Proceedings

at the

Ordinary General Meeting,

27th May, 1936.

The Ordinary General Meeting of the Society was held in the Assembly Hall, Kelvin House, 100, Fox Street, Johannesburg, at 8-0 o'clock p.m. Mr. R. A. H. Flugge-de-Smidt (*President*) was in the Chair.

Forty-two Members : Messrs. T. Coulter, T. Nimmo Dewar, J. M. Dixon, Dr. P. E. Hall, Professor C. W. Biccard Jeppe, T. C. A. Meyer, T. K. Prentice, E. C. Ranson, Professor G. H. Stanley, Professor F. J. Tromp (*Members of Council*), W. N. Amos, J. L. Atkinson, J. H. Bath, H. M. van M. Broekman, F. Brown, H. Clarke, J. G. Finlay, T. L. Gibbs, J. Gibson, C. J. Gray, A. J. Hepburn, P. F. Hirschfelder, J. J. Hupkes, D. M. Jamieson, W. E. John, R. J. B. Kethel, K. Leinberger, Dr. J. T. McIntyre, J. C. Moore, Dr. L. Morgenroth, I. L. Myburgh, O. L. Ochse, A. J. L. Pretorius, F. H. Rogers, C. E. Silverthorne, H. Simon, C. W. van der Spuy, Dr. J. C. Vogel, E. C. Whittaker, J. R. Williams, S. E. Willows-Munro, J. L. P. Wyndham.

Three Associates : Messrs. G. Cumming, T. Waterman, B. H. Ziervogel.

Six Students : Messrs. H. M. Carliell, J. Levin, H. H. McGregor, E. W. Page, I. A. Taylor, E. B. S. Warren.

Fifteen Visitors, including Messrs. J. Buist and R. P. Forsyth. and H. A. G. Jeffreys (*Secretary*), making a total attendance of sixty-eight.

MINUTES.

The Minutes of the Ordinary General Meeting held on Wednesday, the 25th March, 1936, as published in the *Journal* for that month, were confirmed.

NEW MEMBERS.

A Ballot was held for the election of the following new members :—

AHBECK, J. S., Mines Department, P.O. Box 164, Germiston, Transvaal ; Inspector of Mines.

BERINGER, B., Randfontein Estates Gold Mining Co., Ltd., P.O. Box 2, Randfontein, Transvaal ; Mine Overseer.

HILL, F. G., New Modderfontein Gold Mining Co., Ltd., P.O. Van Ryn, Transvaal ; Mining Engineer.

IRWIN, R. W., Gold Fields Laboratories, P.O. Box 6176, Johannesburg ; Chemist. (Transfer from Associates' Roll.)

KIRKMAN, S. F., P.O. Box 206, Heidelberg, Transvaal ; Mining.

KRAMERS, W., Springs Mines, Ltd., P.O. Box 54, Springs, Transvaal ; Reduction Worker.

LAMB. C. L., Brakpan Mines, Ltd., P.O. Box 3, Brakpan, Transvaal ; Mine Manager. (Transfer from Associates' Roll.)

LEGGE, A. J. W., Inspector of Mines Office, Mbarara, Uganda, British East Africa ; Inspector of Mines.

LEINBERGER, J. F., New State Areas, Ltd., P.O. Box 41, Springs, Transvaal ; Chief Surveyor.

MacKENZIE, W. A., New Consolidated Gold Fields, Ltd., P.O. Box 1167, Johannesburg ; Company Manager.

MALHERBE, D. G., Mines Department, P.O. Box 183, Brakpan, Transvaal ; Inspector of Mines.

MARKS, T., Lewis & Marks, Ltd., P.O. Box 1030, Johannesburg ; Director of Companies.

MORTLEMAN, J. W. V., East Geduld Mines, Ltd., P.O. Box 222, Springs, Transvaal ; Mining Engineer.

PITT, L. T. CAMPBELL, New Consolidated Gold Fields, Ltd., P.O. Box 1167, Johannesburg ; Mechanical Engineer.

RICHARDS, M. W., Union Corporation, Ltd., P.O. Box 1156, Johannesburg ; Administration, Gold Mining Companies.

ROBB, T. L., Consolidated Murchison Mine, P.O. Gravelotte, Northern Transvaal ; Mining.

ROUX, P. W. T., New State Areas, Ltd., P.O. Box 41, Springs, Transvaal ; Mine Captain.

SINTON. J. S., Haggie, Son & Love (1936), Ltd., P.O. Box 72. Cleveland. Johannesburg ; General Manager.

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Mine Ambulance Shield Competitions.

SOAR, V. D., National Mining Corporation, Ltd., P.O. Mbeya, Tanganyika Territory ; Mining Engineer.

STEWART, G. C. R., Bushtick Mine, Private Bag 118 M, Bulawayo, Southern Rhodesia ; Mining Engineer.

VAN ZIJL, J. S., "Beverley," Hospital Road, Boksburg, Transvaal ; Assistant Inspector of Mines.

WALKER, PROF. W. J., Mechanical Engineering Department, University of the Witwatersrand, P.O. Box 1176, Johannesburg ; Professor of Mechanical Engineering.

WOOD, K. B., Nourse Mines. Ltd., P.O. Box 32, Denver, Johannesburg ; Mining Engineer.

The Secretary announced that the Council had admitted the following gentlemen as Associate Members :—

BELL, IAN D. W., 16, Westcliffe Flats, Parktown, Johannesburg ; Assistant Inspector of Mines.

FLOOD, E. J., Geduld Proprietary Mines, Ltd., P.O. Dersley, Transvaal ; Shift Boss.

GRAY, K. M., Van Dyk Consolidated Mines, Ltd., P.O. Box 222, Boksburg, Transvaal ; Shift Boss.

HARMER, H. R. F. W., Geduld Proprietary Mines, Ltd., P.O. Dersley, Transvaal ; Mining Engineer.

JURY, R. V., Ashanti Goldfields Corporation, Ltd., Obuasi, Gold Coast Colony, West Africa ; Chief Assayer.

LOEWENSTEIN, P. R., Geduld Proprietary Mines, Ltd., P.O. Dersley, Transvaal ; Metallurgist.

PHILLIPS, G. R., Langlaagte Estate & Gold Mining Co., Ltd., P.O. Box 98, Langlaagte, Transvaal ; Mine Surveyor.

RICHARDS, H., Ashanti Goldfields Corporation, Ltd., Obuasi, Gold Coast Colony, West Africa ; Metallurgy.

THACKWELL, G. J. PENROSE, 35, Belmont Court, Banket Street, Johannesburg ; Sales Engineer.

WYKES, E. R., East Geduld Mines, Ltd., P.O. Box 222, Springs, Transvaal ; Mining Engineer.

and as Student Members :—

KLOPPER, B. D. N., Robinson Station, Transvaal.

MELTZER, J., 606, Mackay Buildings, Rissik Street, Johannesburg.

SCRUTINEERS FOR ANNUAL BALLOT.

The President pointed out that the Meeting had now to appoint Scrutineers for the Annual Ballot, and with the permission of the Meeting he would like to nominate their very old friends : Messrs. G. Melville, S. Newton and F. W. Watson, who had served in that capacity for many years past, and, with the permission of the Meeting, he would like to add the name of Mr. E. C. Ransom, the immediate Past-President.—Agreed.

MINE AMBULANCE SHIELD COMPETITIONS.

The President remarked that on the 10th May he had had the honour of representing the Society in presenting the shields that were presented by the Society some twenty-three years ago. The Society presented two shields—one for underground workers, and one for surface workers. The West Rand Consolidated had won the underground shield, and the Nourse Mines the surface shield. A tremendous number had entered for the competitions, and it had been suggested, as members had probably seen in the Press, that, in future, competitions would be on an entirely different basis. It had been felt that however good the work had been in trying to get men to take up first-aid on the Mines, possibly owing to the fact that the standard was too high, one had not been able to rope in a very large percentage of those who were not compelled to take the first-aid examination.

The idea he believed now was, in future, to substitute a more elementary examination and to try and induce every worker on every mine to take the examination.

LECTURE BY PROFESSOR C. W. B. JEPPE.

The President said he had a very pleasing announcement to make. Members had all been looking forward with very great interest to the inaugural lecture by Professor C. W. Biccard Jeppe, who would lecture on "Some Tendencies in Modern Mining Practice," in the Chemistry Lecture Theatre of the University on the 3rd June, at 5-15 p.m. All members of the Society had been cordially invited to attend, and he hoped the Society would have a very big turn-out to show its appreciation of Mr. Jeppe's work.

He would now call upon Mr. Prentice to give members a short account of the—

EXCURSION TO THE WORKS OF THE NATIONAL MAIZE PRODUCTS, LIMITED.

which he was unfortunately unable to attend.

Mr. T. K. Prentice (*Hon. Treasurer*) said that, by the courtesy of the Directors and Management of the National Maize Products, Limited, members of the Society were privileged to visit their works at Germiston last Saturday, and, notwithstanding the very cold and wet day, there was a very satisfactory turn-out.

The works had not yet started producing, but were very near the producing stage. The visitors were conducted through the works by the Technical Manager, Dr. Viljoen, a South African who had recently returned

from America, and he gave them a very clear and lucid description of the process, which briefly consisted of cleaning the maize, separating the germ, which was sold to manufacturers for the production of edible oil, converting the starchy portion of the mealie into sugar, then into alcohol and carbon dioxide. There was a solid residue left, which was converted into cattle feed. The alcohol, they were told, could be mixed with petrol up to 25 per cent., which gave a mixture for use in internal combustion engines equal to that of petrol itself ; and that, of course, if confirmed, was a matter of very great importance to South Africa.

The carbon dioxide was solidified, forming what is known as dry ice, which is particularly suitable for refrigerating purposes, and it was hoped by the Company that the South African Railways would make use of it. In many ways it was much better than water ice. It was dry ; it changed directly from the solid form into gas, and the gas given off was beneficial in suppressing bacteria, and, of course, the solid carbon dioxide was very much colder than water ice.

The visit proved extremely interesting and Dr. Viljoen extended an invitation to all to come again when production had started.

Mr. Prentice concluded by saying that he was sure the Society wished this new South African industry great prosperity in the future.

The President then called on Mr. R. P. Forsyth to give his paper entitled, "Study of the Direct Production of Chrome Steel."

A STUDY OF A DIRECT METHOD OF STAINLESS STEEL PRODUCTION.

By R. P. FORSYTH.

(*Minerals Research Scholar*)

(*Research Grant Board*)

Object of Investigation.

The object of this investigation was to determine the possibility of producing stainless steel of low carbon content, the required chromium content of the steel being derived from the reduction of chromite by ferrosilicon above the prepared bath of low carbon steel.

Characteristics of Stainless Steels.

In order to understand the difficulties involved in the production of stainless steel it is well to note briefly the metallographic and physical characteristics of this type of material.

The fundamental constituent of most stainless steels is chromium, and it is the effect of this element on the properties of the metal that is important. The metallographic structure of stainless steel resembles generally that of plain carbon steel. That is, the constituent "pearlite" in the fully annealed material and the "austenitic", "matensitic" and "troostitic" structures in the heat treated specimens are very similar in both steels. The variation lies in the widely different carbon content of the constituent "pearlite" for these two steels. With increasing percentages of chromium up to 12%, the carbon content in this constituent drops from 0·9% for plain carbon to a little over 0·3% for 12% chromium steel. A fully hardened 0·3% stainless steel is softer than the corresponding 0·9% plain carbon steel, but the 0·3% stainless is far harder than a 0·3% plain carbon steel. From a micro-structural point of view, annealed stainless material containing 0·5% carbon contains a large proportion of free carbide and corresponds in that respect to a tool steel (plain carbon) containing 1·3 to 1·5% carbon.

Another effect of chromium is that it necessitates a slower rate of cooling to produce a completely annealed structure. As an instance of this plain carbon steel may be cooled down from its austenitic condition in air, and will show a pearlitic structure, while high chromium steel treated in the same way will be martensitic. Due to this intense air-hardening effect, great care is

taken in all operations after teeming an ingot to minimise the serious cracks which form as a result of too rapid or uneven cooling.

It is evident from the foregoing that, if stainless steel is to be processed in the same way as plain carbon steel, it is essential that it should contain not much greater than 0·3% carbon. For ordinary steel practice it is a simple matter to produce such low carbon material. In the presence of chromium in proportions as required in stainless steels, the production of low carbon material is complicated by the peculiar behaviour of chromium under ordinary furnace conditions.
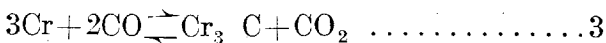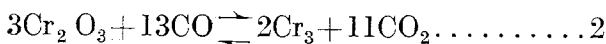
The most notable peculiarity is the affinity which chromium has for carbon. Not only does chromium rapidly absorb carbon when in contact with it at high temperatures, but it also forms carbides similarly to iron and iron oxides by contact with carbonaceous gases. No accurate data are available as to gaseous compositions which effect this carburetion : but from the fact that in practice it has been frequently noted that carburetion can take place simultaneously with oxidation of chromium and that chromium oxide can only be reduced to metal by carbon and then only at high temperatures the following deductions may be made.

(a) The dissociation of $2Cr_2O_3 \rightleftharpoons 4Cr + 3O_2$ on heating only takes place to a very small extent as compared with FeO dissociation. As a result the gaseous phase in equilibrium with the system,

$$Cr_2O_3 + 3C \rightleftharpoons 2Cr + 3CO \quad \dots \dots \dots \dots \dots 1$$

is very much higher in CO content than for similar FeO reduction by carbon

(b) The reactions,

$$3Cr_2O_3 + 13CO \rightleftharpoons 2Cr_3 + 11CO_2 \dots \dots \dots 2$$

$$3Cr + 2CO \rightleftharpoons Cr_3 C + CO_2 \quad \dots \dots \dots \dots \dots 3$$

can proceed from left to right when the concentration of CO in the gas phase is below that in equilibrium with equation 1.

(c) In consequence of (a) and (b) carburised metal can only be decarburised with minimum oxidation of metal when the oxidising agent has an oxygen concentration equal to that for chromium oxide. This is in accordance with practice, since it has been found[1] that high chromium steel is more efficiently decarburised using a high Cr_2O_3 slag than by using a high FeO slag. Under a black slag

chromium oxidises more rapidly than iron or carbon. Silicon, however, is oxidised in preference to chromium or carbon. As an eliminator of silicon from a bath of steel then, a Cr_2O_3 slag will be far more sluggish in action than a black slag.

Having discussed the peculiar characteristics of high chromium steels it is not difficult to understand why such a high price is demanded for steels. This is due to a number of causes, all of which turn on the affinity of high chromium material for carbon and the intensive air hardening affect. In order to produce a low carbon steel, which is high in chromium, then if ferrochromium is to be used for the introduction of this chromium, it must of necessity be low in carbon. Ferrochromium produced from chromite is not expensive, but as produced in one operation from an electric furnace contains 4 to 8% of carbon. In order to lower the carbon, the high carbon ferrochromium goes through a highly specialised and complicated decarburising process. Alternatively, the Goldschmidt product may be used. For the production of stainless irons and steels of less than 0·3% carbon, ferrochrome containing 70% chromium and 0·60% C to 1·0% C may be used. This is naturally a very expensive material.

A second cause of the high cost of chromium steels is the low value of this type of scrap material. This is due to its rapid carbon absorption during remelting which makes the reclamation of scrap a specialised process usually involving the use of expensive induction furnaces. Assuming that a 20% discard is taken from the bottom and top of the ingot the cost of the remaining 80% jumps approximately one fifth, due to the value of the scrap dropping to a practically negligible figure.

The third cause is the great care necessary in all heat treatment ; resulting in extra time and fuel consumption in all heating operations, reduced capacity of rolling m lls and rigid inspection and grinding operations of the surface, to eliminate cracks and flaws in the ingot surface.

Methods of Stainless Steel Production.

There are two main methods of stainless steel production. The first is the present commercial tonnage method which employs as raw materials scrap steel, ferrochrome, and the usual fluxing materials, while the second utilises the direct reduction of chromium

from chromite. The first is carried out in a basic furnace usually of the Heroult type and of 6 to 10 tons capacity[2], as follows :—

Scrap steel of low carbon content is melted down in an electric furnace with the usual black slag of lime, millscale, ore, and fluorspar. The phosphorus and carbon are eliminated in the ordinary way. The carbon is brought down to as low a figure as 0·04%, this being done in order that the cheaper grades of ferrochrome of high carbon content may be used later on. The power is taken off, the slag removed and the metal bath cleared as thoroughly as possible from all slag, since subsequent operations will be carried out under reducing conditions, to prevent loss of chromium and give a sound product.

Lime is then added together with fluorspar to give a fluid slag high in lime. The slags employed at this stage are of the disintegrating calcium silicate or "falling" type. These slags contain calcium orthosilicate ($2CaO. SiO_2$, *i.e.*, 65% Ca) in large proportion, which undergoes a large volume change at about 650°C due to a change in crystalline form. It is not obvious why this type of slag is used, as its functions are dissimilar to those of a "carbide" slag in many respects. This will be discussed later under "Suitable Basic Slags."

In the early days carbon was added to the slag so built up, with the object of obtaining a white slag as for usual steel practice, but as this results invariably in carbon "pick-up", and necessitates the subsequent employment of low carbon ferrochrome, nowadays silicon in the form of 70 to 90% ferrosilicon is substituted. By reason of its low specific gravity the ferrosilicon is held in suspension, and reduces any oxides present in the slag, and incidentally in the metal at the slag-metal interface ; possibly Si forms calcium silicide $CaSi_2$ in these high CaO slags. The deoxidising action of this silicide will be similar to that of CaC_2. In this way a deoxidised bath of low carbon steel is obtained.

The temperature is raised considerably and the large quantity of ferrochrome added. After the usual procedure involving analysis,

temperature adjustment, alloying additions and final deoxidation, the metal is poured into the ladle for subsequent teeming.

Undoubtedly the extension of the field of application of stainless steels is limited seriously by the high cost of this material. Even so, stainless iron is competing with and replacing not only ordinary types of steel, but also a large number of non-ferrous metals and alloys. A decrease in the price of stainless steel would extend further its application, and the reason for the evolution of direct reduction processes is the possibility of lowering production costs. It appears, however, that the process is more complicated and requires greater attention than indirect methods, and the product is no better metallurgically.

There are two important direct processes employing chromite and ferrosilicon as raw materials in place of ferrochrome. There are the "Hamilton-Evans"[*] and "Wild"[†] processes, both of which have been patented. From the limited bulk and sketchy nature of the published descriptions which have appeared in periodicals and technical papers, it is evident that little is generally known about these processes. The available matter will be discussed later.

The "Hamilton-Evans" process[3] seems to have had the greater measure of success. Scrap steel is melted down with a black slag in a basic electric furnace of the Heroult type, and carbon reduced to the neighbourhood of 0·04%. The slag is removed. A so-called "receptor" slag is added but the composition of this is not definitely stated. When in good shape the temperature is raised in preparation for the addition of the chromite ferrosilicon mixture. Chromium is reduced and this finds its way through the slag into the metal. It is stated that the resulting slag is used to refine the metal previous to tapping. The chromium loss is estimated at 25% of that charged.

An efficiency of 60 to 70% of the silicon charged for reducing purposes is found. The following are analyses of steel produced by this process.

Carbon.	Silicon.	Chromium.	Manganese.	Nickel.
0·09	0·04	12·0	0·30	0·23
0·085	0·34	11·5	0·32	0·26
0·095	0·31	11·8	0·28	0·25
0·10	0·33	12·1	0·27	0·26
0·09	0·47	11·7	0·30	0·23
0·085	0·47	12·6	0·32	0·22

*British Patents No. 174,271 ; 198,323 ; 211,210; 213,737 ; 262,206.

†British Patents No. 198,423, June 21, 1922.

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The "Wild" process[4], which may be regarded as a modification of the above, differs in that it is claimed that the silicon content of the metal can be maintained below 0·5% by suitable alternate additions of chromite and ferrosilicon. The covering slag is always maintained high in Cr_2O_3. Robiette states that the ferrocilicon is added to the bath and removed with resulting reduction of chromite from the slag. However, this must of necessity take a long time, owing to the reaction surface being limited by the slag metal interface, the low concentration of silicon reaction at any instant, and the dependence upon diffusion.

The following are ranges of steels produced by the "Wild" process.

Name.	Chromium.	Carbon.	Silicon.	Nickel.
Defirust	12—14	less than ·1	less than ·75	trace
Special defirust	16—18	0·1 max.	·75 max.	trace
Defistain	17—19	·06—·18	·75 max.	7—9
Defiheat	25—30	·20—·25	·75	trace

A third process which may be mentioned is that developed by A. L. Field[5] in which 4 to 6% ferrochrome is added to the bath of steel and elimination of carbon carried out by oxidation with a black slag. The oxidised chromium, which amounts to half that charged, is subsequently reduced by additions of ferrosilicon. It is claimed that the total Cr loss is only 10% of that charged.

The success of the "Hamilton-Evans" process evidently depends on the receptor slag composition. A pointer to the probable composition may be obtained from published data of German practice for the working of a charge in the Basic Open Hearth, by reduction of chrome ore.[6] The analysis of initial and final slags is given. The charge, slag analyses and metal analyses are as follows :

It will be noticed that the final slag is low in oxygen content. It will be argued that Cr_2O_3 is rather high. However, owing to the low O_2 pressure of Cr_2O_3 in comparison with that of FeO, this probably has as much effect on the oxygen content of the metal as 1·5% FeO. Notice also that the silicon content of the metal is rather high, the usual content being less than 0·75% in steel of this type.

Discussion on "Receptor" Slag.

The reaction upon which the process is based is the following :—

$$FeO.Cr_2O_3 + 2Si = Fe.Cr_2 + 2SiO_2 + X cals..4$$
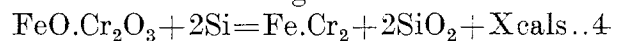
It has been stated that by the reduction of chromite in this way, not more than 13% Cr can be obtained in the metal.[7] This is surprising and if it is so, can only be explained on the following deduced assumptions. Consider the reaction between pure materials represented by the equation.

$$2Cr_2O_3 + 3Si \rightleftharpoons 4Cr + 3SiO_2 + X cals.$$
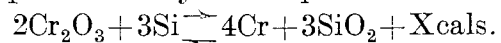
In this heterogenous equilibrium the law of mass action may be applied by considering the vapour pressure of each constituent :—

Let　p1—vapour pressure of Cr_2O_3 at T°C.
　　　p2—vapour pressure of Si at T°C.
　　　p3—vapour pressure of Cr at T°C.
　　　p4—vapour pressure of SiO_2 at T°C.

Then $\dfrac{p1^2 \times p2^3}{p3^4 \times p4^3} = K$.

Assuming that the constituents are present in measurable quantity, p1, p2, p3, p4 will be a constant depending only on temperature.

CHARGE IN POUNDS.

| Hematite.	Scrap.	Lime.	Ore.	Fluorspar.	Chromite and Ferrosilicon.
7700	18700	10230	880	1540	28050

SLAG ANALYSES.

Metal.	CaO.	MgO.	SiO_2.	FeO.	Fe_2O_3	Cr_2O_3	Al_2O_3	MnO
Initial	47·7	0·4	12·5	15·9	4·3	1·7	3·5	1·6
Final	32·6	13·0	34·8	1·5	0·6	6·5	2·9	0·6

METAL ANALYSES.

Metal.	C.	Cr.	Si.	Mn.	P.	S.	Ni.
Initial	·8	—	·01	·06	—	—	—
Final	·16	13·1	1·5	·30	·019	·017	·24

Consider now the reactions as they would take place in practice within a slag which is initially highly basic and finally far less basic. Since no interaction of Si and Cr would take place in such a slag with the components of the slag it is reasonable to assume that p2 and p3 would remain unaltered. However, since Cr_2O_3 and SiO_2 are present as compounds of the slag constituents it is possible that p1, p4 will be affected considerably since their values will no longer depend on the vapour pressure of the pure oxides, but will depend on the dissociation constant of their $Cr_2O_3SiO_2$ compounds.

Assume now that the Cr_2O_3 content of the slag remains constant and that SiO_2 increases from a small value to a relatively high one.

$$K = K1\frac{p1^2}{p4^3}$$

Now if the ratio :—

$$\frac{\text{Dissociation of } Cr_2O_3 \text{ compounds of highly basic slags}}{\text{Dissociation of } SiO_2 \text{ compounds of highly basic slags}}$$

is greater than the ratio :—

$$\frac{\text{Dissociation of } Cr_2O_3 \text{ compounds of less basic slags}}{\text{Dissociation of } SiO_2 \text{ compounds of less basic slags}}$$

then the value of $\frac{p1^2}{p4^3}$ will decrease as the SiO_2 in the slag increases. K will as a consequence (at constant temperature) decrease, i.e., the extent of the reaction from left to right will decrease. This is most probably what does actually happen. (Large losses of Cr to slag when melting down scrap stainless material in an acid furnace, with small losses in a basic furnace.) Other factors such as decrease of reaction velocity due to the above decrease in the value of $\frac{p1^2}{p4^3}$ possibly also have a considerable bearing on the efficient reduction of Cr_2O_3 and the amount of Si finding its way into the metal.

It is essential then to determine the minimum Cr_2O_3, CaO and SiO_2 content of the final slag in actual practice in order to guard against too low a content of Cr and too high a value for Si in the finished product : For example in the German practice shown, Cr_2O_3 in the slag will have to be maintained at 6·5% at least if silica lime and magnesia are as shown in the final slag, and if no chromium loss from the reducing mixture is to be suffered.

For these reasons the slag should be maintained as basic as possible. Chromite contains approximately 30% of "gangue." The bulk of chromite and ferrosilicon together will amount to 50% of the weight of metal in the bath, and has to be heated up to at least 1600° C. It is evident that it is undesirable to attempt to maintain the slag basicity during the reduction period by additions of lime as well. Now the final slag should be capable of acting as a refining slag for the metal, for if a refining slag has to be built up at the completion of the reaction, the process will be excessively prolonged. In order to satisfy these conditions the "receptor" slag will have to be abnormally high in bases.

It will be noticed in the German practice that the initial slag has been made high in FeO and very low in Cr_2O_3. This was presumably done with the idea of

(a) supplying an oxidising environment for any silicon, which would otherwise have entered the metal,

(b) imparting fluidity to the basic slag. However, it will be shown that the reasoning behind both (a) and (b) is erroneous. FeO will be reduced by silicon in preference to chromium. The result is that, while this is proceeding, the unreduced Cr_2O_3 will build up in the slag and the slag will become viscous as a result. Besides this an expensive reducing agent is being used to reduce iron oxides. As pointed out above, the correct slag to use is a Cr_2O_3 slag, containing as little FeO as possible. In this way, must Si be prevented from entering the metal. Furthermore, if the slag is initially of correct viscosity, the viscosity will not at any stage be lowered due to increase of Cr_2O_3.

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If an abnormally basic slag, containing Cr_2O_3 is to be maintained initially in a fluid condition, then according to standard practice a large quantity of fluorspar will have to be added. The disadvantages of this are the following.

(a) Fluorspar corrodes the siliceous side walls and furnace roof by volatilisation and by this means the SiO_2 content of the slag invariably increases. Besides this, CaF_2 has a corrosive action on the basic lining at the slag line. In the direct process this will be intensified owing to the extended treatment time.

(b) Although the initial slag with fluorspar may be of correct fluidity, the fluidity will increase as the slag becomes less basic due to silica introduced by the reacting ferrosilicon, and by conditions in (a). The resulting thin slag is totally unsuited for the process, reflecting as it does the heat of the arc on to the furnace roof, and interfering with efficient heat transference. Further, a thin slag allows greater carbon "pick up" by the metal from the electrodes than does a slag of correct body.[8]

Importance of Correct Temperature Control.

Equation 4 represents an exothermic reaction. Therefore, according to Le Chatelier's principle, the extent of the reaction from left to right will be greater at low temperatures than at high temperatures. Theoretically then, the receptor slag should be kept as near to the melting point of the metal bath as possible. In practice this temperature will have to be not less than 1600° C. otherwise serious risk of chilling the bath will result.

Suitable Fluid Basic Slags.

It is frequently stated that magnesia in finishing slags is to be avoided, since this material causes a "pasty[9] heavy slag which is difficult to clear" of dissolved oxides. While this may apply to finishing slags for alloy steels where calcium carbide is the deoxidising agent, it is difficult to understand how it applies to stainless steel manufacture where calcium carbide is not a constituent of the slag. Here carbon is replaced by silicon. Calcium then is not an essential constituent of the deoxidiser, and the replacement of CaO by MgO suggests itself. The conversion costs would certainly be lowered. In the experimental work to be described it was actually found that MgO

produces fluidity in a basic slag. Further, it was found that all operations of slag formation could be carried out without the use of fluorspar and that the viscosity of the slag remained very much the same throughout. This departure from usual practice seems to be of considerable importance and should facilitate the working of a direct process. It has, moreover, the support of authoritative research work carried out by McCaffery and Oesterle on blast furnace slags.[10] Extracts from their papers will be given. The range of slags reviewed by them is

SiO_2 ..	..	..	25 to 55%
Al_2O_3	..	..	0 to 35%
Lime ..	..	..	25 to 55%
Magnesia	..	..	0 to 20%

"When reviewing the literature it is astonishing to observe the diversity of opinion that has been expressed as to the effect of magnesia in slags...... The results of our work have cleared up some of these differences."

"Perhaps the most noticeable thing in our work is the effect of magnesia *in reducing the viscosity of slags. As the magnesia content increases the viscosity decreases at a fairly rapid rate. When slags have a magnesia content of from* 15 *to* 20% *there are no great differences in viscosity at any one temperature with considerable changes in the silica, lime, alumina, components. Moreover, in this same range of components* (15 *to* 20% *MgO) the actual viscosity is only* $\frac{1}{6}$*th or* $\frac{1}{5}$*th the viscosity if no magnesia or only a small percentage of magnesia is present.*"

"*Quite definitely our work shows that magnesia has much more influence in reducing slag viscosity than has lime.* This statement is true in all the ranges of composition in which slags have been made in the past or are likely to be made in the future. There has been and still is considerable discussion of the relative merits of lime and magnesia as flux. The viscosity is certainly part of what is called "fluxing quality" and our work definitely shows that magnesia is quite different from lime in its effect on this property."

The curves (1) shown have been taken from the paper referred to and illustrate the above extracts. Take for instance, slags containing 50% CaO. It will be noticed here that for low viscosities where MgO is

small, SiO_2 must be large, whereas with higher proportions of MgO, SiO_2 can be replaced in large measure by Al_2O_3 and the viscosity will remain the same. This fact is of the utmost importance in this investigation, since Al_2O_3 has a negligible cutting action on a basic hearth in comparison with SiO_2. By replacing SiO_2 in the receptor slag by Al_2O_3 a slag of low viscosity and high
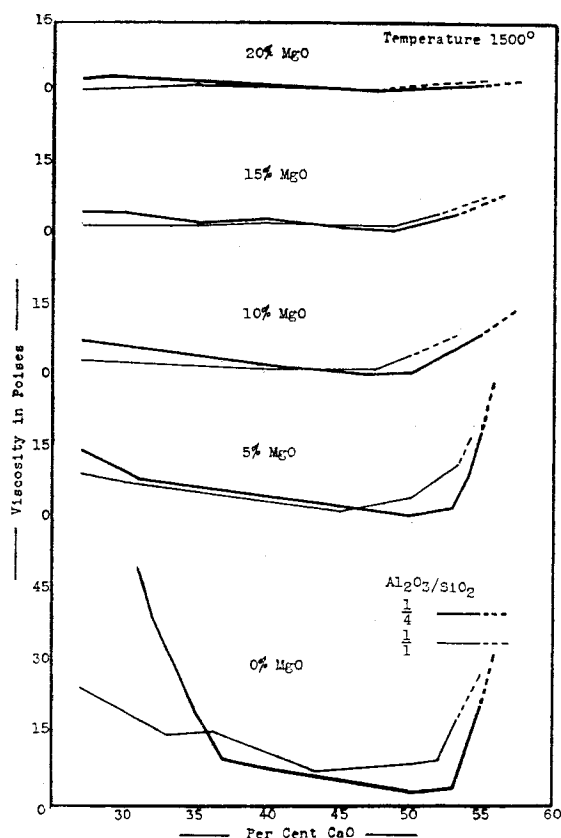

FIG. 1.

basicity may be obtained. In this way, it is possible to cut down the large slag volume, this being hitherto a severe handicap to the direct reduction processes.

Summarising the above considerations affecting slag composition, we find that the initial or receptor slag should be—

(i.) as low in bulk as possible and therefore

(ii.) as basic as possible,

(iii.) composed of at least 20% MgO

(iv.) as high as Al_2O_3 as in silica,

(v.) composed of at least 6% Cr_2O_3 and

(vi.) as low in FeO as possible.

Calculation of Charges.

In preliminary calculations it was found that if a slag of MgO and CaO content equal to 50 to 60% was made as the receptor slag, and if silica was allowed to alter from 15% in the receptor slag to 30% in the final slag, that the MgO and Al_2O_3 contents remained practically the same in both receptor and final slag. The following limiting compositions for receptor and final slags were decided upon. It is probable that the more basic receptor slag could have been used without sacrificing slag fluidity.

Constituent.	Receptor.	Final.
SiO_2 not greater than ..	15%	30%
Al_2O_3 not less than ..	15%	15%
CaO + MgO not less than	55%	45%
MgO not less than ..	20%	20%

Because of unavoidable extraneous oxidation more than the theoretical quantity of ferrosilicon will have to be used. The efficiency of the reactions is not known. Charges were made up assuming efficiencies from 70% to 90% for silicon in combining with the oxygen attached to both iron and chromium in the chromite ore. The same slags were used for all charges except No. 1. The quantity of slag of course, varied with the quantity of ferrosilicon charged. If the above limits are to be adhered to, the weight of slag may be calculated for each charge as follows. The weight of metal product was 500 gm. in each case.

Let x = weight of receptor slag.

y = weight of SiO_2 introduced by oxidation of all silicon added.

z = weight of "gangue" introduced by reduced chromite.

x + y + z = weight of final slag......1

Then $\cdot 15x + y = \cdot 30(x + y + z)$

i.e., $\cdot 15x = \cdot 85y - \cdot 3z$

Solve for x, since y and z are known.

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The following mixtures of ferrosilicon and chromite for different efficiencies of Si in removing O_2 of FeO and Cr_2O_3 were made up, for 500 gram of finished steel.

Description of Experiments.

The furnace used was a small laboratory indirect arc electric furnace. It was lined with Austrian magnesite brick which had

TABLE No. 1.

Charge No.	1	2	3	4	5
Per cent. Cr required	14	13	13	13	13
Per cent. Effy. FeSi	69	75	69	77	88
Chromite gms.	215	200	203	203	203
Fe-Silicon gms.	77	66	73	65	57
Receptor Slag gms.	420	370	400	370	340
Final Slag gms.	582	517	554	518	482

Slag for charge No. 1 was made up of pure calcined magnesite, limestone, silica powder, corundum and chromite. The theoretical slag composition contained—

CaO	MgO	SiO$_2$	Al$_2$O$_3$	FeO	Cr$_2$O$_3$
50·0	13·0	15·0	13·0	3·0	6·0

The receptor slags for charges 2, 3, 4, 5 were composed of a pure calcined magnesium limestone, corundum, silica powder, and chromite. Sufficient of this was prepared in bulk for all these charges, and weighed off as required. The theoretical composition of this slag was as follows :

CaO	MgO	SiO$_2$	Al$_2$O$_3$	FeO	Cr$_2$O$_3$
37·0	22·5	16·2	15·6	2·7	5·9

The theoretical compositions of the final slags are given below.

been ground to pass a 10-mesh screen. This was moistened with water and rammed in to give a bowl shaped cavity, 7″ wide at the top and having a depth of $3\frac{1}{2}$ inches. The roof was composed of a mixture of 60% alundum powder, 20% of a good grade firebrick ground to 10 mesh, and 10% of fireclay. This was moistened with water and rammed to form a dome shaped roof. After drying thoroughly the hearth and roof were fired to 1400° by introducing gas burners at the electrode holes. This heating was allowed to proceed for five hours. The burners were then removed and the electrodes introduced and the temperature raised in preparation for a charge. This took about 10 minutes.

TABLE No. 2.

Charge.	1	2	3	4	5
CaO	36·0	26·5	26·7	26·4	26·1
MgO	13·5	20·4	21·8	20·4	20·6
SiO$_2$	29·4	30·4	29·0	29·5	29·4
Al$_2$O$_3$	14·6	16·6	16·3	16·6	16·8
FeO	2·2	1·9	1·9	1·9	1·9
Cr$_2$O$_3$	4·3	4·2	4·3	4·2	4·2

The following is a table of complete charge weights for the production of 500 gm. of finished steel.

Charge No. 1.

The hoop iron was introduced in portions at a time. The power input was 11·4 K.W.

TABLE No. 3.

Charge.	1	2	3	4	5
Hoop Iron	372	400	400	400	400
Chromite*	215	200	203	203	203
F.-Silicon*	77	66	73	63	57
Receptor Slag	420	370	400	370	340

*Both ground to 60 mesh.

(300 amps at 38 volts). Charging took about 7 minutes. The furnace was then well closed and left for a further 7 minutes, when all the metal was fluid. The receptor slag was now charged, in about 50 gram. portions at a time, and the furnace closed until this had melted before the next portion was charged. Within 15 to 20 minutes all had been charged and was in a molten condition. 25 gm. of fluorspar was added to this charge, this being the only charge where fluorspar was used.

The chromite ferrosilicon mixture was now added in 15 to 20 gm. portions at a time, the arc of necessity being maintained during the whole of this operation. The furnace was sealed after each addition. Reaction took place fairly quickly, the small prills of reduced metal floating on the surface of the slag for a short time before sinking down into the metal. All chromite and ferro-silicon had been charged in 15 minutes.

The charge was now thoroughly stirred with an iron rod and allowed to settle with the power on for a further 10 to 15 minutes. The tap hole was opened, this having been closed during the above operations with asbestos wool, and the metal and slag poured into a heated cast iron mould by tilting the furnace. The ingot was cooled down to about 400° C. in the mould and then removed.

The ingot obtained measured $5'' \times 1\frac{1}{2}'' \times \frac{3}{4}''$, had a smooth dipped surface and appeared to be sound throughout. It weighed 630 gms. the excess iron having entered from the stirring rod. In subsequent experiments the precaution of covering the rod end with slag before stirring cut down this increase.

On attempting to sample this specimen no impression could be made with a drill, and a section could not be cut with a hack-saw. This was to be expected, since the rate of cooling was rather rapid.

The sample was wrapped up in asbestos wool and fireclay, and placed in a muffle furnace. The temperature was raised to 800° C. and maintained at this temperature for 10 hours. It was then cooled down in the furnace to 400° C. over a period of 30 to 40 minutes. The specimen could now be sawn and drilled, it being in the fully annealed condition. The metal analysis is shown in Table No. 4. Slag was not sampled or analysed.

Charge Nos. 2, 3, 4, 5.

These were all treated in exactly the same manner as Charge 1, except that no fluorspar was added at any stage. The slags initially were of a creamy nature, but were rather thinner finally. All the specimens were heated to 850° to 880° C. for from eight to ten hours and then cooled over the range 850° to 500° C. for 1½ hours. A completely annealed material should result. The samples were all easily drilled and sawn. Table No. 5 gives the slag analyses, these samples being taken twice for each charge. One sample after the receptor slag was in good shape and a second just before tapping.

All metal specimens were $3\frac{1}{2}'' \times 1\frac{1}{2}'' \times \frac{5}{8}''$, and appeared to be sound, all showing the characteristic dipped surface of a well-deoxidised steel. The analyses of these samples is given in Table No. 4.

TABLE No. 4.

Charge Number.		1	2	3	4	5
METAL	Chroium	10·2	12·2	9·1	11·7	9·5
	Carbon	0·24	0·26	0·49	0·46	0·73
	Silicon	1·50	0·60	3·50	1·12	1·40
ANALYSIS	Manganese	0·12	0·14	0·28	0·25	0·22
	Phosphorus	0·02	0·01	0·01	0·01	0·01
	Sulphur	0·02	—	—	—	—
Weight on ingot (in grams)		630*	500	500	520*	500
Weight of chromium added (grams)		70	65	65	65	65
Weight of chromium obtained (a) grams		66	61	46	61	47·5
Per cent. Chromium lost		5·7	6·2	29·2	6·2	27·0
Weight of silicon added, grams		54·8	47·0	52·0	46·3	40·6
Weight of silicon in metal, grams		9·4	3·0	19·5	5·8	7·0
Weight of silicon for reduction, grams		45·4	44·0	32·5	40·5	33·6
Weight of Si for theoret. reduction of (a)		35·4	32·9	24·8	32·9	25·6
Per cent. Effy. of Silicon in Reduction		78	75	77	81	76

*Fe entered bath from the mild steel stirring rod.

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<div align="center">TABLE No. 5—SLAG ANALYSES.</div>

	Receptor Slags.				Final Slags.			
	2	3	4	5	2	3	4	5
CaO	35·1	33·2	34·8	32·0	23·1	24·0	19·2	19·6
MgO	21·4	25·0	20·7	25·4	26·0	25·4	35·0	27·2
SiO_2	17·6	20·4	18·4	20·1	28·2	26·5	27·2	27·2
Al_2O_3	16·4	16·5	16·2	17·8	17·3	18·0	16·3	17·0
Cr_2O_3	4·4	4·1	4·0	3·2	2·3	6·0	2·7	5·6
FeO	4·0	3·8	4·4	3·3	1·4	2·4	1·7	1·5
MnO	0·30	0·27	0·37	0·39	0·26	0·22	0·28	0·32

Discussion of results.

In a small furnace it is difficult to control the temperature of the charge. No temperature readings were taken ; but it is worth noting in two cases where the furnace was used for two experiments, the one following the other immediately, that the second experiment showed a smaller chromium reduction and greater silicon content in the metal. This is evident from table No. 4. Charge 3 followed 2 and 5 followed 4. Although not conclusive, the deduction which may be made and which is supported by theory, is that in experiments 2 and 4, the temperature was lower than in 3 and 5, and as shown elsewhere in this paper, *low temperatures favour the reduction of chromium*.

The efficiency of ferrosilicon in reducing the Cr_2O_3 of chromite, as calculated on the metal analyses, varies in the five experiments from 75% to 81% (Table 4). From the chromite ore analysis it will be noticed that the percentage of oxygen combined with chromium and iron. From this it appears that silicon is almost 100% efficient in reducing the oxides of iron and chromium in chromite, in the presence of these receptor slags. Experiments 1 and 4 show efficiencies of greater than 100%, and it is probable that FeO has been reduced from the receptor slag as well as from the chromite added.

The figure given in a paper read before the Manchester Association of Engineers is 60% to 70%. This figure will naturally vary, being affected considerably by the furnace atmosphere and the FeO content of the chromite. A furnace which can be well sealed during the reaction will show a higher efficiency than otherwise.

The slags are unavoidably bulky, being about equal in weight to the bath of metal treated. This will decrease considerably the output of the furnace. The chromium in the receptor slag was rather lower than the 6% aimed at. By increasing this to say 10% and maintaining the added silicon close to the reduction efficiency, as calculated above, there is good reason to believe that the desired Cr content of the metal can be obtained, and that the silicon content of the metal will be low.

In the final slags for charges 3 and 5, the Cr_2O_3 content has actually increased notwithstanding the fact that the bulk of slag increased 1·4 times. This is in accordance with the low chromium and high silicon contents of the metallic product. As remarked above, this is probably due to too high a temperature.

Slags 2 and 4 are low in dissolved oxides and correspond in analysis to white deoxidising slags. They suggest that the metal bath is in a deoxidised condition. Polished sections examined under the microscope showed that charges 2, 3, 4, 5, were remarkably free from solid and gaseous inclusions, considering that the bulky slag and metal were poured together directly into the mould. Charge No. 1 was rather high in inclusions. In practice it would probably be advantageous to remove most of the final slag at the end of the reduction period, and supplement the remaining slag by lime fluorspar and ferrosilicon, as in standard practice for the indirect processes. The final slag besides degasifying the metal is also a strong desulphurising agent, if high in lime and fluorspar. Sulphur probably would have to be reduced in this way.

Slags for charges 2, 3, 4, 5 were all sufficiently fluid without the addition of fluorspar. Besides this, the slags were always basic. The corrosion of the furnace walls and roof would, as a result, not be excessive. In practice such slags can be made up using

a limestone high in magnesia. This would mean an important saving in conversion costs.

By the observance of correct temperature limits, controlled slag composition, sealing of the furnace during the reduction period, and suitable proportions in the chromite-ferrosilicon mixture, steel containing at least 12% Cr and not greater than ·75% Si should be possible. The reactions upon which this process depends are apparently of a subtle nature. If "off" heats are to be avoided, a strict observance of all factors contributing to the disturbance of the reactions must be maintained.

From the experiments carried out, no statement can, with certainty, be made with regard to the carbon content of the metal. In this small furnace, using D.C. the carbon arc was not more than $\frac{3}{4}''$ away from the slag surface during reduction and had to be maintained continually during this process in order to prevent chilling of the charge. The electrodes behaved differently in all experiments. At times only small amounts of carbon were derived from the electrodes, while in other cases flakes of carbon could be observed falling on to the slag. A.C. would probably obviate this. However, from a study of the tables on page 7, it will be seen that low carbon material is possible by this process. The use of high voltages and consequently, long arcs is known definitely to cut down carbon pick up. To-day these long arcs are being used in the remelting of high Cr scrap steel with a good measure of success.

Power Consumption Estimates.

In order to increase electrical and overall efficiency of electric arc furnaces, the modern trend has been towards higher voltages ; especially during the melting down period in the working of a charge. High arc voltage means the use of a long arc, and this is ideal for the production and melting of high chromium material. Quoting Robiette[11] "Long arcs are essential......in order to avoid contact of the charge with the electrode,

and to melt down as rapidly as possible without allowing the unprotected charge to be in contact with carbonaceous gas." The phase voltage generally in use to-day is 225, *i.e.*, a voltage of $\frac{225}{\sqrt{3}}$ or 140 V between electrode and bath. Electrical efficiencies of 90% are readily obtained and power factors of 80 to 90% are attainable. In the estimate for power consumption which follows a figure of 75% overall thermal efficiency will be taken, *i.e.*, 75% of the power supplied to the electrodes is available for heating the charged materials.[12]

An estimate of power consumption in any metallurgical operation naturally demands a knowledge of the theoretical heat requirements for the various melting, reaction, and slagging operations. In the direct method of high chrome steel production, since the slag bulk constitutes a large proportion of material charged, it is essential to know fairly accurately the thermophysical data of the slag obtained. In this connection very little is available. The thermophysical data used below have been taken from the "International Critical Tables."

The closest approach to the slag which will be used in practice is the compound $3CaO.Al_2O_3.2SiO_2$, this containing 43% CaO, 26% Al_2O_3 and 31% SiO_2. The latent heat of formation of this silicate from the constituent oxides works out at 63 K.W.H. per ton.
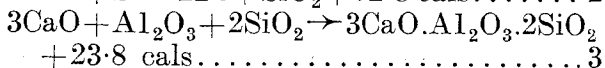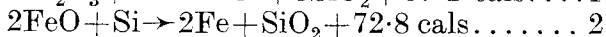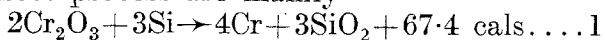
In computing the thermophysical data for slag the melting point of the slag has been taken as 1400°C. The specific heat of the molten slag has been taken as 0·3 cals per °C, this being the average of the specific heats of the constituent oxides at 1400° C. The mean specific heat of the constituent oxides from 0° to 1400°C average out at 0·26 cals per °C. The following table gives the heat in K.W.H. per ton required to raise the temperature of the materials given from 0° to 1650°C. Below this is the heat which would have to be supplied by the electrical equipment assuming an efficiency of 75%.

TABLE A.

	Heat of Formation on Slagging not allowed for.				Latent Heats and Heats of Formation allowed for.			
	CaO	MgO	Al_2O_3	SiO_2	Fe	Cr	70% Fe.Cr	Slag
Theoretical K.W.H. ..	386	535	477	448	330	282	296	411
Actual K.W.H. supplied at Electrodes	515	713	635	597	440	376	394	548

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Physical Chemistry of Reduction Period.

The reactions during this period in the direct process are mainly

$$2Cr_2O_3 + 3Si \rightarrow 4Cr + 3SiO_2 + 67\cdot4 \text{ cals} \dots 1$$
$$2FeO + Si \rightarrow 2Fe + SiO_2 + 72\cdot8 \text{ cals} \dots 2$$
$$3CaO + Al_2O_3 + 2SiO_2 \rightarrow 3CaO.Al_2O_3.2SiO_2$$
$$+ 23\cdot8 \text{ cals} \dots 3$$

The above heats of reaction are those obtained if the reactants and resultants are assumed to be at 18° C. The resultants have to be heated to 1650° C. If chromite and 70% ferrosilicon are used during the reduction period, then Table B gives the quantities of reactants and resultants, together with the thermophysical data for the reactions per ton of chromium produced. Ferrosilicon has been assumed to be 90% efficient in reducing chromite.

The method of computing the power consumption will firstly be applied to the indirect method for stainless steel production. If this compares favourably with published information for modern practice, then the methods outlined in the following table can be applied to the direct reduction process with reasonable justification.

power off. In the table below the K.W.H. requirement for melting the alloying additions has been considered as being introduced simultaneously with the alloying elements. This has been done for the sake of simplicity. Alternatively the heat requirement could have been considered as being introduced before the alloy additions, by superheating the charge to the correct temperature. This is what is done in practice; but the effect is the same in both cases.

The operation of a 6 ton charge in a Heroult furnace will be considered. It is the practice to employ high K.W. input for the melting down period. For subsequent operations a lower heat input is used in order to protect the roof and furnace banks from intense arc radiation[13]. During the settling period, for instance, once the charge is at the correct temperature for tapping the only heat input required is that necessary to balance the heat losses from the furnace. In a 6 ton furnace this will not be greater than 50 to 60 K.W. per ton of charge.

TABLE B.

Tons Chromite.	Tons 70% F-Silicon.	Chromium Tons.	Iron Tons.	Tons slagged SiO₂ and Gangue.	K.W.H. Total for Fe, Cr, Slag
3·00	0·84	1·00	0·77	2·12	
K.W.H. Evolved		342	355	135	832
K.W.H. Required for Resultant to 1650°		282	254	920	1456
Theoretical K.W.H. to be supplied for Reaction					624
Actual K.W.H. to be supplied at Electrodes					834

Power Consumption for Indirect Process.

There are five major operations employed in the making of alloy steels in basic electric furnaces.

(i.) Melting down of iron and slag.
(ii.) Oxidation of impurities and slagging.
(iii.) Making the deoxidising slag.
(iv.) Adding alloying elements.
(v.) Settling.

Each of these operations involves definite heat exchanges and the time for any one operation will depend on the rate at which heat is supplied by the arc. The addition of alloying elements is usually done with the

The oxidising slag usually constitutes from 4 to 8% of the charge. Owing to the fact that the carbon content of the bath has to be reduced to 0·04%, a figure of 8% will be used. The final deoxidising slag constitutes 4 to 8%. A value of 7% has been taken. A loss of 4% of scrap charged has been allowed for[14].

Table C explains itself.

Excluding the time for charging, pouring, and furnace repairs, the time to work a 6 ton heat will be a little over 4 hours with a power consumption of 697 K.W.H. per ton of steel poured. Allowing for charging, pouring and repair time, a 6 *ton heat will*

TABLE C.

Operation.	Charge.	Tons (2,000 ℔)	K.W. Input.	Total K.W.H. Required.	Time of Operation.
Melting[15]	Scrap	6·0	1350	2640	1·96
Down	Slag	0·5	1350	274	0·20
Oxidation Slagging[15] . .	—	—	900	450	0·50
			zero	zero	0·25
Deoxidation[15]	Slag	0·4	900	219	0·24
Alloying[15]	70% F-Chrome	1·07	900	421	0·47
Additions	F-Silicon	0·05	—	—	—
Settling[15]	—	—	360	180	0·50 ·
Total Power Consumption and Operating Time				4184	4·12

take from 5 to 5½ hours with a power consumption of say 700 K.W.H. per ton. These figures agree remarkably well with those for standard practice. A. L. Field gives 650 K.W.H. per ton and 5 hours[16]. Reference to data given by Stansfield, Barton, Robiette and Sisco corroborate the above estimate[17]. It should be noticed that in some cases the K.W.H. consumption per ton refers to the metric ton and others to the long ton (2,240 lbs.). In Fig. II. a graphic representation is given.

Power Consumption for Direct Reduction Process.

Computation of the power requirement for the direct reduction process can therefore,

in view of the foregoing, be made in a similar manner with some confidence.

The major operations to be carried out are :—

(i.) Melting down scrap iron and slag ;
(ii.) Oxidation of impurities and slagging ;
(iii.) Building up a receptor slag ;
(iv.) Adding the chromite ferrosilicon reducing mixture ;
(v.) Removing the final slag ;
(vi.) Building up a deoxidising slag ;
(vii.) Settling.

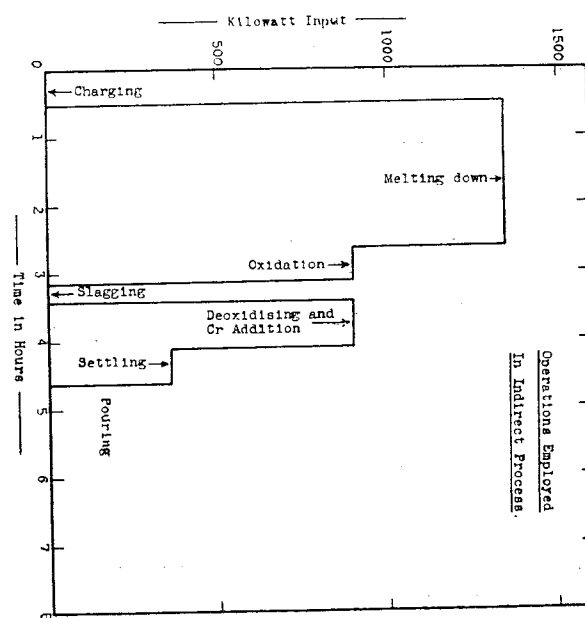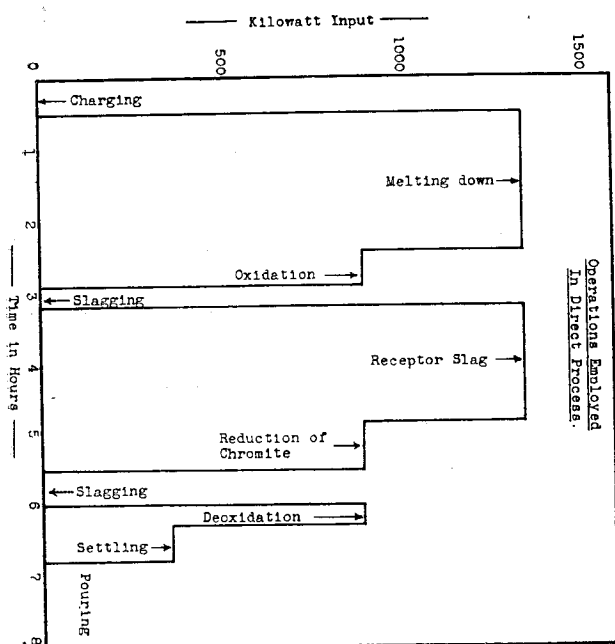Table D, overleaf, outlines the power requirement and time taken for each operation for a 6 ton charge.

Fig. 2.

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TABLE D.

Operation.	Charge.	Tons (2,000 ℔)	K.W. Input.	Total K.W.H. Required.	Operation time. Hours.
Melting .. .. Down	Scrap Slag	5·42 0·50	1350 1350	2390 274	1·77 0·20
Oxidation .. .. .. Slagging	—	—	900 zero	450 zero	0·50 0·25
iii.	Receptor Slag	4·00	1350	2200	1·63
iv.	Reducing Mixture	2·88	900	624	0·70
v.	—	—	zero	zero	0·50
vi.	Slag	0·40	900	219	0·24
vii.	—	—	360	180	0·50
Total K.W.H. Input and total Operating Time..				6337	6·29

The time taken to produce 6 tons of steel will be 6·3 hours with a power consumption of 1056 K.W.H. per ton. Taking into consideration time taken in repairs, charging and pouring *the time which will be taken to produce 6 tons of stainless steel by the indirect method will be between 7 and 8 hours with a power consumption of 1100 K.W.H./ton.* This also is represented graphically in Fig. II. and a comparison demonstrates clearly the differences in the two processes.

PRODUCTION COST ESTIMATES.
1.—*Power.*

The most suitable locality for the production of stainless steels from raw chromite will probably be Witbank, since here a cheap source of power is available. Besides this, Witbank is close to the source of chromite, and near a large consuming centre for metallurgical products.

The following tariffs have been taken from the "Licence granted to the Electricity Supply Commission by the Electricity Control Board under the Electricity Act of 1922, for the supply of electricity from an undertaking to be established at Witbank, in the District of Middelburg, Transvaal Province."

The standard prices chargeable by the Licensee for the supply of electricity shall, subject to the provisions of the Act, be as follows :

(*a*) A charge of £25 per annum payable in advance, plus

(*b*) A charge of 8*s*. 4*d*. for each kilovolt ampere of the maximum demand supplied in each month, plus

(*c*) A charge of 0·045*d*. for each unit supplied.

"Maximum demand means the highest load in kilovolt amperes supplied by the Licensee to the consumer during thirty consecutive minutes in each month."

(*a*) *Demand Charge.*—The diagrammatic representation of power consumption indicates that the maximum demand supplied to the electrodes is 1350 kilovolts. Assuming a power factor of 0·8—this being the lower limit for modern furnaces—the maximum demand, as defined above, will be 1700 kilovolt amperes. For both the direct and indirect processes then, the demand charge will amount to

£1,700 × 0·417 per month,
i.e., £708·33 per month.

(*b*) *Energy Charge.*—Studying the diagram again gives the maximum output per month for the two processes as

(i.) 900 tons per month for the indirect process ;

(ii.) 600 tons per month for the direct process.

The energy charges, based on the estimates obtained above for power consumption will be

(i.) $\dfrac{900 \times 700 \times 0\cdot045}{240} = £118\cdot12$ per month

(ii.) $\dfrac{600 \times 1100 \times 0\cdot045}{240} = £123\cdot75$ per month.

(c) *Fixed Charge (Annual).*—This amounts to £2.08 per month.

The above expenses are tabulated below.

4.—*Fixed Charges.*

Barton[20] groups the following charges under the head "Fixed Charges"—

TABLE E.

Process.	Monthly Tonnage.	Demand Charge.	Energy Charge.	Fixed Charge.	Total per month.	Power cost per ton.
Indirect	900	£708·33	£118·12	£2·08	£828·53	£0·92
Direct	600	£708·33	£123·75	£2·08	£834·16	£1·39

Cf. BARTON and ROBIETTE.

2.—*Cost of Labour.*

In the absence of any published information concerning the rates of pay of men employed on Electric Steel Furnaces in South Africa, figures will be taken from the Labour Section of the Government Mining Engineer's Report, 1934, for smelters, assistant smelters and unskilled whites, as employed on metallurgical works in the Transvaal.

Referring to Barton and Robiette [18] the following will be assumed as the direct labour required to operate a 6 ton Heroult unit per shift. Opposite are given the rates of pay applicable to the Transvaal :

1 melter	at £38·48 per month
1 helper	at £22·4 per month
1 ladleman	at £22·4 per month
4 chargers	at £7·2 per month each.
1 scrap craneman ..	at £27·0 per month

The cost of direct labour per month for both processes will be £359·64, *i.e.*,

Cost per ton indirect process = £0·40 (*cf.* Barton and Robiette).

Cost per ton direct process = £0·60.

3.—*Superintendence Charges.*

According to Barton[19] superintendence will require—

1 superintendent	at £1,000 per year.
1 melter	at £600 per year.
1 foreman	at £600 per year.
1 clearing room foreman	at £600 per year.

The yearly superintendence charge will be £2,800 per year., *i.e.*,

Superintendence for indirect process = £0·26 per ton.

Superintendence for direct process = £0·39 per ton.

Depreciation.	Laboratory Charges.
Amortisation.	Water.
Repairs.	Oil.
Ladles.	Electrodes.
Tools.	Office.

Barton estimates this for normal practice as £1·17 per ton. Robiette gives a figure for similar expenses as £1·25 per ton. Elsewhere Barton gives for actual practice £1·43 per ton.

Accepting Robiette's figure of £1·25 per ton for direct process, the indirect process will have an increased figure for this item since the yearly tonnage is less than for the direct process. It will be increased in the ratio $\dfrac{900}{600}$.

In the Union these fixed charges will, with the exception of water, oil, office, be greater than for oversea practice. It will be advisable, therefore, to increase these charges by 50% giving—

Fixed charges for indirect process = £1·88.

Fixed charges for direct process = £2·82.

5.—*Ferroalloys for Deoxidation.*

Barton[21] estimates this as £0·26 to £0·30/ton.

Robiette[21] gives £0·25 for fluxes and additions.

Assume Robiette's figure + 50%, *i.e.*, for direct and indirect process alloys = £0·38 per ton.

6.—*Slagging Material.**

1. Low silica hematite is probably best obtained from Postmasburg. This ore runs 60—67% Fe and could probably be put on rail at 3*s.* per ton. It is doubtful whether Iscor will be prepared to part with any of its iron ore at Thabazimbe.

*Kindly supplied by DR. BOSMAN, Department of Commerce and Industries.

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2. Dolomite occurs fairly extensively in the Transvaal and it may be possible to obtain suitable deposits near the factory site. Its price would probably be about 3/- per ton f.o.r. at mine. Pretoria would probably be the best source. Delmas, which is a bit nearer, may also prove to be a suitable source.

3. The Steelworks in Pretoria are using a 90% fluorspar, which costs 21s. 6d. per ton f.o.r. Warmbaths.

The railway rate on South African crude, untreated ores and minerals for South African foundries or factories is given as follows :—

In full truck loads : Tariff No. 9 less 10%.

If we consider Witbank as a suitable site the following table dealing with transport charges may be of value.

For a 6 ton charge the total quantity of chromite (48% Cr_2O_3) which will be required for the receptor slag and reducing mixture will be 2·6 tons. The cost of chromite at the Steel Works site will be reckoned as close on £1 10s. per ton.

Therefore cost of chromite per ton of steel produced = £0·65.

8.—*Ferrosilicon and Ferrochromium.*

The curves, Fig. III., give the values of contained chromium and silicon for certain grades of the ferroalloys. It will be noticed how the value of the chromium increases as the carbon content drops from 5% to 0·5%. In the case of ferrosilicon, it appears that the cheapest source of silicon is obtainable in the 50% to 70% grades. The 70% grade will probably be the best to use

Ore.	Transport.	Distance miles.	Rate per ton. (Tariff 9)
1. Iron Ore	Postmasburg to Witbank	550	16s.
2. Dolomite	Pretoria to Witbank	72	6s. 5d.
3. Fluorspar..	Warmbaths to Witbank	138	7s. 10d.

All amounts less 10%.

The materials used in slagging operations common to both processes, i.e., for oxidation and deoxidation, will be for a 6 ton charge of metal.

     Lime (calcined) .. .. 0·6 tons
     Iron Ore .. .. .. 0·1 tons.

For the direct process the additional lime and aluminous material required will be for the same charge—

     Lime (calcined) .. .. 3·0 tons
     Aluminous material .. 1·0 tons

The cost of iron ore, and slagging material will probably work out at £1 per ton on the foundry site for iron ore, and £2 10s. per ton for calcined magnesium limestone* Therefore cost of slagging materials per ton of steel will be

For indirect process £0·27 per ton of steel
For direct process £1·42 per ton of steel.

7.—*Chromite for Direct Process.*

South African Chrome Ore may be obtained from the Ore and Metal Company, Johannesburg. Its cost may be reckoned at 14/-—15/- f.o.r. Steelpoort.

since it contains least iron. This will be advantageous in cutting down the bulk of material to be charged as reducing mixture.

A communication from a shipping company gives the freight on ferrochromium and ferrosilicon from Southampton to Lourenco Marques and to Durban as 51/- per ton (2,000 ℔) and 43/6 per ton (2,000℔) respectively. Freight per rail is given under Tariff No. 4. The respective charges per 100 lbs. for transport from Durban to Witbank and from Lourenco Marques to Witbank will be 69d. and 38d. respectively, i.e., importing through Lourenco Marques total freight charges are £5·71 per ton (2,000℔) while through Durban the total freight is £7·93 per ton. Importation through Lourenco Marques will be assumed, and £6·0 per ton will be taken as total cost of transport, cartage, etc. The market prices of ferrochromium and ferrosilicon are given in trade journals as follows :

     Ferrochrome 70% Cr per 2,000℔ = £39·5
     Ferrosilicon 70% Si per 2,000℔ = £14·7

Ore.	Transport.	Distance miles	Rate per ton. (Tariff 9)
Chrome	Steelpoort to Witbank	203	10s. 0d.

Less 10%.

*Private communication from Consolidated Rand
     Brick Pottery and Lime Co., Ltd.

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Clearing and forwarding charges will amount to approximately £0·5 for ferro-silicon and £1·0 for ferrochromium per ton.

The ferrosilicon and ferrochromium requirement for the two processes per 6 ton charge will be—

Indirect process ferrochrome 1·07 tons (70% Cr)

Direct process ferrosilicon 0·62 tons (70% Si)

The cost of ferrosilicon and ferrochromium at the foundry works out at—

Ferrosilicon = £21·2 per ton

Ferrochromium = £46·5 per ton.

Therefore cost of ferroalloys for indirect process = £8·30 per ton

Cost of ferroalloys for direct process = £2·20 per ton.

9.—*Scrap Steel.*

Scrap steel will be valued at approximately £2·7 per ton at the foundry in Witbank. The indirect process requires 6·0 tons and the direct process 5·42 ton. for the production of 6·0 tons of finished product.

Therefore—

Cost of scrap per ton of produced steel indirect process = £2·70.

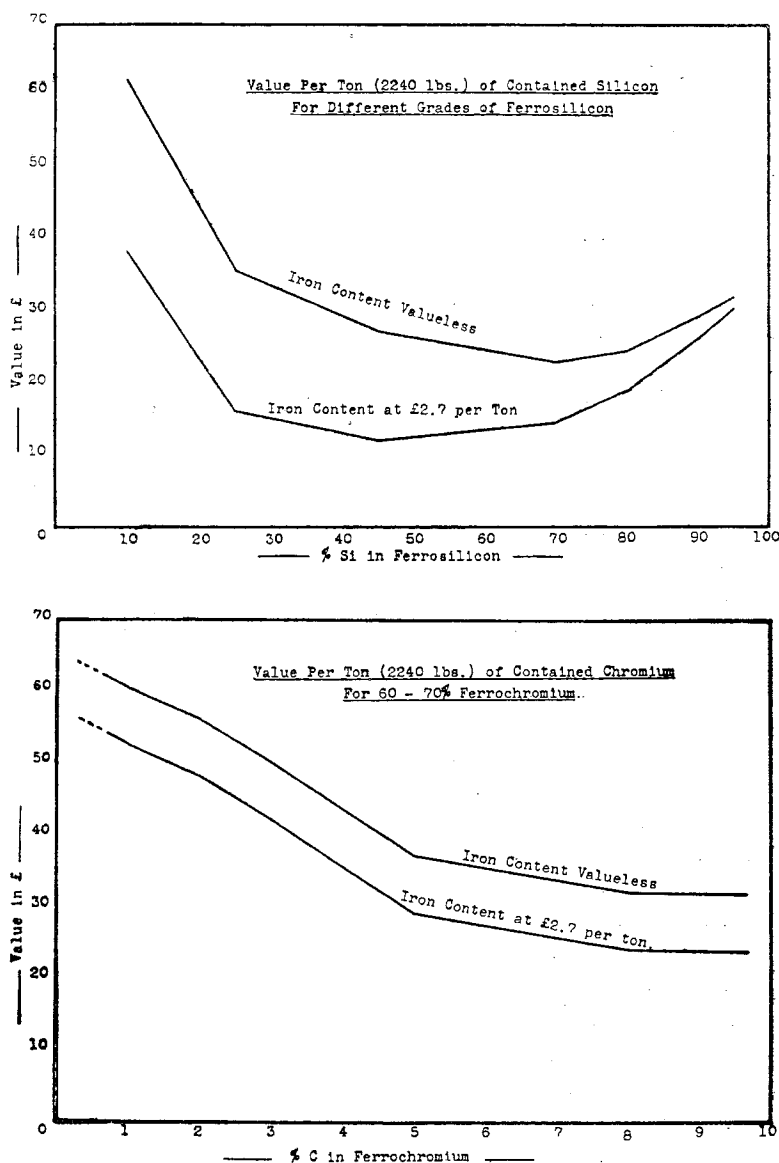Cost of scrap per ton of produced steel direct process = £2·44.

FIG. 3

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Tabulation of Total Production Costs.

Items of Expense.	Cost per Ton £	
	Indirect.	Direct.
Power	0·92	1·39
Labour	0·40	0·60
Superintendence ..	0·26	0·39
Fixed Charges	1·88	2·82
F.Alloys (Deoxidation)..	0·38	0·38
Slagging Materials ..	0·27	1·42
Chromite..	—	0·70
Ferrosilicon	—	2·20
Ferrochromium	8·30	—
Scrap Iron	2·70	2·44
Total	15·11	12·34

Comparing the above figures it appears that the direct reduction process offers attractive possibilities. It is not claimed that the above estimates will approach closely the actual production costs. It is, however, probable that the estimates for the two processes give relative costs and indicate that the direct process will be the cheaper of the two to operate. It should be noted also that no allowance has been made for the probable larger proportion of "off-heats" for the direct as compared with the indirect method.

CONCLUSION.

Assuming a sufficient demand for the product, the production of stainless steel containing 12% chromium by a direct reduction process appears to be a metallurgical and economic possibility, provided that all operations, together with physical conditions of the receptor slag, are rigidly controlled during the chromite reduction period. In the preliminary experiments described, although indications of the successful use of high magnesia slags in the absence of fluorspar have been obtained, further investigations under controlled temperature conditions will be required before more definite conclusions can be reached.

This work was carried out at the Minerals Research Laboratory, Witwatersrand University, under a Research Scholarship, granted by the Research Grant Board. The author wishes to express his indebtedness to Professor G. H. Stanley, the Director of the Research Laboratory, for valuable criticism and suggestions during the investigation, and in the compilation of the report.

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  ROBIETTE : "Electric Melting Practice," p. 70.
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    pp. 70, 343, 344.

The President said he now had very much pleasure in calling upon Mr. Clarke to propose a vote of thanks to Mr. Forsyth.

Mr. H. Clarke : I have not had time to give a detailed appreciation of the paper, but we get so few papers presented to the Society on the manufacture of steel that it gives me great pleasure in proposing the vote of thanks to the author. I must congratulate him on the extent of detail into which he has delved. As a maker of tool and alloy steels for many years, both crucible and electric, there are a few practical points and difficulties I would like to put forward. These practical points are unfortunately sometimes contradictory to accepted theory and are only run up against when one comes to produce on a practical commercial basis.

Making stainless steel in the electric furnace has never presented many difficulties, but, as the author points out, chromium, having a great affinity for carbon, the manufacture of stainless iron is a far more difficult task.

As against the author's description of the practice detailed on pages 321 and 330, it would be as well to state that as far back as 1920 stainless steel was made direct from a mixed charge of Swedish iron—ferrochrome and ferrosilicon. A reducing slag was obtained without slagging off. Charges of 25 cwts. were produced in 1½ hours. The type of furnace used was a Greaves Etchells, the hearth of which is conductive and therefore generates heat. This type of furnace I consider best for the making of high alloy steels as it is quicker in melting the alloys and also the bath of molten metal has a tendency to circulate. Many hundreds of tons have been made economically by this process. To-day steel, 35% chromium, 1% carbon, is being produced. In the case

of stainless iron, the electric furnace is not an economical proposition as the carbon content is most difficult to keep under ·12%. Therefore the process as described on pages 7 and 8 is the one generally employed to-day. It must also be borne in mind that makers of stainless iron are also producers of stainless steel. Therefore if a charge of metal is originally intended for iron and the bath analysis proves too high in carbon, the maker can always convert said charge to stainless steel.

The modern high frequency induction furnace must not be overlooked in the making of these high alloy irons and steels from ferro alloys.

One admits that the initial outlay is very expensive, but the running costs are comparatively low and the resultant product has been proved to be of an exceptionally high quality. This type of furnace is now extensively used and there are induction furnaces of over 5 tons capacity now in operation.

The author's costs of production are rather difficult to follow, and it has always been my experience that what should work out on paper does not do so in practice. I would, however, point out that the expense of stainless materials is not only because of the high cost in producing the ingot. I am afraid the cost of the steel as it exists in the ingot is relatively small compared with the cost of actually producing the finished articles, for which stainless steels and irons are required. These processes are numerous and the heat treatment plant alone is an expensive item.

It has long been appreciated that the cost of producing stainless iron and steel has retarded its use in many ways. Much research work has been carried out with a view to cutting down the cost of production and it is interesting to note that a Sheffield metallurgist (Mr. F. F. Gordon, my old chief) has patented a new process for producing stainless steels. I have not, as yet, received full details of this method but understand that it will revolutionise the stainless steel industry, not only from the point of view of cost, but also widening extensively the field of its uses.

Again I would like to thank the author for his most interesting paper and hope this will produce an interesting discussion on this subject.

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Symposium : Problems arising out of Temperature and Humidity in Deep Mines.

The President remarked that he now had pleasure in calling upon Mr. Delfos to say a few words in seconding the vote of thanks.

Mr. F. P. Delfos said he had much pleasure in seconding Mr. Clarke's vote of thanks to the author. He also had found the paper an extremely interesting one. He was not an electric steel man, but an open-hearth man.

One of the items which had struck him as most interesting in Mr. Forsyth's paper was his contention that slags with high magnesia were workable. They had tried to use the local dolomites instead of lime-stone, because they were cheaper in building up open-hearth slags, but up to now they had not been successful.

Another point which Mr. Clarke had mentioned was the matter of costs. He was afraid that Mr. Forsyth had been very optimistic in the matter of costs. He (the speaker) happened to have a few more details than Mr. Forsyth had presented to the Meeting, but he doubted very much whether one could get the men here he had budgeted for at the price which he had put in. He hoped that Mr. Forsyth would be able to continue his work on high magnesia content slags, for that was, from their point of view, of the utmost practical importance to-day. He wished to thank Mr. Forsyth very much for his paper.

The President remarked that he also would like to congratulate Mr. Forsyth on his paper, and hoped it was only the beginning and that Members would often hear him at Meetings of the Society.

Mr. Clarke had deplored the paucity of papers on steel. In that connection, he would like to make a suggestion—that was, as it seemed impracticable in this country to have a separate Steel Institute, the Society would welcome all the officials of Iscor and the Vereeniging Steel Works as Members of the Society, and it was hoped that in that way the Society would have many more papers on steel in the future.

The President then said he had much pleasure in calling upon Dr. Tromp to give his contribution to the—

"SYMPOSIUM : PROBLEMS ARISING OUT OF TEMPERATURE AND HUMIDITY IN DEEP MINING ON THE WITWATERSRAND."

SOME FACTORS AFFECTING THE CONDITION OF MINE AIR.

By PROFESSOR F. J. TROMP.

Mr. J. P. Rees, in a paper on "Dry Mining and the Cooling Power of Air" (this *J.* 1935, 8, 248), states that, "My object is to make it clear that a very great deal can be gained by bringing the air in as dry a condition as possible to the working places, even though the air leaves these places nearly saturated with water vapour." Mr. Ranson, in seconding a vote of thanks to Mr. Rees for the above paper, states that, "No benefit would appear to be derived at all on the mines of the Rand by having lined shafts as a means of maintaining dry walls or surfaces in the intake shaft system, when the use of water is compulsory in the mine workings or remainder of the circuit."

This divergence of opinions is intriguing, in view of the importance of the subject and also in view of the fact that a considerable number of factors entering into the problem depend upon fairly simple physical conceptions.

It is the object of this paper to examine some of the factors bearing on the wet versus dry mining controversy. The author has had neither the training nor the experience to arrive at a definite solution, but it is hoped that the results obtained may stimulate research in connection with this problem. As the calculations are somewhat involved the subject will be presented to the Society in an unusual form, in that the discussion and conclusions will precede the theoretical treatment.

Discussion and Conclusions.

A.—Let (comparatively) dry air of the same temperature and pressure pass down a