, but, unfortunately, their extended use is limited by their high cost. They have a low coefficient of expansion, and are quite unaffected by the most abrupt changes of temperature; the neutral behaviour of bauxite enables it to be placed in contact with either silica or magnesite bricks, so that it can be used as a neutral parting in special cases. The finest bauxite will resist higher temperatures than silica, and although costly, may sometimes prove more economical for roofs of high temperature furnaces operating intermittently. Bauxite bricks are, however, liable to be fluxed by fumes of metallic oxides or lime, so that their value as a refractory is greatly reduced in basic lined furnaces. Patching cement, of the same material as is used for pressing into bricks, is a most useful and better substitute for magnesite or ganister, since it is more plastic and more readily adheres to damaged brickwork. Although bauxite resists chemical fusion with either basic or acid refractories, it must not be assumed that it is equally inert to the action of slags which, however acid or basic, contain silicates and therefore readily combine with the alumina (Al_2O_3) to form more complex silicates.

Alundum.—Alumina as a refractory material has so far been considered in its amorphous state. Considerable development has been made in the preparation and use of fused crystalline alumina, which in moulded form is known as "Alundum". This refractory has been used for electric furnace roofs in America, but it is said that difficulties are encountered owing to the action of lime vapours arising from the intensely heated basic slag. This action will be accentuated by the silica present, which enters the brick as a constituent of the binder employed.

Chromite.—This mineral, consisting of mixed chromium and iron oxides, is used to a limited extent as a refractory material, and resists fusion at exceedingly high temperatures.

The crude ore, when crushed to pass about ¼ inch screen, is sometimes employed as a parting between magnesite and silica brickwork; this practice was at one time largely followed in lining basic open hearth furnaces, but has now been practically given up. When used in electric furnaces there is always the danger of reduction should any become displaced and enter the highly reducing basic slag. Chromite bricks are very heavy and mechanically weak, and are seldom used.

Acid Materials.—*Silica.*—Silica is the predominating constituent of all acid refractories, and ranges from 98·5 per cent. in the Sheffield "black ganister" to 80 per cent. in the common grade of "cupola" ganister. Silica rock, containing over 97 per cent. of SiO_2, is generally used for the manufacture of bricks, whereas the various grades of ganister containing rather less silica are usually employed for ramming and patching. The higher the SiO_2 the less will be the tendency to bind, either before or after burning, so that in the manufacture of bricks from the purest quartzite about 1½ per cent. of lime is added, which frits together the quartz particles during the burning operation to form a concrete mass. It is well known that silica can exist in several different forms according to the temperature to which it has been exposed.

Quartzose silica, as used for the manufacture of silica bricks, has a specific gravity of 2·65 which slowly falls to 2·3 under the action of heat, which at the same time causes a slow transformation to an allotropic variety. The complete change is equivalent to an expansion of 16 per cent., which is, however, partly counteracted by the natural shrinkage that accompanies incipient fusion at the highest furnace temperatures. Once this transformation is complete, the degree of expansion and contraction resulting from temperature changes will become far less marked. This explanation falls in with the observed fact that a well-seasoned roof is far less sensitive to changes of temperature than when new. The importance of burning silica bricks at the highest kiln temperature possible is evident, as by this treatment the resistance to "spalling" will be increased. The softening or fusing point of silica bricks has been variously given as lying between 1650° C. and 1800° C., so that great care must be taken in the operation of electric furnaces in which higher temperatures are so easily and accidentally reached. Fusion at normal casting temperatures is not such a serious factor as the fluxing action of basic dust and heavy oxide fumes, which together form fusible double silicates and consume the brickwork by chemical action.

The greatest trouble to be met with in the use of silica bricks is their tendency to "spall". "Spalling" is partly due to stresses set up by unequal expansion due to rapid heating, alternate expansion and contraction, and compression caused by the considerable expansion of untransformed quartzose silica.

Unless sufficient provision is made to allow for expansion during the initial heating, silica bricks will be unable to withstand the crushing stresses and burst. It must be admitted that the roof has to withstand harsher treatment in electric than in gas furnaces, where mechanical provision is more easily made to allow for this expansion. For this reason only the best silica bricks should be used, and great care taken not to hasten the initial drying out and heating to the full furnace temperature. For the first two or three heats the lining should be carefully nursed, and never exposed to an unnecessarily high temperature until properly glazed and seasoned.

The grain size of silica bricks and the degree of burning are just as important as their chemical composition, since these factors alone influence their resistance to "spalling". As a general rule a high quality brick will have a very pale yellow fracture, showing a structure composed of numerous sharp irregular-shaped particles of various sizes, closely bound together in a fine grained matrix. The presence of small cavities is not detrimental. Bricks coloured by red oxide of iron may behave well in oxidising atmospheres, but are not suitable for electric furnaces in which highly reducing gases are generated. They should be mechanically strong and ring when struck; large blocks are seldom properly burnt in the centre, and this point should be considered when designing special roof bricks or other special shapes. Silica bricks readily absorb water owing to their porosity, and should always be stored under cover. Any moisture absorbed by a brick is vaporised in heating, but may only escape with difficulty, so that the tendency to "spall" or burst is greatly increased unless special precautions are taken to dry out very slowly. Some brick-makers consider that moisture is not completely expelled below a red heat, and if this be the case, it is not safe to assume that a roof is perfectly dehydrated when steam ceases to rise.

Ganister.—The term "Ganister" applies more properly to silica rock of lower silica content than is used for the manufacture of bricks, the so-called "black ganister" being a true quartzite. Ganister rock, when crushed and moistened, is slightly plastic owing to the small percentages of alumina and lime present as silicates. The plasticity of the less refractory grades of ganister increases with the percentage of these natural silicates.

Crucible ganister is a high grade ganister rock with just sufficient plasticity to enable it to hold together when rammed, and is used for lining crucible steel pot holes. This grade is very light in colour and is a very useful material for patching silica brick linings and door jambs. It is also used for building up the hearths of acid lined electric furnaces, and is, at the same time, most useful as a fettling material.

Cupola ganister is the least refractory grade, and is only used for making up furnace launders or spouts, and in other places that are not exposed to very high temperatures.

Silica Cement.—This material is used for bedding silica bricks, and generally consists of a finely ground mixture of old silica bricks and fresh quartzite rock to which a small quantity of a lower grade ganister is added to make it sufficiently plastic. Fire-clay is sometimes used as a substitute for the ganister, but this practice destroys the refractory properties of the cement and is not to be recommended.

Silica Sand.—Highly refractory sand, approaching pure silica as nearly as possible, is used for fettling the banks of large electric furnaces. In small furnaces the banks are usually too steep and ganister is then used. Originally, silver sand for this purpose was imported from Belgium and Holland but, when these sources of supply failed, English sands were used, which have given equally good results.

Analyses of Acid Materials.—Typical analyses of silica bricks and other acid materials are given below :—

	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	Alkalies.	MgO.	Loss on Ignition.
Dinas silica brick .	96·8	·92	·5	1·2	·2	—	—
Yorkshire silica brick	96·2	·92	·63	1·5	·19	·22	—
Crucible ganister .	95·3	2·10	·80	1·10	·28	·23	—
Scotch silica sand .	99·4	·6	—	—	—	—	—

Fire-clay.—Although fire-clay is not commonly regarded as an acid refractory its analysis brings it within this category. Few fire-clay bricks are capable of withstanding the temperatures of steel furnaces, so that they are only used in a limited extent as a backing to either magnesite or silica bricks and for lining doors. Fire-clay is, however, extensively used for lining ladles and in the ingot pit, so that it deserves mention when dealing generally with the various refractories used in the manufacture of steel.

Fire-clays consist essentially of alumina and silica with small and varying amounts of iron oxide, lime, magnesia, and alkalies, which are usually present as impurities. Neglecting the effect of impurities, there is, according to Prof. H. Le Chatelier, a definite compound of alumina and silica, consisting of approximately 15 molecules of SiO_2 and 1 molecule Al_2O_3, which has the lowest melting-point, and any increase of either the SiO_2 or the alumina content above these proportions increases the refractoriness. At one end of the scale there is pure alumina, and at the other pure silica, the latter having the lower melting-point. This serves as a rough guide for judging the working properties of a fire-brick from its analysis, other conditions being equal.

The behaviour of fire-bricks will depend, like silica, upon their structure, chemical analysis and conditions of service, and they must be selected for any particular use from the results of practical experience. For lining furnace doors, the finest brick should be used to avoid constant renewal and badly fluxed door sills.

Fire-clay, whether used for bedding bricks or for ingot pit work, is mixed with water and allowed to "temper" several days before use, otherwise it does not hold the water so well and is not sufficiently plastic.

A few typical analyses of fire-clays and fire-bricks are given in the following table :—[1]

	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	Alkalies.	Loss on Ignition.	H_2O.
Stourbridge fire-clay	65·0	22·0	2·0	·5	—	·5	10·0	—
Stourbridge fire-brick	69·0	27·3	1·86	·27	·32	·91	—	—
Glenboig fire-clay .	62·0	28·0	2·0	·5	·5	·5	6·5	—
Glenboig fire-brick .	63·0	32·0	2·85	·79	·36	·94	—	—

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[1] Transactions of the Faraday Society, June, 1917.

CHAPTER XIV.

FURNACE LINING AND LINING REPAIRS.

THE materials used in the hearth construction of electric furnaces will be either basic or acid, according to the nature of the process employed. Basic and acid linings are best considered independently, the roof and upper portion of the walls being substantially the same in both cases. The methods of hearth construction also depend upon the presence or absence of bottom electrodes, so that further subdivision is necessary. The following methods are those generally adopted, but may be slightly modified to suit certain types of furnaces, of which special mention has already been made.

Basic Linings without Bottom Electrodes.—The brick lining immediately next to the shell plates is generally built of best quality fire-bricks, which may be carried up from the bottom to a point beyond which there would be a possible risk of contact with slag. From that point upwards magnesite bricks must be used for the purpose of resisting the corrosive action of basic slag in the event of it penetrating the dolomite hearth lining. The magnesite bricks are always built up to a level at least two courses above the charging door sill, so that no serious injury can be done to the uncovered walls by the corrosive action of oxidised scrap, which may rest against them during the melting operation. Silica bricks, of the best quality procurable, are used for the upper part of the wall lining and roof. In the case of furnace walls thicker than 9 inches the magnesite and silica wall bricks are usually backed with fire-brick, and in some cases the magnesite courses will be only 4½ inches thick in place of the full 9 inches, which is more usually preferred. It was at one time customary to use magnesite bricks in place of fire-bricks for lining the bottom and lower walls, but this practice, which was far more expensive and resulted in greater heat losses, owing to the better heat conductivity of magnesite, has now been

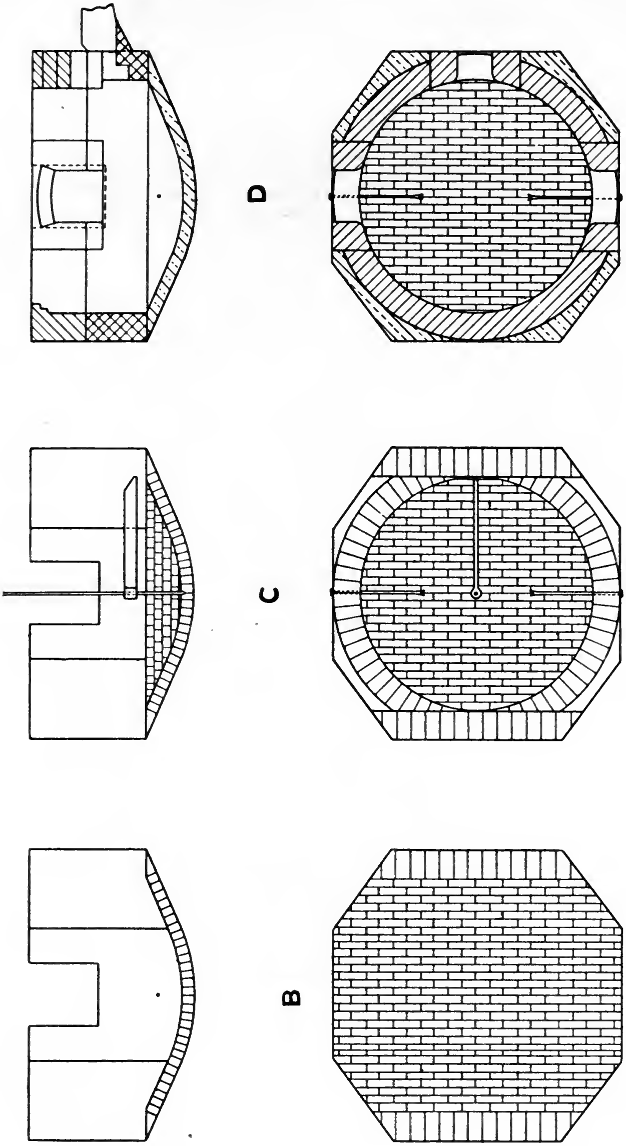

Fig. 124.

given up. The thickness of brickwork on the bottom and next to the side shell plates varies according to the design of furnace, but the general disposition of the magnesite, fire-clay and silica bricks will in all cases be alike.

Three successive stages in lining a furnace are shown in Fig. 124, a plan and transverse section being given to illustrate the brickwork at each stage. The bottom plate is first covered with a course of best quality fire-brick carefully bedded and bonded to obtain the best solidity possible, as shown in the sketch B. The fire-brick is continued upwards, as shown in C, to form a flat seating, upon which the magnesite walls are then built; two short ½ inch iron rods penetrate the brickwork so as to ensure electrical contact between the shell plate and a hot and conductive portion of the dolomite bottom, when the furnace is in operation. The walls of magnesite brick are then built with the aid of a carefully centred trammel, as indicated in D, up to a level two courses above the charging door sill; beyond this point there is no risk of any corrosive action of rusty scrap or slag on the silica brick. Magnesite bricks are usually bedded with a magnesite slurry, which consists of finely ground magnesite bricks mixed with just enough clay to make it slightly plastic. They should be set as close as possible, using only sufficient slurry to secure tight joints. The silica brick walls are carried upwards from the top magnesite course without any neutral parting, the bricks being bedded in silica cement and all joints made as close as possible. Rammed dolomite has also been used with success for the construction of walls of small furnaces, bricks being used for the door openings only. It is better not to bond the door jambs to the wall lining, otherwise their renewal or repair, which is usually necessary once during the lifetime of a lining, is more difficult to carry out without risk of damage to the walls themselves. The top two or three courses of silica brick are, in some cases, stepped forward to offer greater protection to the lower angle of the roof frame, which supports the skewback blocks. A method frequently adopted for constructing walls of 14 inch thickness in the case of furnaces of 7 tons capacity and upwards, is shown in Fig. 125.

Before putting in the dolomite bottom, the brick lining should be thoroughly dried out, to prevent the possibility of moisture coming into contact with the dolomite during or after the process of baking. This drying out can be done either before the silica walls are begun, or after, as may be most convenient. The furnace, after drying out and cleaning, is ready to receive the dolomite hearth.

The hearth, composed of rammed dolomite, is the most important part of the lining, and has to be formed with great care to ensure uniformity and perfect solidity. Imperfect ramming or the use of bad material may lead to serious trouble, and in extreme cases to a break-out. Considerable attention is given, therefore, to the preparation of the dolomite and the subsequent mixing with a binder, which consists of tar and pitch in suitable proportions. Such a mixture is commonly called "Black Basic" in open-hearth and converter plants, where it is usually prepared in special mills and mixers. Unfortunately, the size of most electric furnace plants does not justify the installation of special plant for preparation of the "black basic," and therefore recourse is made to hand-mixing, which, to be conducted successfully, entails much hard work and patience. If a crushing mill is available, the dolomite, preferably stored in lump form, should be freshly crushed to pass a ½ inch or ¾ inch mesh sieve, while sufficient "smalls" and dust should be made when crushing to form a perfectly solid matrix for the larger pieces when rammed. The correct porportion of large to small is soon determined by experience, and then there should be no further difficulty in obtaining a uniform grading. Partly slaked

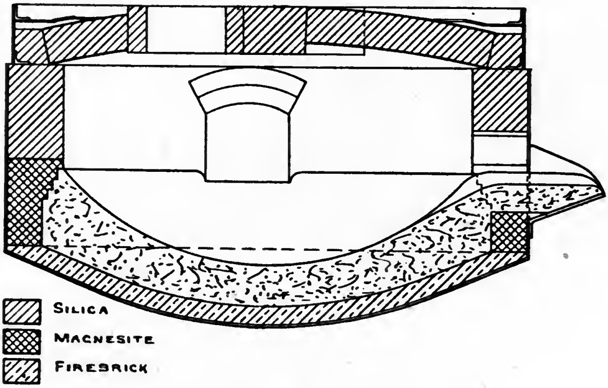

FIG. 125.

dolomite should not be used, as the moisture in the dust, which helps to form the matrix, is later driven off, and the solidity of the material is then destroyed.

The crushed dolomite is heated to about 100° C., and then mixed with a small quantity of binder. The correct proportions of boiled anhydrous tar and hard black pitch, of which the binder is composed, are best found by trial in each case, as the viscosity and melting-points of boiled tar and pitch are very variable quantities. As a general guide, pitch should be added until the resulting tar and pitch mixture sets hard when pressed on to a cold metal surface. The dolomite is continually mixed until every particle is well covered with binder and the entire mixture is black. The quantity of binder should be only just sufficient to make the mixture adhesive without giving it a wet-looking appearance.

This mixing may be conveniently carried out in a small sand-mixing runner mill, which soon assumes the temperature of the dolomite and then has no further tendency to chill and clog up. All brick surfaces on to which the dolomite is rammed are first brushed over with thin tar to facilitate adhesion. The prepared dolomite, after being allowed to cool down to between 100° and 150° F., is spread in a layer not exceeding 1½ inches thick, and rammed hard until it rings under the blow of the ramming iron. Only a small area is covered and rammed at one time, otherwise imperfectly rammed portions might escape the notice of the men and result in soft porous patches. A hard rammed surface, when once cold, should be roughed all over with a pick to ensure a proper bond with any fresh mixture rammed upon it. The actual bottom is usually rammed up flat to the correct thickness in the middle, the sides being afterwards sloped up until the entire hearth has assumed its proper shape.

When the furnace body lining has been completed, the roof is placed in position, care being taken, before finally bolting it down, to centre it to the electrode holders in order to ensure proper clearance for each electrode.

Basic Linings with Bottom Conductors.—Basic lined furnaces, in which a conductive hearth becomes an integral part of the load circuit, are lined in a rather similar manner to that just described, the only material difference lying in the construction of the hearth itself. Since it is necessary to distribute the current from the bottom conductor proper to the conductive hearth over as large an area as possible, special methods have to be followed, which vary only in detail for the several different types of furnaces belonging to this class. For those types of bottom electrode furnaces in which metallic bottom electrodes penetrate the hearth and convey the current direct to the metallic charge independently of the hearth itself, no special precautions are necessary to secure definite distribution of current through the hearth itself.

True, conductive hearth furnaces, on the other hand, present a different problem and demand special consideration. "Black basic," consisting of magnesite or dolomite mixed with a pitch-tar binder, is generally used for forming the hearth and is prepared and rammed to shape just in the same way as for non-conductive bottom furnaces. It is not a good conductor of electricity when cold, even after baking, but when heated the conductivity progressively improves with rise of temperature. This property points to a source of danger arising from the presence of any low resistance circuits or, in other words, from unequal distribution of the current flowing from the bottom conductors to the charge in the furnace. To take an extreme example, assume that there is only one small bottom electrode embedded centrally beneath the conductive hearth and supplying current to it. It is evident that, as soon as the hearth becomes hot enough to conduct, the current will take the shortest path through it to the metallic charge and therefore be concentrated in a small zone of high current density. The hearth would then become heated locally by the passage of the current and, becoming more conductive, would allow a heavier current to flow. This might progressively continue until the local hearth temperature became sufficiently high to cause disintegration of the refractory material. For this reason it is of great importance to convey the current from the bottom conductors to the hearth in such a manner that the current density is as uniform as possible at all points, so that all risk of local heating and disintegration may be avoided. It was at one time the practice to embed two or more bottom electrodes in the conductive hearth, which were securely connected to the conducting bus bars or cables. These bottom electrodes consisted in some cases of carbon blocks and in others of water-cooled metallic blocks. This method is open to objection on account of the local heating in the conductive hearth, due to the high current density along the shortest paths between the metallic charge in the furnace and the embedded electrodes. In order to overcome these disadvantages the following method has been successfully adopted, and can equally well be applied to any bottom electrode furnace in which the entire conductive hearth acts as a single unit. This method can best be explained by reference to Fig. 126. Two narrow rectangular openings D are cut in the bottom shell plate B, each opening being about 18 inches from the end plates and extending transversely across the furnace to within a similar distance from the back and front plates. A fire-brick covering A, $4\frac{1}{2}$ inches thick, is then laid on the bottom plate, leaving openings E to coincide with those in the bottom plate. A number of copper bars F about 5 inches × $\frac{5}{8}$ inch, bent at right angles at each end, are laid on the brickwork so that the short bent portions pass through the narrow openings and project downwards about 6 inches. These copper bars are further connected together at both ends by heavy collector bars G, which in their turn are provided with terminal heads for connection to the bottom conductor cables. By adopting this system the bottom electrode may be regarded as a single copper plate covering an area immediately below, but rather larger than, the actual working bottom of the furnace hearth.

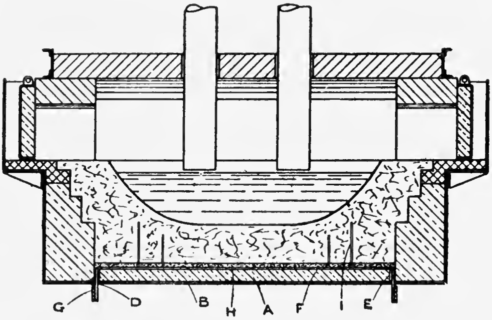

FIG. 126.

Carbon paste H is then sometimes rammed over the floor of the furnace, filling all spaces between the copper strips and covering them to a depth of 1 to 2 inches. "Black basic" I is then rammed upon this carbon layer in the usual manner and with a minimum thickness of about 12 inches. To ensure proper current distribution over those areas underlying the foot of the furnace banks, $\frac{3}{4}$ inch steel bars are spaced at intervals and connect the interior of the hearth to the carbon floor, the "black basic" being rammed around them in the usual manner.

Furnace bottoms constructed in this way have given very good results, and have lasted upwards of nine months before being renewed. It should be pointed out that, according to the method of preparing "black basic" as previously described for non-conductive hearth furnaces, a minimum quantity of carbonaceous binder is used, whereas in order to improve the conductivity of the hearth of bottom conductor furnaces, it is sometimes advisable to mix in a small quantity of ground carbon, more especially for ramming that portion of the hearth nearest the bottom electrode; this practice is not, however, always followed. To improve the conductivity of the hearth material a small quantity of magnesite and iron nails are also sometimes added, but with doubtful advantage.

Furnace Roofs.—Furnace roofs are usually built of silica bricks in a rigid steel framework, the correct position of the electrode openings being obtained by setting the bricks on a prepared template upon which the roof frame is carefully centred. Provision is made for free expansion of the brickwork by interposing a large number of thin wood sheets, $\frac{1}{32}$ inch to $\frac{1}{16}$ inch thick, between the bricks, both in the cross and face joints. Without taking this precaution the brickwork, especially in the case of large roofs, might crush and "spall" badly during the preliminary heating.

Drying out and Baking in Furnace Linings.—Before actual melting operations are begun, it is customary to dry out the brickwork and bake in the basic bottom. This operation is important for several reasons and should never be omitted.

Silica brickwork, which generally constitutes the upper wall and roof lining of basic furnaces, requires slow and careful heating to remove all moisture, and to minimise the unequal expansion due to the difference in temperature between the inner and outer surfaces, which might give rise to "spalling". It is advisable also during this preliminary heating to finish at a temperature that is just sufficient to glaze the silica brickwork without producing any tendency to drip. During the first three or four heats the roof should be carefully nursed and never heated to a prolonged temperature higher than the softening point of the brickwork. By following this practice the roof can be well "seasoned" and its life considerably prolonged for the reasons given in Chapter XIII. The heating and baking-in of the basic hearth demands equal care, and is best carried out slowly. During this operation the volatile constituents of the pitch-tar binder are expelled, and if driven off too rapidly, will cause the soft plastic material to swell and lose its solidity.

The preliminary warming of the furnace is therefore a matter that must be regulated by the necessarily slow rate of drying and heating the silica roof, which is usually raised to a very dull red heat in about 5-8 hours or longer, according to the humidity of the brickwork and span of the roof. After this point has been reached, the temperature may be slowly raised until the interior of the furnace is about 1100° C.; this temperature is then maintained until the gases given off by decomposition of the pitch-tar binder are no longer seen to burn at any crevices inside the furnace formed by shrinkage of the dolomite away from the brickwork. The temperature is then further raised and held for about half-an-hour in order to glaze the silica brickwork and frit the surface of the basic hearth.

This completes the prepartion of the furnace lining prior to receiving its first charge.

Mode of Heating.—In the case of indirect arc furnaces the heat can be applied simply by striking an arc between the electrodes, so that the entire lining is heated by radiation just as when melting. This method is not applicable to direct arc furnaces, which require a conductive charge to complete the circuit of the current flowing through each electrode, irrespective of whether the path be between two or more vertical upper electrodes, or from one or more upper electrodes through a conductive hearth to a bottom electrode and return conducting cables.

In those cases where the circuit is entirely independent of the furnace hearth, as in the Heroult type, it is necessary to provide a conductive charge on to which the direct arcs may strike. Hard coke or broken lumps of amorphous electrode may be used for this purpose, in which case the lumps are charged on to the hearth up to the charging door-sill level. The arcs will then strike on to the carbon bed, and the load be controlled and properly balanced, just as when melting a charge of cold scrap. If the coke used is in small pieces, the resistance of the bed when cold may be considerable, and it will then be necessary to lower the electrodes on to the charge to effect passage of the current which generates heat by resistance. Further, since the resistance of the coke is high and by no means uniform, the position of the true neutral point in the case of three-phase furnaces may vary considerably, and balance of current through each electrode will not necessarily indicate uniform heating of the coke bed. For this reason it is sometimes best to control the preliminary heating by actual observation, the greatest heat being produced where the resistance of the charge is highest. Thus, by lowering one electrode on to the bed and raising another, the zone of heating may be transferred as desired. Directly the coke becomes heated throughout to dull redness its resistance rapidly falls and becomes uniform, and balance of current then correctly indicates uniformity of heating. Lamps, which are sometimes connected between each electrode and a common point in the furnace hearth for the purpose of roughly indicating balance of arc voltages, fail to operate at the commencement of heating owing to the high and irregular resistance of the coke bed and the low conductivity of the cold furnace hearth in which their common point is embedded. As heating proceeds, the coke bed becomes highly conductive, and longer arcs are then formed under each electrode in place of the multitudinous small arcs striking between the individual lumps of coke; resistance heating at the same time gives place to arc heating. These free direct arcs cause the electrodes to slowly bore downwards to the bottom, and, unless the holes formed are filled with fresh coke, the close proximity of the arcs, which should not be less than 9 inches from the bottom, will cause disintegration of the basic material. Such burning of the dolomite or magnesite hearth is generally accompanied by large volumes of a heavy grey-black smoke, resulting from the reduction of magnesium compounds by carbon.

The above procedure requires to be modified in the case of furnaces provided with conductive hearths, which in normal operation constitute a part of the load circuit. A basic hearth, when cold, is never sufficiently conductive to perform its proper function, and remains inoperative until thoroughly heated. In most types of conductive hearth furnaces, a load circuit can be formed independently of the hearth, and, although it is then impossible to obtain full load or proper conditions of balance, it is nevertheless possible to convert sufficient electrical energy into heat for the purpose of drying out and baking in the basic hearth with the aid of a coke or carbon bed as above described. As the bottom becomes hot, the basic material will begin to conduct current, and finally permit the proper electrical conditions of balance and load distribution to be secured. The passage of the current through the warm unbaked conductive hearth promotes internal resistance heating, which may considerably influence the normal rate at which the volatile constituents of the binder should be driven off. For this reason, together with the danger of internal local heating previously mentioned, the load is kept at a low figure, even after the hearth has begun to carry current.

In certain types of furnace, notably the original Stobie two-phase design, the hearth is an integral portion of the only possible load circuits. In this case the furnace must be preliminarily heated by gas or oil, until the hearth becomes sufficiently conductive to allow internal electrical methods of heating to be followed.

Furnaces in which metallic conductors are employed for the passage of the main current from the conductive charge to the bottom return cables belong to yet another class which requires special mention. The Snyder and Girod furnaces are the only well-known examples of this type, and in each case the metallic electrode or pole is embedded in the hearth which it penetrates. The simple method of heating electrically with a charge of coke can be again followed for the baking-in operations, but it is advisable to take precautions against excessive local heating and disintegration of the basic material surrounding the nose end of the metallic conductor. This can readily be done by placing a few magnesite bricks round the exposed end to form a short stack about 6 inches high: a short length of graphite electrode is then placed vertically within it and protrudes 2 inches or 3 inches above the level of the magnesite bricks, the whole being then covered over with coke in the usual way. On passage of the current, the contact between the graphite stub and the metallic pole piece will rapidly improve by fusion, until there is no longer serious local heating at that point, which is level with the basic hearth. The current will then naturally pass through the graphite stub to the coke in contact with it, so that the zone of heating is lifted at least 6 inches above the bottom.

Acid Linings.—The use of acid linings is at present confined to furnaces whose load circuits are entirely independent of the linings, or in which a load circuit through the hearth is composed of a metallic conductor acting quite independently of it.

The only material difference in the construction of acid and basic linings lies in the substitution of silica brick for magnesite brick, and of rammed ganister in place of dolomite. No further mention need be made as regards the brick lining, beyond pointing out that it is unnecessary to dry out the brickwork thoroughly before ramming in the hearth material. The working hearth is generally composed of ganister, which is mixed with a pitch-tar binder and rammed to shape. A mixture consisting of 75 per cent. calcined crucible ganister, 25 per cent. crushed quartz pebbles, and just enough binder to facilitate ramming, gives excellent results and is used in the same way as "black basic". The ganister should be calcined at about 1000° C. to expel water and volatile matter, which together amount to about 2 per cent. The proportions of crushed calcined ganister and broken quartz rock should be such as will form a compact mass in which all the coarser pebbles are firmly held in a solid matrix. Should metallic conductors penetrate the hearth, the ganister is rammed round them without taking any special precautions. The preliminary drying out and heating is best conducted in the same way and with the same care as for basic linings.

Furnace Lining Repairs.—The life of a furnace wall lining is usually determined by the life of the roof, which suffers most and cannot be temporarily patched in the same way as the door jambs, walls and hearth. A roof may sometimes fail prematurely owing to the use of inferior material, faulty construction, or improper treatment during the initial heating; in such cases the furnace walls may be hardly worn, and the roof can be removed without causing any damage to the body lining.

In normal circumstances a roof by the time it requires to be renewed will have become firmly fritted to the upper courses of the silica brick walls, and therefore cannot be detached without seriously damaging them, which usually necessitates a complete relining of the walls. A short life of a roof, therefore, results in unnecessary waste of refractory material, and so influences to a great extent the repair costs per ton of steel. Under satisfactory working conditions the walls and hearth should require renewal or repair at the same time as the roof, so that the maximum amount of wear will be obtained from all the refractories used.

The methods of relining basic and acid furnaces are very similar, and can be dealt with jointly.

After pouring the last heat of a campaign, the electrodes and coolers are dismantled preparatory to removing the roof, which is best done while the furnace is still red hot. This is sometimes followed by immediate demolition of the silica walls, which part readily from the top course of magnesite bricks, the brickwork being then withdrawn with the aid of long hooks. The above practice is, of course, not always followed when the duration of the repair is not of great importance. After demolition and removal of the silica brick, the furnace is allowed to cool off sufficiently for work to be proceeded with from within the furnace body. The courses of magnesite brick that have been badly eroded and require renewal are then removed, together with the entire semi-fused slaggy covering of the dolomite hearth. Particular attention is paid to this, and at the same time care is taken to remove all trace of steel remaining on the bottom or absorbed by the hearth material in porous places. All this loose material is then thrown on one side, and the brickwork and bottom brushed over with a stiff brush to remove all powder and loose material. This will enable better examination of the bottom to be made. The hearth material should, after brushing over, appear quite black and solid. Black basic or ganister should never be rammed on to steel or slag, which, on subsequent fusion, may cause the newly rammed portion to become detached; for this reason it should only be rammed on a solid foundation, which necessitates careful removal of all friable portions of the old hearth.

After the above preparation of the bottom has been completed, it is usual to rebuild the entire walls. The hearth, after being thoroughly cleaned, is then brushed over with a very small quantity of thin tar to effect adhesion of the fresh material, which is then rammed up in the customary manner. After completion of the latter operation the furnace body is ready to receive a new roof, which is carefully centred to the electrodes in cases where the latter pass through it. Refixing of the sundry accessories, such as coolers, water pipes, etc., concludes the furnace repair. The roof and furnace walls require the same careful heating up as previously described, but the time required for baking-in the newly rammed hearth may be reduced according to the quantity and depth of the fresh material rammed.

Life of Furnace Lining.—The various chemical and physical conditions which so greatly influence the life of a refractory lining have been fully dealt with in Chapter XI. These conditions are not only inherent to the metallurgical process performed, but depend very largely upon the electrical and constructional design of the furnace employed. For these reasons, the life of a furnace lining is most variable and only a general idea can possibly be given.

The basic lining of a 3 to 6-ton polyphase direct arc furnace, working with a low arc voltage and in operation day and night, should last four to five weeks before requiring renewal of the upper walls and roof. The hearth will, of course, also require considerable repairing but not entire renewal; this applies to furnaces melting and refining cold scrap. In the case of furnaces being used for only eight to twelve hours a day, the life of the lining will be certainly prolonged, but not in proportion to the diminished output; the cost of repairs per ton of steel will in consequence be heavier, representing an increase of about 10 to 20 per cent.

CHAPTER XV.

PROPERTIES AND MANUFACTURE OF CARBON ELECTRODES.

In all electro-chemical and electro-metallurgical work dependent upon electrolytic or electro-thermal action, the term "electrode" is applied to any terminal conductor which conveys electrical energy into an apparatus in a manner consistent with a particular mode of current distribution. The material under treatment may be solid, liquid, or gaseous. In all metallurgical arc furnaces there are at least two electrodes, one of which must be adjustable and electrically insulated from the furnace body and earth. For this purpose a movable electrode is always used as one terminal of the arc, and is, therefore, exposed locally to the highest arc temperatures.

As regards arc furnaces, the materials used for electrode manufacture must either resist physical change at arc temperature, or must be water-cooled to prevent destruction. In the latter case, the loss of heat is far too great for the economical operation of metallurgical furnaces, so that carbon can alone be used for electrodes other than those which are not under the direct influence of the arc. Carbon exists in both amorphous and crystalline states, which have very distinct physical properties.

Amorphous Carbon.—There are numerous varieties of amorphous carbon, both natural and artificial, of which those usually employed in electrode manufacture are:—anthracite, gas retort carbon, pitch coke and petroleum coke. Their density, porosity, and electrical conductivity vary, and the properties of an electrode made from these materials will depend largely upon the proportions in which they are present and their physical condition when used.

Anthracite.—High grade anthracite selected for electrode manufacture contains over 90 per cent. fixed carbon, the remainder being volatile matter and ash. Below are given the analyses of a suitable grade before and after calcination.

	Before Calcination.	After Calcination.
Fixed carbon	91·8	95·81
Volatile matter	5·3	1·4
S and P	·8	·89
Ash	1·75	2·0

Anthracite is hard, brittle and shiny, and leaves no mark when rubbed on paper.

Gas Retort Carbon.—This form of carbon is a product of coal distillation, and is deposited on the lining of gas-works retorts as a hard crust up to 2½ inches thick. It is finely porous, hard, varies in degree of density, and has a high electrical conductivity. It is usually crushed and ground in mills, and, owing to its great toughness, the small angular fragments are liable to become rounded. A typical analysis of a dried sample is as follows :—

Fixed carbon	94·54
Volatile matter	·76
S and P	1·3
Ash	3·42

Petroleum Coke.—This material is the residue left after the complete destructive distillation of crude mineral oils. It is light and porous, and is nearly pure carbon containing a small percentage of volatile matter. It is readily converted into graphite, and is therefore used for the manufacture of electrodes suitable for graphitising. Owing to its freedom from mineral impurities it is also extensively used in the manufacture of electrodes for aluminium reduction.

Pitch Coke.—This is the final retort residue resulting from the destructive distillation of coal tar. It resembles petroleum coke in character, and usually contains sufficient volatile matter to render a preliminary calcination necessary.

Crystalline Carbon.—*Natural Graphite.*—This is a crystalline form of carbon which is better known as "plumbago". It is usually contaminated with mineral matter, which in the lower grades rises up to 30 per cent. The finest Ceylon flake graphite contains as much as 99·8 per cent. carbon. Natural graphite is very soft and is a good conductor of both heat and electricity.

Artificial Graphite.—Artificial graphite was discovered in 1896 by E. G. Acheson, who found that amorphous carbon could be converted into graphite under the influence of prolonged heating at temperatures exceeding 2000° C. This process was at once developed for graphitising moulded articles of amorphous carbon. Acheson considered that the presence of small quantities of mineral matter was beneficial rather than injurious to the promotion of the physical change, but recent investigations have shown that the transformation is dependent upon the variety of amorphous carbon used rather than upon the impurities present. An article of high degree of purity, containing 99·5 per cent. carbon can be easily made by this method from the less pure amorphous varieties, the impurities being volatilised during the process of conversion.

Artificially graphitised articles are very soft and can be easily machined to exact sizes. The electrical conductivity is about four times that of amorphous carbon when cold, and increases but slightly at high temperatures.

Arsem defines graphite, both natural and artificial, as an allotropic form of carbon, having a specific gravity of 2·25 to 2·26.

Amorphous Carbon Electrodes. — *Preparation of Raw Materials.*—The several varieties of amorphous carbon previously mentioned have all been used for the manufacture of electrodes, although the present day tendency is to use a preponderating quantity of anthracite, and in some cases, to eliminate pitch coke entirely. Whatever mixture may be chosen, the preliminary treatment before mixing with the binder applies equally in all cases. The volatile matter present in the solid constituents of amorphous electrode mixtures should not exceed about 1·0 per cent., so that a preliminary calcination at high temperatures is always essential.

The grain size of the crushed materials is also of great importance and is carefully controlled; it is, however, largely dependent upon the character of the materials used. Retort carbon, owing to its extreme hardness, has a tendency to pulverise by abrasion, the larger grains becoming rounded as a result; this is a disadvantage, as the cohesive power is less than for sharp angular particles. Pitch and petroleum cokes are friable and, owing to their extreme sponginess, require to be broken down to a fine state of division. Anthracite can be easily crushed to a correct and uniform size without making too large a proportion of fines, and, for this reason alone, is superior to the other materials. It is usually calcined in specially designed cupolas, where by partial combustion it provides the necessary heat for the elimination of volatile matter. In certain localities, where electric power is cheap, the calcination is performed in electric shaft furnaces, designed so as to make the process continuous, as in cupola practice.

Binders.—Straight-run gas-works pitch is generally used for binding together the various carbon materials in the pressing operations. It is kept at a fixed standard of viscosity and softening point by constant examination, and the addition of hard pitch or tar.

Outline of Manufacture.—The prepared and preheated carbon materials are mixed in pan mills in the correct proportion, with about 11 per cent. of pitch binder, drawn direct from a special boiler. The mills are heated to prevent any chilling of the paste and to maintain the desired degree of plasticity up to the moment of passing to the moulds. The so-called "paste" is slowly transferred to heated moulds, where it receives a preliminary ramming under a mechanical ramming head. This enables a sufficient quantity of paste to be charged into the mould for a given length of electrode, and avoids air cavities. The mould is then placed under a press head and the paste subjected to a pressure of about 1 to 1½ tons per square inch, which is maintained for a few minutes. After stripping the moulds, the "green" electrodes are placed vertically in the stalls of a baking stove and packed tightly with sand after bricking up the front wall of each stall. This packing prevents oxidation and gives support to the tender electrodes while the first portion of the volatile matter is being driven off. Extrusion presses have been recently applied with satisfactory results in the manufacture of even the largest sizes of amorphous electrodes. Electrodes formed in this way have, of course, to be machined and bored afterwards at both ends for the screwed sockets.

The baking operation needs considerable care and is controlled by recording pyrometers. The heat is applied very slowly to prevent too rapid evolution of volatile matter, which may either burst or distort the electrodes. The full furnace temperature of 1200° C., which is only reached after seven days' firing, is kept fairly constant for a day or two, after which the fires are drawn. It usually takes about ten days for the kiln to cool off sufficiently for the removal of the baked electrodes, and should the sand packing be drawn prematurely, there is risk of the electrodes burning. This duration and method of cooling refers to the above-mentioned type of furnace, comprised of several stalls heated by flues built in the dividing walls, and each containing a large number of electrodes. In America the "green" electrodes are packed with sand or fine coke breeze in separate mild steel boxes, which can be withdrawn from the stoves while still hot, without any risk of burning the electrodes. This latter method of charging the "green" electrodes is to be preferred, as there is far more opportunity for the full furnace temperature to penetrate to the heart of each electrode, and thus ensure more perfect removal of volatile matter. The cost of the mild steel boxes, however, renders their use prohibitive in many cases.

The baked electrodes are cleaned, and their end faces and screw threads trimmed if necessary. An electrical conductivity test is also applied before dispatching.

The screw nipples are made in substantially the same manner, being pressed in split moulds and at a slightly different pressure.

Defects of Amorphous Electrodes.—*Fracture.*—The most serious defect of amorphous carbon electrodes is their frequent inability to withstand sudden subjection to high temperatures without cracking and subsequent fracture. In the case of arc furnaces using vertically suspended amorphous electrodes, there is always some occasion when a new electrode is exposed to the interior of a hot furnace. Should failure result the defective electrode is replaced by another, which is then exposed to the same conditions and may probably behave in like manner. When electrodes fail in this way, even after slow heating to a full furnace temperature, no preliminary precaution will suffice to remedy the inherent defect.

Fractures of pressed electrodes occur along transverse or inclined irregular planes, which do not necessarily mark the junction of zones of unequal temperature. This fact would seem to disprove any theory based upon the setting up of unequal mechanical stresses by irregular heating of different transverse sections, which, as in the analogous case of a silica brick, results in fracture or "spalling". Experience also shows that fracture only occurs after the electrode has attained a bright yellow heat, although a skin crack may be sometimes seen at lower temperatures. It is probable, therefore, that internal forces are set up at a temperature which is certainly higher than that at which mechanical stresses, due to unequal heating of the skin relative to the centre, are likely to cause fracture. Unequal and sudden expansion in the case of very dense and fine-grained electrodes may cause fracture, but it is undoubtedly the presence of volatile matter in the baked electrode that is the primary cause of failure.

Screw joints are frequently a marked source of weakness, and sometimes break without any blow being given to the electrode. This type of screw failure usually occurs when the joint is only dull red or even black hot, and very seldom when it shows considerable resistance heating. Failure in this case is probably due to unequal expansion of the screw and the electrode sockets, especially when the joint has been too tight.

Disintegration.—The resistance of the skin of an electrode to combustion and disintegration has an important influence on its life. Combustion of the smooth skin usually commences at a dull red heat and, when once started, promotes more rapid disintegration of the matrix, which leaves the coarser grains standing out and exposes a larger surface to oxidation. Combustion of the coarser grains proceeds slowly and disintegration might be arrested, were it not that these grains are easily detached and a fresh surface then exposed to oxidation.

Local Heating at Joints.—Amorphous electrodes, when joined together by a screw nipple, frequently show local resistance heating, extending over a region 5 or 6 inches above and below the joint. When the joint is raised above the combustion temperature of the carbon, disintegration follows, which may cause very considerable "necking". This, however, is seldom accompanied or followed by the fracture of a nipple, but results in undue electrode consumption. The screw socket walls may become thinned by disintegration to such extent that they are no longer strong enough to support the weight of a short stub end, which would otherwise be consumed before such a condition of the socket walls was reached; for this reason short stub ends will be lost, and, if not purposely removed beforehand, may fall into a bath of steel and cause considerable metallurgical difficulties and delays.

Surface Irregularities.—The larger sizes of amorphous electrodes after removal from the moulds are packed vertically in the baking furnaces and, since they support their own weight, the bottom end will accommodate itself to any unevenness of the supporting tile or floor during the process of baking. The bottom may easily be distorted from a plane at right angles to the axis of the screw socket; this is a serious matter in joining, and may often lead to broken nipples, owing to the side strain exerted upon them when screwing the electrodes together. The nipple will not necessarily break when cold, but may do so later in service, if a small crack or fault has been once developed within it at the time of joining.

Surface irregularities in the screw socket are also troublesome and are generally removed at the electrode factory before dispatch.

Defects in the form of surface deformations are occasionally found, and may give rise to considerable trouble when lowering the electrode through its holder, and by causing an uneven contact surface.

Mechanical Disintegration by the Arc.—This defect is not common, but when apparent sometimes causes metallurgical difficulties. The coarser particles of the electrodes are mechanically detached from a loose matrix by the disruptive action of the arc; the ejected particles float on the slag and reduce its powers of oxidation.

Chemical and Physical Characteristics.—*Volatile Matter.*—Volatile matter is responsible for the majority of failures by fracture. The manner in which faulty electrodes break up has been already described, and it was explained that fracture could hardly be accounted for by the unequal expansion of the skin relatively to the core. In an investigation to determine the cause of bursting, sufficient evidence was obtained from the behaviour of both good and bad electrodes to attribute definitely the cause of failure to excessive quantities of volatile matter. The determination of volatile matter in baked electrodes and raw carbon materials by the standard chemical method is more comparative than absolute, and the results obtained cannot be accepted with confidence. A modification of the standard method, embodying the use of vacuum furnaces, may yield reliable results, but is too elaborate for ordinary works practice. Reliable results may, however, be obtained by a crude but sufficiently accurate method, which merely consists of heating a large crucible containing 1 kg. of the finely powdered dry material in a small assay wind furnace for 2 to 3 hours. A lid is carefully luted on beforehand, and is not removed until the crucible has become cold. The powdered material is then transferred to a physical balance and weighed. Any error through loss in transfer is insignificant, and reheating in the furnace will not produce any further loss of weight. The temperature of the crucible should be at least 1200° C., and is therefore more comparable with working conditions.

It is to be naturally expected that the rapid heating of an electrode will cause sudden volatilisation of any volatile matter at a high temperature, which, moreover, cannot readily escape if the electrode is very dense and compact. The gas then exerts high internal pressures, which will eventually cause fracture along any plane of weakness. Faulty electrodes, when used in open top ferro-alloy furnaces, burst off in slices, which have become heated to a temperature well above that to which the electrodes were exposed during the process of baking; small flames may also be seen issuing from cracks in cases where actual rupture has not occurred. The volatile matter should not exceed 1·75 per cent. in a baked electrode, and should be below this figure if the body of the electrode is very dense and compact.

Grain Size and Porosity.—The crushing and grinding of the raw materials before mixing is always carefully controlled to ensure uniformity of the electrode structure. The difficulty of preventing pitch or petroleum coke being over-pulverised during the grinding operation has already been mentioned, but this difficulty does not occur to such an extent with anthracite, owing to its toughness and tendency to break along definite cleavage planes. The ideal sizing analysis for a very dense electrode would be such that the interstices between the largest particles contiguous with one another would be filled with the minimum number of particles successively of smaller size, so that the amount of dust required to fill the very smallest interstices would be reduced to a minimum. It is, of course, impossible to achieve such conditions in practice, and, in fact, it is better to avoid making a proportion of fines which would satisfy such a case. A certain degree of porosity is necessary, as it undoubtedly assists the escape of volatile gases during baking operations and reduces the possibility of fracture in service by releasing any pressure exerted by a rapid internal generation of gas.

The porosity of a baked electrode is calculated from the actual and apparent specific gravity figures and is usually about 25 per cent. The grading analysis of the dry crushed material for this degree of porosity should not show more than 20 per cent. capable of passing a 40 mesh sieve. The above figures, in both cases, refer to electrodes that are composed entirely of anthracite mixed with only 11-12 per cent. of binder. The degree of porosity is limited by the increase of electrical resistance and the greater tendency of the electrode to burn and disintegrate, and should be only high enough to permit the free escape of volatile matter during the baking process and to minimise risk of failure by further expulsion of volatile matter in service.

Ash.—The percentage of ash is not of primary importance, as within the normal degree of variation it in no way affects the mechanical strength of electrodes and does not make any appreciable difference in electrode consumption. English anthracite electrodes generally contain less ash than American, owing to the greater purity of the anthracite used, and will usually show about 2·5 per cent. on analysis.

Electrical Conductivity.—The electrical conductivity of an amorphous electrode is not in itself a matter of great importance, since the heat generated by resistance in an electrode of proper dimensions is small compared with the heat conducted through it from the interior of a hot furnace. Assuming there is no internal resistance heating, then the temperature of the electrode at a given distance above the roof will depend upon the thermal conductivity of the carbon material, and the rate at which heat is absorbed by the portion inside the furnace and radiated to the atmosphere from the exposed portion above the furnace roof. On the other hand, if the normal resistance heating due to the passage of a heavy current were the only source of heat influencing the temperature of an electrode outside the furnace, it would be found that the actual temperature rise at various points is far lower than it is in actual practice, where the effect of thermal conductivity outweighs that of electrical resistance heating. This argument, it must be understood, does not apply to electrodes when deliberately overloaded, and to screw joints which often exhibit local heating.

An electrical conductivity test is made not so much for the simple determination of specific resistance as for the purpose of indicating whether the baking operation has been successfully carried out. The variation of the resistance is never great in electrodes made of the same materials and under the same manufacturing conditions, but any small difference that may be found is considered to be due to the varying extent by which the carbon grains are bound together. If the baking is not carried sufficiently far the volatile constituents of the binder are improperly removed, and the added mechanical adhesion of the particles due to the final coking of the non-volatile residue of the binder is not attained. It has been explained how all serious failures of electrodes are primarily due to the presence of volatile matter, so that a rapid test which will give an indication of their behaviour is of great value. In some electrode factories all electrodes that fail to come up to a definite standard of conductivity are rejected and returned to the baking stoves. The specific resistance is about ·00124 ohm per inch cube, or ·00315 ohm per centimetre cube.

The apparatus used for conducting the resistance test is shown diagramatically in Fig. 127. The method is based on the voltage drop between certain fixed points along an electrode when a known current is passing through it. To save constant calculation the current is adjusted to definite values by means of a variable resistance, and the distance along which the voltage fall is measured is also kept at a definite figure, according to the diameter of the electrode. A "constant" for each set of conditions is usually worked out, and this figure, multiplied by the millivoltmeter reading, gives at once the resistance either in ohms per inch or centimetre cube.

Graphite Electrodes.—*Manufacture.*—The electro-thermic process for converting the several forms of amorphous carbon into graphite has already been mentioned, but it might be added that anthracite is difficult to convert, whereas petroleum or pitch coke is more easily transformed than any other variety of carbon.

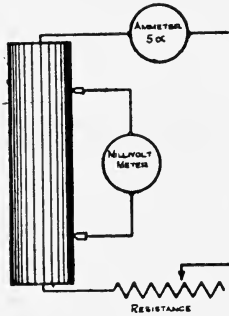

Fig. 127.

The electro-thermic process for converting amorphous carbon into graphite has been generally used for the manufacture of graphite electrodes. Recent attempts, however, have been made to use natural graphite, and for this purpose a special method of purifying crude impure plumbago is now being developed commercially in Italy. Artificial graphite electrodes are made by subjecting amorphous carbon electrodes to a prolonged heating at a very high temperature in specially designed resistance furnaces.

The amorphous carbon electrodes, consisting almost entirely of petroleum coke, are formed by extrusion, cut off to length, and undergo the usual baking treatment. These electrodes are then carefully packed in coke breeze on the furnace hearth to form a rectangular pile between heavy carbon terminal blocks. The whole is then covered with more breeze and, finally, a refractory covering. A current of very high density is passed through the charge from end to end, and the voltage varies as conversion proceeds. Heat is generated internally by electrical resistance which enables the critical temperature of conversion, said to be about 2000° C., to be easily reached. The resistance of the entire charge falls until conversion to graphite is complete, which serves as the only indication of the progressive change taking place. The furnace takes several days to cool off sufficiently for demolition. The graphitised electrodes are carefully cleaned and sent to the machine shop, where they are socketed for screw nipples and faced up true at both ends. The ease with which graphite can be machined is a distinct advantage, as the electrode joints can be made far more perfect than with amorphous electrodes. The commonest sizes for steel furnaces are 4 inches, 5⅛ inches, 6 inches, 9 inches, and 12 inches in diameter.

Chemical and Physical Characteristics.—The raw materials from which graphitised electrodes are made are themselves pure forms of carbon, containing, in the case of petroleum coke, less than ·2 per cent. ash. Whatever the mineral content may be, it is almost entirely volatilised during conversion, and the resulting electrodes will always contain over 99·5 per cent. of graphitic carbon. Graphitised electrodes are tough and resist combustion far better than other varieties. The specific electrical resistance is low, being only about ·00030 ohm per inch cube, but the thermal conductivity is correspondingly high.

Electrode Joints.—Electrodes which can be fastened together by means of a simple male and female screw joint are now universally used for the operation of steel furnaces. The electrical resistance between the threaded surfaces of the nipple and socket is always considerable, so that the safe current-carrying capacity of the joint will largely determine the most suitable diameter for an electrode. Jointing should be done with considerable care, as upon this will greatly depend the life of an electrode. Since the screw threads of most amorphous carbon electrodes are press formed, considerable clearance has to be allowed to ensure an easy fit; this, however, does not apply in the case of machined graphite electrodes. Different methods of joining are therefore adopted to meet both cases.

Amorphous Electrodes.—Modern electrodes are manufactured with screw sockets at both ends, and are joined together by male screw nipples, which are rather shorter than twice the depth of a socket. To ensure good electrical contact between the threaded surfaces, it is necessary to use a highly conducting cement, which must be sufficiently plastic to flow under uniform pressure at all points. The cementing material, which has given the best results in practice, consists of very finely pulverised graphite, mixed with a pitch-tar binder to give a consistency of treacle at ordinary temperatures. This paste is well smeared over the contact surfaces before screwing a nipple into the screw socket. If the paste does not adhere to the carbon, the surface should be brushed over with the least possible amount of thin tar before application. Having securely screwed the nipple into the upper socket of the lower electrode in this manner, the upper electrode, with its lower socket already prepared with the graphite paste, is brought immediately above it; the butt ends and the protruding half of the nipple are then liberally smeared with paste before screwing down. The upper electrode should be screwed down just tight enough to squeeze out the paste from between the butt end faces, and should not be jerked forward to secure an exceedingly tight joint. Experience proves that nipples show the least tendency to break when the electrode joint is not absolutely rigid; this seems to indicate that there is unequal expansion of the nipple and electrode, which can only be satisfied without fracture, if the joint is not excessively tight. Loose joints, on the other hand, result in poor electrical contact and excessive local heating, which may cause the loss of an unconsumed stub end owing to excessive combustion and consequent thinning of the socket walls.

It is not always an easy matter to join ponderous amorphous electrodes, and it requires considerable skill and judgment on the part of a crane-driver to hold the weight of the electrode and lower it at exactly the same rate as it is fed downwards by screwing on to the nipple. If the electrode is lowered too fast, the full weight is carried on the thread of the nipple and, even if no damage is done to it, further screwing is difficult; on the other hand, if the electrode is screwed faster than it is lowered, it will bind on the thread and may fracture the screw nipple if the lower electrode is rigidly held. To overcome this difficulty and facilitate joining, a special screw plug (Fig. 128) has been used, consisting of a central plug screwed inside an outer plug which fits into the electrode socket. The central plug is suspended from the crane-hook and its threads are cut to the same pitch as the socket thread. With this simple appliance, the vertical feed downwards of the suspended electrode is exactly correct at all times for the degree of rotation. The plug should be provided with a safety locking device, which is only released when the electrode is just above the nipple and ready to be screwed on to it.

It has been proposed to improve the conductivity of the cementing paste by the admixture of metallic filings, but it is doubtful if any advantage is gained, as a suitable paste made with a high grade graphite, and not ordinary impure plumbago, is a better conductor than the electrode material.

In many works it is preferred to join a new electrode to another without removal of the short piece from the holder, but when the headroom between the electrode-holder and the crane-hook will allow, it is certainly quicker and more advisable to have a jointed electrode ready for replacing a short length. In this way the join can be made more carefully than is otherwise possible on a hot furnace roof.

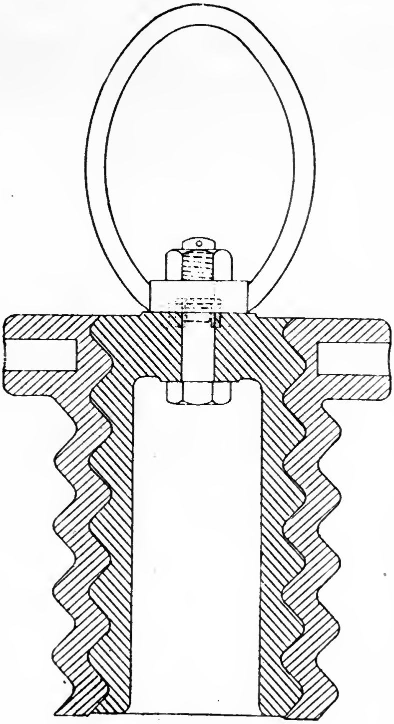

Fig. 128.

Amorphous electrodes are pressed with a $\frac{1}{2}$-inch taper to a six foot length, so that care is therefore always taken before joining to arrange for the larger diameter to be uppermost. This precaution is necessary to prevent an electrode slipping through a large holder which at times may expand sufficiently to relax its grip.

Graphite Electrodes.—Graphite electrodes are now universally joined by using a male nipple which screws into female sockets. The screwed surfaces are all machine fits, so that perfect joints can be made without resorting to the use of cements or dry powder. The usual practice is simply to screw the two electrodes together until they just touch; the least strain to secure a very tight joint invariably causes the nipples to break either at the time of joining, or, what is far worse, later during service.

Storage.—Amorphous electrodes, owing to their considerable porosity, should always be stored under cover, and should be raised clear of any floor which is likely to become damp. When possible, they are best stored near a drying stove or annealing furnace.

Current-Carrying Capacity of Electrodes.—The safe current-carrying capacity of amorphous and graphite electrodes cannot be calculated from purely theoretical figures, and must be based upon the results of practical experience. Sufficient has already been said to indicate the weakness of the screw joint and the trouble that may ensue from excessive contact resistance and from the increased current density in the nipple due to poor contact between the butt ends of the electrodes.

The screw joint, therefore, must be regarded in the same light as the weakest link in a chain, and will determine the smallest and most economical size of electrode which can be used. If the screw joint could be ignored, the most economical diameter to select, from a purely theoretical standpoint, would be such that half the rate of internal heat generation by electrical resistance equals the rate at which heat is normally conducted from the hot furnace by the electrode and radiated to the atmosphere. For practical purposes the following figures may be regarded as typical of the diameters of amorphous and graphite electrodes commonly adopted for various current-carrying capacities, due allowance having been made for resistance of joints:—

Current in Amperes.	Amorphous.	Graphite.
2000-3000	14 inches	$5\frac{1}{8}$ or 6 inches
3000-4000	14 ,,	6 inches
4000-5000	16 ,,	8 ,,
5000-6000	16 ,,	8 ,,
6000-7000	18 ,,	9 ,,
7000-8000	18 ,,	9 ,,
8000-9000	20 ,,	10 ,,
10,000-12,000	20 ,,	12 ,,

Comparison of Amorphous and Graphite Electrodes.—All users of electric steel furnaces are confronted with the question of the relative merits of amorphous and graphite electrodes. A few years ago amorphous carbon held the field, but recently results with small graphite electrodes have shown such remarkably low electrode consumptions per ton of steel that it is not surprising to find a growing demand in their favour. Both types have their distinct advantages and disadvantages, and it will be as well before considering these in detail to compare briefly their physical properties as given by the Acheson Graphite Co. in one of their publications:—

	Amorphous Carbon Electrodes.	Graphite Electrodes.
Specific resistance ohms per in. cube	·00124	·000320
Specific resistance ohms per cm. cube	·00315	·000813
Comparative sect. area for same voltage drop	3·8	1
Weight: lb. per cube in.	·0564	·0574
Weight: grms. per cu. cm.	1·56	1·59
Density	2·00	2·21
Tensile strength, lb. per sq. in.	1000 to 5000	800 to 1000
Temp. of oxidation in air	500° C.	640° C.

Apart from the relative merits indicated in the above table from a purely physical standpoint, there are others, which, although not apparent from actual electrode consumption, may indirectly influence the total manufacturing cost per ton of steel. For this reason an electrode consumption figure, which may be actually very small, should not always be regarded as an indispensable factor towards economy of production. It is obviously worthless to draw a comparison between the electrode consumption of two furnaces of equal load capacity, one having perhaps three amorphous carbon electrodes and the other only one of graphite. In this case the undoubted economy of electrode consumption in favour of the graphite electrode is due solely to factors which are entirely independent of the relative behaviour under exactly similar conditions. Therefore, to draw a true comparison between the two varieties, it is necessary to use them in furnaces of similar design operating under the same working conditions.

Effect of Combustion.—Consumption of electrodes by natural combustion inside and outside the furnace is naturally greater in the case of amorphous carbon. For the same current-carrying capacity, the diameter is larger, and to this must be added the far greater tendency of the amorphous carbon grains to burn. Hot joints are far more frequent and pronounced with amorphous electrodes, and sometimes account for considerable wastage both by combustion and loss of stub ends. Graphite electrodes are not tapered, and, owing also to their comparatively small size, are more readily adaptable to the several gas sealing devices now used, and also to the cruder method of blocking the annular opening by the liberal use of ganister. For this reason, the combustion of graphite electrodes above the roof may be more readily prevented than in the case of amorphous carbon.

Loss by Fracture.—Amorphous carbon has a higher tensile strength than graphite, and for equal current-carrying capacity requires a sectional area at least four times as great. Therefore, an amorphous electrode of equivalent conductivity would be at least four times stronger than graphite, and offer far less risk of fracture by accidental blows received during manipulation of the furnace. This is, of course, assuming the amorphous electrode to be of good quality and not subject to the defects due to improper grain size and volatile matter.

The amorphous electrode screw-joint has a liberal margin of strength and will usually withstand any light blows given to the lower end of the electrode. On the other hand, the weakness of the screw-joint of graphite electrodes is often a serious drawback; this results from the small diameter of the nipple, which, although strong enough to support the weight of the stub, is easily fractured by side blows. To prevent a high consumption with graphite electrodes it is essential, therefore, to charge the furnace with considerable care, more especially when heavy, irregular shaped scrap is being used. The danger of a nipple breaking is not so apparent in the larger sizes of 9 inches diameter or more as in those of small diameter.

The fracture of a joint always causes a delay occasioned by removal of the broken piece from the furnace and replacement of a new length; sometimes also a bath of steel may be carburised by contact with the electrode, in which case the "heat" will be further delayed by boiling down. Such difficulties are common to both types, but there is a slight advantage in favour of graphite electrodes, which, by reason of their much smaller diameter, can be removed more expeditiously and with less risk of disturbing the metallurgical conditions. Apart from the difficulties of removal, the fracture of a nipple joining two amorphous electrodes results in complete loss of the stub-end. This is not necessarily the case with graphite electrodes, since, with a little care and patience, the halves of the broken nipple can often be removed from their screw sockets and the same electrodes joined together again by a fresh nipple.

Effect of Weight and Size.—Graphite electrodes, by virtue of their lighter weight and smaller diameter, cause less trouble with holders than the heavy amorphous electrodes. A reduced diameter is also a point of great advantage in roof construction and its ultimate life. On the other hand, amorphous electrodes of large diameter shield the roof to a far greater extent from the heat directly radiated from the arc, which is a point of considerable importance, and has a great bearing upon the life of a furnace lining and the cost of repairs. Excessively high roof temperatures generally lead to metallurgical difficulties, more especially in basic lined furnaces, and this introduces a further element affecting the one figure which can alone be indicative of comparative economy, namely, the over-all manufacturing cost of producing one ton of steel in the ladle.

Relative Cost.—The question of the relative economy of graphite and amorphous electrodes is very involved, and can only be satisfactorily settled by a comparison of the manufacturing costs obtained from two furnaces of the same type, operating under similar conditions, but fitted with the two varieties of electrodes. Figures of electrode consumption per ton of steel convey little beyond a comparison of the standard of excellence attained in different furnaces of similar design, operating under similar conditions.

If the relative cost of amorphous and graphite electrodes is based solely on the consumption per ton of steel, it will be generally found that the cost is about the same in either case, provided economisers are used with the graphite electrodes. The market prices of the two products are still, however, very variable, and until conditions become more settled, it is not possible to say which will eventually be the cheaper form to use.

Before the war the principal source of amorphous electrodes was Germany, though works were in operation in the United States, Sweden, France, and Italy. Graphite electrodes were only produced in the United States. During the war four amorphous and one graphite electrode factory were erected in England, the former having a total capacity of about 16,000 tons and the latter of about 1000 tons per annum. A large proportion of the amorphous electrodes are used for the manufacture of ferro-alloys and carbide, and of the graphite electrodes for the electro-chemical industry. Great Britain, accordingly, now produces not only the greater part of the electrodes consumed in this country, but exports a large tonnage to Scandinavia and other countries.

Electrode Economisers.—It must be acknowledged that the escape of hot and sometimes combustible gas through the annular openings between electrodes and the furnace lining is an objectionable feature of furnace operation. Not only is the consumption of electrodes augmented, but the life of the lining is also influenced. There are, therefore, great advantages to be gained by effectively sealing the electrode openings, some of which have been already mentioned in connection with electric furnace design.

The problem is far more difficult than would appear at first sight and is materially affected according to whether amorphous or graphite electrodes are employed. The former type when mould pressed is tapered, so that, apart from the effects of surface combustion, the annular opening will be constantly varying in width. Local heating in the neighbourhood of joints is a common occurrence, and often results in combustion to a marked degree, even while the joint is still outside the furnace lining or cooling jacket. Again, the screw sockets into which the male nipples are screwed are moulded in the electrodes, and, as ample clearance has also to be allowed between the threads of the socket and nipple, it is not unusual to find an overlap of a $\frac{1}{4}$ inch at the joint between two electrodes.

On the other hand, graphite electrodes, which have been originally formed by extrusion, are cylindrical and comparatively free from surface deformations. The sockets and nipples are machine threaded to make a perfect fit, and, if the electrodes are carefully mounted when boring there is no reason for any overlap in joining two electrodes together; in practice, however, there is sometimes an overlap not exceeding one-eighth of an inch. Although local heating at the joints is by no means infrequent, the diameter will not be visibly reduced by combustion, at all events until the joint is situated well within the furnace.

There are two possible methods of sealing the annular opening between an electrode and the furnace body:—

I. By the use of a sealing sleeve, gland, or collar.

II. By totally enclosing the electrode within a flexible jacket connecting the furnace body to the electrode holder.

Whatever device may be used, it need not entirely prevent the escape of gas, provided that its volume and temperature are both reduced to limits which will render it incapable of causing excessive combustion of the electrode, or of harmfully affecting the life of the lining.

To appreciate the exact nature and extent of the duties which a satisfactory economiser is called upon to perform, it is first necessary to understand clearly the manner in which combustion of the electrodes is promoted by the escape of gases through unrestricted annular openings, and from other causes. Combustion may proceed both inside and outside the furnace body. In the former case, the annular passages act like chimney flues and so induce currents of air, which enters the furnace through loosely fitting doors and, becoming heated, rises upwards and finally escapes from around the electrodes. Combustion, therefore, proceeds over the entire surface exposed to the action of the air currents, and is naturally intensified at the points of escape where the velocity of the gases is at a maximum. It is evident then that simple combustion caused by the oxidising action of such induced currents of air may be minimised by carefully sealing all door openings—which is difficult in practice—or by preventing the escape of gas through the electrode openings.

The combustion of an electrode over an extended region outside the furnace body may be caused by the escape of combustible furnace gases, which, burning at their point of issue, heat the electrode above its combustion temperature. Apart also from the heating effect of such gases, an electrode, by virtue of its thermal conductivity, or by reason of its slow or rapid withdrawal from a hot furnace, or of electrical resistance heating, may be at a temperature above that of combustion for a distance of several inches beyond the furnace body or cooling jacket; for this reason it becomes necessary to protect such a highly heated zone from the action of the surrounding air.

The essential features of a satisfactory economiser can now be enumerated :—

I. The annular space between the furnace body and the electrode should either be closed, or so restricted that the volume of gas capable of escape at any time is too small to induce harmful currents of air through the furnace.

II. If the annular passage is not entirely closed, the gas still capable of escape should be cooled during its passage through the economiser to a temperature below that of combustion before finally mixing with the surrounding atmosphere.

III. Highly heated portions of an electrode outside the furnace lining should be protected from the surrounding atmosphere.

IV. The apparatus should be inexpensive and capable of rapid and easy removal.

V. No added difficulties should be encountered in the event of an electrode joint breaking at a time when the lower electrode does not fall clear of the roof and so cannot be removed except by withdrawal outwards through the roof openings.

VI. The apparatus should be self-adjusting and its operation entirely independent of the relative positions of the electrode axis and the roof, which are constantly changing. In the case of a roof that rises badly, the surface of the cooler, originally normal to the axis of the electrode, will often be inclined at an angle of 10° or 15°.

VII. Weight should be reduced to a minimum to avoid added pressure on the roof as far as possible.

VIII. The total height of the apparatus should on no account seriously reduce the effective travel of an electrode as originally provided for.

IX. The apparatus should be constructed so as to withstand moderately rough usage, and its life should be compatible with its initial cost.

X. Simplicity should be aimed at and complexity of parts avoided.

XI. Overlapping joints, hot joints, surface deformations and "necking" at joints should be quite incapable of causing damage to the economiser, although in the latter case its effective operation may be somewhat impared.

Several types of economisers have been introduced, but it must be admitted that perfection has not been reached, and probably never will be unless certain desirable features of furnace construction are entirely subordinated to the one paramount object of sealing the electrode openings.

The desirability of closing the annular openings was realised by the earliest furnace designers, the use of the simple water-cooled ring being introduced by Heroult as early as 1903. These cooling rings, which have been so universally adopted, are constructed to allow a minimum clearance of about three-quarters of an inch around the electrode. They are always very shallow, being only about 3 inches high, and so do not offer any material resistance to the passage of escaping gases.

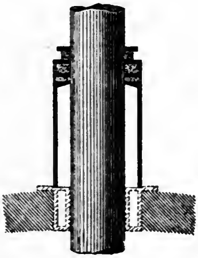

FIG. 129.

Stobie in 1916 made further attempts to restrict the passage of gases, and also to prevent the combustion of an electrode in that zone where the temperature is sufficiently high to promote it. The economiser which is shown in Fig. 129 consists of a tubular metal jacket resting upon the roof and enclosing the electrode. The height is so determined that the portion of the electrode projecting above the jacket is never likely to become heated above the temperature of combustion. The upper extremity of the annular opening is closed by suitable packing, which rests loosely upon the jacket; any gas that may pass through the loosely fitting gland will be below combustion temperature and so rendered harmless. This simple apparatus is effective, provided the diameter of the electrode remains fairly uniform. A reduced diameter at the gland will allow more gas to escape; this can only increase the rate of combustion of the electrode inside the furnace by causing a greater indraught of air, unless the gas itself burns at its point of exit, when external combustion as well would be promoted. The clearance between the electrode and the jacket is considerable, which allows for moderate displacement of the latter without interfering with its effective operation. The height of the economiser is such that it cannot be applied to existing furnaces having a limited range of electrode travel. The Stobie furnace is designed with a rather more elevated raising gear, so that ample electrode travel is provided for. Excellent results have been obtained and, according to Stobie, the electrode consumption per ton of steel has been reduced to 6 lb. with a Stobie furnace having four graphite electrodes. An economiser of this design is far more effective with graphite than with amorphous electrodes, and will not satisfactorily meet the conditions raised by high local heating at the joints between those of the latter variety. Stobie has also designed another type of economiser, which takes the form of a telescopic jacket entirely enclosing the electrode between the furnace roof and the holder. All those objections inherent to the gland type, such as overlapping joints, "necked" joints and natural taper, at once disappear, and, furthermore, this design is more adaptable to existing furnaces having a limited electrode travel, the minimum height of the telescopic jacket being less than that of the tubular type.

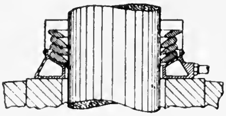
Fig. 130.

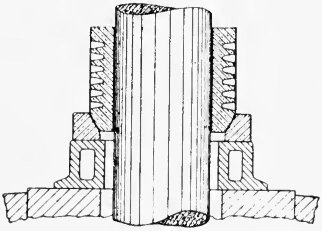
Fig. 131.

Several attempts have been made to produce a simple shallow type of economiser, acting on the principle of a flexible, self-adjusting gland, which could be used on any furnace whose design does not embody a specially elevated electrode raising gear. The economiser shown in Fig. 130 was introduced for use with the Electro-metals furnace. The apparatus consists of four separate glands, each gland being composed of three iron segments, which are free to move independently of one another, and always tend to butt against the electrode. The construction of the glands imparts flexibility of movement, and enables the gas sealing device to accommodate itself to small changes in the diameter of an electrode, and to axial displacement due to roof distortion.

Another type of economiser, based upon rather different principles of operation to the foregoing, is shown in Fig. 131. This apparatus is intended to check the flow of gas to a volume incapable of inducing air currents of harmful proportions. The sleeve, which is made from a refractory material of high thermal conductivity, is also designed to cool down the small quantity of gas to a temperature below that of its ignition. Furthermore, the portion of the electrode outside the furnace body, which is often inevitably above the temperature of combustion, is protected from the surrounding atmosphere by the close fitting sleeve, which extracts the heat rapidly by virtue of its high thermal conductivity and large surface of radiation. The use of a close fitting sleeve, which will retain an unbroken joint between the furnace roof and itself, irrespective of roof distortion, is made possible by mounting the sleeve upon a ring, with which it makes a ball and socket union. The seating ring rests upon the usual cooling box, or a plain ring of the same refractory material, and in either case is free to slide laterally in all directions. The sleeve will retain an unbroken joint at its seating, provided the inclination of the cooling box to its original position, normal to the axis of the electrode, does not exceed 15°; this angle allows for the worst possible degree of roof distortion. It is obvious that this type of economiser is more effectual when using graphite electrodes, which do not present the same difficulties as the amorphous variety, as previously mentioned. The height of the sleeve is such, however, that, even when considerable necking at a joint has taken place, there will always be a full diameter section of the electrode enclosed within it, so that the flow of gas is nevertheless restricted, although to a lesser extent.

APPENDIX.

METHODS OF BATH SAMPLING AND RAPID ANALYSIS.

THE great importance attached to the rapid working of electric furnaces, uninterrupted by delays due to mechanical or metallurgical causes, has been emphasised in those chapters relating to their technical and economic operation. It has also been shown that the process of steel-making is generally conducted in electric furnaces under careful chemical control, so that chemical analysis becomes of great importance.

It is seldom attempted to produce steel to a specified analysis by a single addition of ferro-alloys and carburising material to a bath of undetermined chemical composition. The general practice is to carburise in two stages and to determine the carbon, and sometimes also the manganese content of the bath, after the preliminary carbon addition and at a time when further variation of composition is unlikely. This method presents an accurate means of calculating the final additions necessary for bringing the analysis within the specification limits.

In the manufacture of alloy-steels, the raw materials often consist of scrap steel containing one or more of the alloying metals required in the final steel, and it is then essential to know the percentage of these constituents in the bath at the same time that a bath sample is taken for the analysis of carbon. This enables correct final additions of alloying metals to be made simultaneously with the final adjustment of the carbon content. It is unfortunate that practically every metallic constituent of any alloy steel is subject to variation until the chemical condition of both the bath of steel and slag are such that no further reaction involving oxidation or reduction of metals is possible. When such a stage has been reached, either in the basic or acid process, the steel would be normally nearly ready for tapping, provided the quantity of subsequent additions were known. For this reason it is obvious that the bath sampling, preparation of the sample, and the analysis of drillings should all be done with the least possible delay without sacrificing the care necessary both in sampling and analysis. The use of rapid, accurate methods of analysis, based as far as possible on volumetric principles, is therefore invaluable. In this Appendix, following a brief description of bath sampling and sample preparation, a selection of rapid and sufficiently reliable methods of analysis are given. All these methods have been widely used for controlling the analyses of electric steels, and in some cases represent the standard practice in most steel works laboratories.

Preparation of Sample.—A bath of steel is sampled by withdrawing a small quantity in a special sampling spoon and pouring into an iron mould.

The spoon (see Fig. 81) should be warmed and covered with a coating of slag before immersion into the steel, this being done to prevent steel freezing on to it. It is convenient to add a very small piece of aluminium to the mould or to the steel in the spoon before pouring, so that the sample may be sound and more suitable for drilling. When sampling a bath of steel after an addition of a carburiser, ferro-alloy, or an alloying metal, it is most important to ensure that the spoon sample is representative of a homogeneous bath, and for this reason a preliminary rabbling is given with a heavy slag-covered skimming tool or bar. The sample ingot may be either cylindrical or rectangular in shape, but should be capable of being firmly held under the drill. If the underside is convex, the sample should be flattened under a hammer while still at a forging temperature, and centre-punched later to facilitate drilling. Samples of steels subject to water or air hardening are either allowed to cool down to below redness in air before quenching in water, or may be more rapidly cooled by slowly quenching the side remote from the face to be drilled. The rapid cooling of water-hardening steels in this latter way must be done with caution, otherwise considerable delay may be caused by inability to drill the sample.

A heavy power driven drilling machine should always be available for use, and should be kept clean and free from oil for the special purpose of preparing samples. Surface drillings are always rejected to prevent contamination of the clean drillings with scale or dirt.

Methods of Analysis.

Phosphorus Determination.—The determination of phosphorus in a bath of steel is only necessary when the extent of phosphorus removal by an oxidising basic slag is uncertain; such cases may arise when melting cold charges of miscellaneous wrought-iron, cast-iron, and steel scrap containing high and irregular percentages of phosphorus. The sample is taken and analysed before removal of the dephosphorising slag, and an allowance should be made for a slight subsequent rise as shown on analysis of the pit sample. The observed difference in the phosphorus contents of samples taken before and after deoxidation of the steel bath cannot be accounted for by the possible reduction of any phosphorus bearing slag remaining in the furnace after skimming. If the phosphorus found in an oxidised bath sample is about ·010 per cent. the phosphorus content of the finished steel will generally be nearer ·02 per cent., even after the most perfect skimming possible. The following method of phosphorus determination is rapid and very widely used.

Principle of Method.—The method consists of a rapid precipitation of the phosphorus as ammonium phospho-molybdate, which, instead of being collected, dried, and weighed, is washed clean and dissolved in a solution of caustic soda. The exact weight of phosphorus in this precipitate is determined volumetrically, from which the percentage in the sample of steel can be accurately calculated. The reaction takes place according to the following equation:—

$$2[(NH_4)_3 12MoO_3PO_4] + 46NaOH + H_2O = 2[(NH_4)_2HPO_4] + (NH_4)_2MoO_4 + 23Na_2MoO_4 + 23H_2O.$$

Standard Solutions.—(*a*) *Ammonium Nitro-Molybdate Solution.*—Mix 150 grms. pure MoO_3 with 105 c.c. strong ammonia and 315 c.c. water. Stir and filter off any residue. Pour the filtrate with constant stirring into 1875 c.c. of nitric acid (1·2 sp. gr.).

(*b*) *Caustic Soda Solution*—To 100 grms. of pure NaOH add sufficient water to dissolve all but a few grms.; the undissolved residue, which will contain any sodium carbonate, is allowed to settle and the decanted liquid diluted to 2000 c.c.

The phosphorus equivalent to 1 c.c. of the caustic soda solution after making it equivalent to the nitric acid solution is determined by applying the method of analysis to a standard sample of known phosphorus content in exactly the same manner as described later for the ordinary samples.

(*c*) *Nitric Acid Solution.*—Make up a solution containing about 20 c.c. strong nitric acid (1·42 sp. gr.) diluted to 2000 c.c.

This or the caustic soda solution should then be diluted so that 1 c.c. of acid exactly neutralises 1 c.c. of soda, using an alcoholic solution of phenol-phthalein as an indicator; this being done as follows:—

Dilute 10 c.c. of the soda solution in a conical flask to 250 c.c. and add a few drops of the phenol-phthalein solution; run in the acid slowly from a burette until the pink colour just disappears. From the number of c.c. of acid required it is easy to calculate the quantity of water to add to either the acid or soda solution to make them exactly equivalent.

This preliminary equalising of the two solutions is not absolutely necessary, but simplifies the calculation for each determination of phosphorus.

(*d*) *Phenol-Phthalein Indicator.*—Dissolve ·1 grm. phenol-phthalein in 50 c.c. of alcohol.

Method of Operation.—Dissolve 2 grms. of drillings in 75 c.c. nitric acid (1·1 sp. gr.) in a conical flask and boil off all red fumes. Add a 3 per cent. solution of potassium permanganate to produce a permanent pink coloration or a slight brown precipitate on boiling for a few minutes. Remove the flask, add just sufficient H_2SO_4 to clear the precipitate, and then cool. Add 13 c.c. strong ammonia and 50 c.c. of molybdate solution, shake vigorously and allow the precipitate to settle for a few minutes in a warm place.

Filter and wash the precipitate with 2 per cent. nitric acid solution, then with a 2 per cent. KNO_3 solution until free of acid. The flask should also be washed clean with KNO_3, the washings being passed through the same filter.

Transfer the precipitate on the filter paper to the same flask and add a sufficient and known quantity of soda solution to dissolve it; dilute to about 50 c.c., add a few drops of phenolphthalein indicator and titrate with the nitric acid solution until the pink colour is discharged.

By deducting the number of c.c. of acid, or its equivalent of soda solution, from the amount of soda solution added to dissolve the precipitate, the number of c.c. of the soda solution actually consumed by the reaction with the phosphorus precipitate is known. Having determined the phosphorus equivalent of the soda solution by using a steel sample of known phosphorus content the per cent. of phosphorus in the steel can be directly calculated.

Manganese Determination.—The manganese remaining in a bath of steel after the oxidising slag reactions have ceased may vary considerably according to the character of the scrap used in the furnace charge, and may amount at times to a considerable part of the specification percentage. In such cases it is necessary to determine the manganese remaining in the bath at a time when there is no likelihood of either increase by the reduction of manganese oxide in the slag or removal under oxidising slag conditions.

Principle of Method.—The steel is dissolved and the solution strongly oxidised to destroy all carbonaceous residue.

The solution is then reduced to destroy all higher oxides of manganese and chromium, if any. The cooled solution is again strongly oxidised, filtered, and reduced by a measured excess of a standard solution of ferrous ammonium sulphate. The unoxidised excess of the latter is titrated with a standard permanganate solution, whose equivalent value in terms of the ferrous ammonium sulphate solution has been determined; the amount of permanganate solution equivalent to the ferrous ammonium sulphate oxidised by the steel solution can then be calculated. The value of 1 c.c. permanganate in terms of manganese is separately determined by applying the method to a sample of steel of known manganese content. Having obtained this figure, the amount of permanganate equivalent to the ferrous ammonium sulphate originally oxidised by the permanganate formed from the steel, multiplied by the determined manganese equivalent, gives the weight of manganese in the weighed sample, from which the percentage in the steel is calculated.

Chromium below 1·5 per cent. does not affect the accuracy of the method, provided that the second oxidation of the reduced solution is done when cold. Chromium then remains in the reduced state, and so has no influence on the ferrous solution added later.

Standard Solutions.—(a) Potassium Permanganate Solution.—Dissolve 3·16 grms. $KMnO_4$ and dilute to one litre, making a decinormal solution.

(*b*) *Ferrous Ammonium Sulphate Solution.*—Dissolve 39·6 grms. ferrous ammonium sulphate in 3 per cent. sulphuric acid and make up to 2000 c.c., using the same acid. This solution is also decinormal and 1 c.c. should exactly satisfy 1 c.c. of the permanganate.

The strength of the solution will slowly vary, so that it should be occasionally titrated against the permanganate, which remains unaltered, to determine the correct equivalent values. It simplifies the calculations if the two solutions are diluted until just equivalent, although this is not essential.

Method of Operation.—Dissolve 1·1 grm. in a conical flask in 35 c.c. of 1·2 sp. gr. nitric acid. Cool and dilute to about 80 c.c., and add sodium bismuthate until a permanganate colour persists. Reduce with ferrous ammonium sulphate and cool. The presence of chromium is indicated by a brown coloration which must be destroyed by the ferrous ammonium sulphate. To the cool solution add a slight excess of sodium bismuthate; usually about 1 grm. is sufficient. Filter through asbestos, using a filter pump, and wash with a 2 per cent. nitric acid solution until the filtrate is colourless. Add a measured excess of ferrous ammonium sulphate and titrate back with permanganate to a very faint coloration. Each c.c. of the permanganate used is equivalent to ·1 per cent. Mn in the steel, if the permanganate solution is decinormal.

Chromium Determination.—Alloy steels containing chromium are frequently made from scrap charges containing an appreciable percentage of chromium, which it may not be desirable to eliminate under a strongly oxidising slag. In such cases, when an addition of ferro-chrome has to be made to raise the chromium to the specification figure, it is necessary to take a bath sample for analysis at a time when slag reactions, either oxidising or reducing, are no longer capable of varying the chromium in the bath.

Solutions required.—(a) Decinormal Solution of Potassium Permanganate.—Dissolve 3·16 grms. of $KMnO_4$ and make up to 1000 c.c.

(*b*) *Decinormal Solution of Ferrous Ammonium Sulphate.*—Dissolve 39·6 grms. of ferrous ammonium sulphate in 3 per cent. sulphuric acid and make up to 2000 c.c., using the same acid. This solution is decinormal strength and should be equivalent to the $\frac{N}{10}$ permanganate solution.

(*c*) *Solvent Mixture for Steels.*—Make up a stock solution consisting of the following:—

300 c.c. HNO_3.
300 c.c. H_2SO_4 (one part in three).
300 c.c. H_2O.
$1\frac{1}{2}$ grms. manganous sulphate.

Method of Operation (N. M. Randall).—Dissolve 1 grm. of drillings in a conical flask in 25 c.c. of the solvent solution, avoiding evaporation. When solution is complete, including all light particles of metallic carbides, add 20 c.c. of H_2O and then 1 grm. of sodium bismuthate. Boil for three minutes and add just enough HCl—1 or 2 c.c. is usually sufficient —to clear the solution and destroy the permanganate coloration, after which boil again for a further two minutes.

Dilute with 200 c.c. of cold water and add a measured slight excess of ferrous ammonium sulphate. Titrate with permanganate, which gives the amount of ferrous ammonium sulphate added in excess of the quantity required to reduce the CrO_3. This amount, deducted from the total quantity added, gives the amount oxidised, which multiplied by ·001736 gives the weight of chromium in solution.

Nickel Determination.—Nickel steels are largely made from cold charges containing nickel steel scrap. In such cases the charge will generally contain rather less nickel than is required in the finished steel, so that it is necessary to know the percentage in the bath before a correct addition of nickel can be made to raise it to the specification figure. Nickel is not easily oxidised like chromium and manganese which pass into the slag, so that a bath sample taken whilst the slag conditions are still oxidising will give results sufficiently accurate.

Solutions Required.—(*a*) *Silver Nitrate.*—Dissolve 5·79 grms. recrystallised $AgNO_3$ in distilled water and dilute to 1000 c.c.; 1 c.c. of the solution is then equivalent to ·001 grm. of nickel.

(*b*) *Potassium Cyanide.*—Dissolve 5 grms. purest KCN and 5 grms. KOH in water and dilute to 1 litre.

Titrate this solution against the $AgNO_3$ solution and dilute or strengthen until just equivalent; this is not essential but it is convenient for calculation purposes.

(*c*) *Ammonium Sulphate.*—Dissolve 400 grms. $(NH_4)_2SO_4$ and make up to 1 litre.

(*d*) *Ammonia.*—Make up a 10 per cent. solution.

(*e*) *Potassium Iodide.*—Make up a 2 per cent. solution.

Method of Operation.—Weigh 1 grm. of drillings into a beaker and dissolve in 20 c.c. of 50 per cent. HCl.

When dissolved add 5 or 6 c.c. strong HNO_3. Boil to expel nitrous fumes and then wash into a 500 c.c. registered flask and dilute to about 120 c.c. with cold water.

Carefully neutralise with 10 per cent. ammonia, shaking vigorously between each addition; cool and add a measured quantity of KCN solution, using 10 c.c. for every 1 per cent. Ni expected in the sample plus 5 or 10 c.c. in excess. Shake and immediately add 50 c.c. of 10 per cent. ammonia, make up to 500 c.c. and pour on to a large rapid filtering paper.

Take 250 c.c. of the filtrate, add 6 c.c. Am_2SO_4 solution and 2 c.c. of KI solution. Titrate carefully with the standard solution of $AgNO_3$ until a very faint permanent opalescence persists. Some prefer to add a slight excess of $AgNO_3$ and then again just clear with the KCN solution.

The total quantity of $AgNO_3 \times 2$ represents the excess of KCN added to the original steel solution before making it up to 500 c.c. and halving its bulk.

The quantity of KCN consumed in the formation of $KNi(CN)_2$ is then obtained by subtraction, and the equivalent c.c. of $AgNO_3$ solution divided by 10 gives the percentage of Ni present.

The results by this method are sufficiently accurate for general purposes, but sometimes, in order to arrive at a more accurate result, the actual $AgNO_3$ used in titration is increased by one-tenth before calculating the percentage of Ni.

A nickel determination can be made in twenty minutes.

Tungsten Determination.—Tungsten behaves similarly to chromium so far as it is oxidised under an oxidising slag and reduced from its oxide or combined oxides dissolved in a highly reducing slag. Therefore, a bath sample must be taken at a time when the tungsten is no longer subject to variation, or, in other words, when the slag is free from tungstic oxides and has no power of oxidation.

Method of Operation.—Weigh out ·5 grm. of drillings or powder crushed in a percussion mortar and fuse in a large platinum crucible with 15 grms. of potassium bisulphate. Only one-third part of the bisulphate is at first used, as the action is violent. Keep the crucible well covered, and heat until fumes are evolved. Remove the flame for a minute or so, allowing the boiling action to subside. Allow to cool somewhat, add another third part of the bisulphate and gradually raise to redness. Repeat until all the bisulphate is fused. Usually fifteen minutes is sufficient for the complete fusion. Allow to cool, transfer the melt to a beaker and boil with water until dissolved, keeping the bulk down to 75 c.c. Add 20 c.c. strong HCl; boil until the WO_3 precipitated becomes pure yellow, and leave in a warm place for half-an-hour. Filter through a small filter, washing the precipitate with 10 per cent. ammonium nitrate. Dissolve the precipitate on the filter in hot dilute ammonia, allowing the solution to run into a weighed platinum dish and washing the filter with ammonium nitrate. Evaporate to dryness and heat until all the ammonium salts are decomposed. Treat the residue twice with HF to drive off any silica and weigh the WO_3. The filtrate from the first precipitate of WO_3 will contain a small amount of W in solution, and should be evaporated with HCl to effect precipitation. The WO_3 is then filtered and weighed up with the first precipitate or treated separately, as above.

The weight of WO_3 multiplied by ·793 = the weight of tungsten.

Carbon Determination.—*Bath Sampling.*—When it is required to know the carbon content of a bath of steel certain precautions must first be taken to obtain a truly representative sample. A bath of steel after carburising will not be perfectly homogeneous until it is moderately hot and has been mechanically agitated, which applies more particularly to medium and high carbon steels. The sample should only be taken when the bath is covered by well fused slag low in metallic oxides and incapable of exerting any oxidation of the carbon in the steel; a crude heat test should also be taken to make sure that the steel is not cold and this should be followed by a vigorous rabbling with a slag-covered skimming tool. Unless these precautions are taken the bath sample analysis will give vitiated results and the calculated addition of carbon subsequently made in the form of ferro-alloys or pig will probably produce a finished steel with a carbon content outside the specification limits.

In many works it is customary to use the colour method of carbon determination, which is quite satisfactory where no great degree of accuracy is required and when the carbon is moderately low. The colour method is not suitable for alloy steels or for medium and high carbon samples, owing to the comparison of the colour tints being vitiated either by the alloying metals or by the rate at which the sample is cooled. Whenever the colour method is adopted, the standard samples chosen should resemble as far as possible the bath samples, both as regards analysis and process of manufacture; it would, for example, be unwise to use a highly oxidised steel as a standard for comparison with a sample of "killed" steel.

The direct combustion method of analysis is almost invariably used for the production of alloy or plain carbon steel ingots to a close specification.

Principle of Method.—The sample of steel is completely burnt in a stream of oxygen, oxidation being assisted by covering the sample in the combustion boat with a small quantity of either lead chromate, red lead, or manganese tetroxide. These compounds may themselves contain traces of carbon which must be carefully determined beforehand and allowed for in every sample of steel analysed.

The carbon in the steel is burnt almost entirely to CO_2, any CO that may be formed being oxidised to CO_2 by red hot cupric oxide before leaving the combustion tube.

The CO_2 is slowly swept out of the tube by a stream of oxygen and air, and is purified before reaching the absorption tube. The CO_2 is best absorbed by soda lime for rapid working.

Preparation of Purifying Absorbents.—(a) Caustic Potash Solution.—This solution is made up to 50 per cent. strength and is used for removing any CO_2 from the oxygen or air before entering the combustion tube.

(*b*) *Cupric Oxide Gauze.*—A long strip of fine meshed copper gauze about 3 or 4 inches wide is rolled up tightly to form a plug, which can be pushed into the combustion tube. This plug occupies a position towards the outlet end of the tube, where it will be raised to a bright red heat. The copper gauze is later converted into oxide by heating to a high temperature in the tube through which a stream of air or oxygen is passed.

(*c*) *Chromic Acid Solution.*—A saturated solution of chromium trioxide in water is generally used but may be substituted by a saturated solution of chromium trioxide in dilute H_2SO_4.

(*d*) *Pure Sulphuric Acid.*

(*e*) *Anhydrous Fused Calcium Chloride.*—The prepared chloride is crushed to the size of small peas, freed from dust, and dried at 200° C. before filling into the U tubes.

(*f*) *Soda Lime.*—Soda lime may be prepared by dissolving CaO and NaOH in their molecular proportions in water, and evaporating the solution to dryness. The dried residue is broken up and crushed to pass a 10-mesh sieve and is slightly damped with water, containing a few drops of a 5 per cent. phenol-phthalein solution in alcohol, before packing into the absorption tube.

Apparatus and Method of Determination.—The apparatus, consisting of gas washing bottles, electric combustion furnace, SO_2 absorbing bulbs, drying bulbs, CO_2 absorption U tube and aspirator, is shown assembled in Fig. 132.

The two gas washing bottles A and B, shown to the right-hand side of the apparatus, are filled with strong sulphuric acid and with KOH solution respectively, the latter being nearest the oxygen cylinder. Various types of washing bottles or absorption bulbs may be used in place of the Wolff's bottles shown. The H_2SO_4 bottle is attached by a short length of pres-

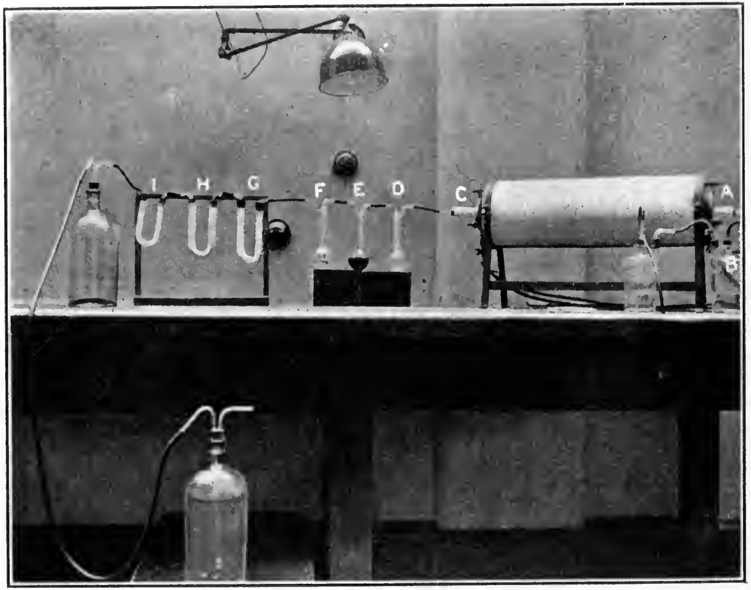

Fig. 132.

[*To face p.* 340.

sure tubing to a rubber cork, which fits tightly into the combustion tube C.

The combustion tube may be either porcelain or fused silica about 1 inch internal diameter or large enough to take the combustion boat, which is usually fire-clay or porcelain. Fire-clay boats should be strongly ignited in a muffle before use to ensure perfect freedom from carbon. The outlet end of the combustion tube is connected to an empty bulb D which removes the bulk of the moisture carried over by simple condensation. The next bulb E contains the chromic acid solution which oxidises any SO_2 passing over with the gas to SO_3, which is then absorbed. The bulb F contains strong H_2SO_4 which further dries the gas before it passes through the U tube G containing anhydrous calcium chloride which abstracts the last traces of water vapour. The U tube H is the weighed absorption tube, and is packed with soda lime in the right-hand limb and with calcium chloride in the other. The calcium chloride in this U tube absorbs any moisture which might be driven off from the soda lime. Lastly, a U tube I filled with calcium chloride completes the combustion train, and prevents the weighed absorption U tube H from absorbing any moisture from the atmosphere. The Winchester quart bottles shown serve as a convenient form of aspirator and are so arranged that water syphons over from the upper bottle into the lower one, drawing either oxygen or air through the apparatus.

Before using a newly charged weighing tube for an actual carbon determination it should be connected up in the train through which air is then passed, the furnace being at a full heat to resemble actual working conditions. After passing air for about fifteen minutes, the weighing tube is disconnected and carefully weighed; it should then be again connected in the train, and the process repeated until its weight is constant. The apparatus is then ready for use.

Two grms. of fine drillings or powder (in the case of self-hardening steels) are mixed with 2 grms. of red lead and placed in the combustion boat, which is then quickly pushed into the hottest part of the combustion tube; the latter should be at a temperature of at least 800° C. The washing bottles A and B are then connected to the oxygen cylinder or a gas container and to the combustion tube. The pressure in the train is then reduced by coupling up the aspirator so that the water endeavours to syphon over from the upper bottle. In about three minutes the oxygen is turned on and a stream of gas passed through the apparatus so that the Winchester bottle is emptied in about five minutes.

The oxygen is then shut off, the aspirator bottles reversed, and a stream of air drawn through until the upper bottle is again emptied. The weighing U tube H is then disconnected and weighed, the increase of weight being due to the CO_2 absorbed. The phenol-phthalein loses its pink colour when the soda lime becomes exhausted, and requires renewal.

A carbon determination by this method should be completed within twenty minutes of receiving the prepared sample.

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