Cupola Design, Operation and Control

BCIRA

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The efficient and economic operation of ironfoundry cupolas is important both in the production of castings meeting stringent quality requirements and in the operating economics of a foundry. In this booklet consideration is given to how such operation can be achieved by improvement in existing practices and by application of developments in cupola design and operation resulting principally from investigational work at BCIRA.

The papers in the booklet provide a comprehensive review of cupola design and operation, factors affecting cupola operation and their control, materials used in cupola melting, controls and equipment that can be applied on the shop floor, and the handling of materials and stockyard layout. Attention is given also to present and likely future requirements to comply with clean air legislation concerning emission from the cupola furnace. The benefits to be derived from modifications to cupola operation and design, and treatment of molten metal are also considered.

In reviewing existing design and operating practices and ways of improving these and in considering application of developments, emphasis is placed not only on the technical aspects concerned but also on the economics involved.

Contents

Basic Design Principles

The design of a cupola has an important bearing on the efficiency and economy of its operation. Design features affecting the performance of the furnace are discussed.

Optimum blast rate as basis for design

A cupola melts most efficiently and economically when it is operated at a certain blast rate. Fig. 1, showing the relation between blast rate and metal temperature at a given metal:coke ratio, illustrates that, as blast rate is increased, the tapping temperature of the metal increases until a maximum value is obtained; after this a further increase in blast rate causes the temperature to decrease.

In practice, the optimum blast rate varies to some extent with the metal:coke ratio and the nature of the materials melted. However, it has been found, both experimentally and practically, that this optimum blast rate approximates fairly closely to 115m³/min per square metre (375ft³/min per square foot) of cupola cross-sectional area at the tuyeres. This figure may be taken as the first, fundamental basis of design. In Table 1, the optimum blowing rate is given, in column 2, for cupolas of varying cross-sectional area (col. 3) and diameter (col. 4).

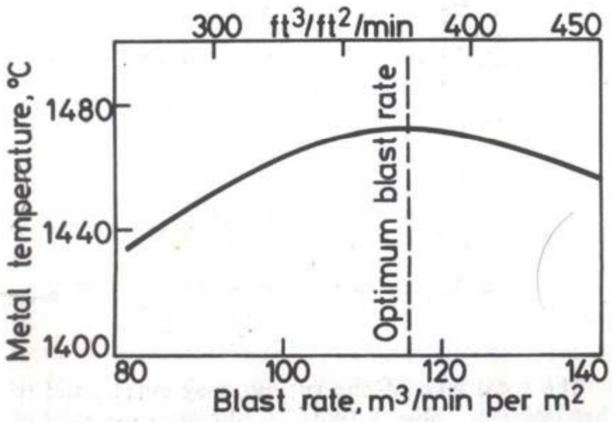

Fig. 1. Effect of blast rate on metal temperature at constant coke charge. (Schematic).

Cupola diameter related to melting rate

The melting rate of a cupola is dependent upon the metal:coke ratio and the blast rate. The melting rate of a cupola of given diameter operated at its optimum blowing rate will, therefore, depend upon the metal: coke ratio. Column 1 (Table 1) gives the melting rate obtained at the recommended (or optimum) blast rate, as a function of the metal:coke ratio.

Thus, to obtain a melting rate of 10t/h at a metal: coke ratio of 8.1 a cupola having an internal diameter of 122cm (48in) is required. The blowing rate should be 133.7m³/min (4720ft³/min).

It must be stressed that, in this example, the melting rate of 10ton/h is that obtained under steady uninterrupted blowing conditions. If allowance is made for off-blast periods, de-slagging and for the slower rate of melting at the start of the melt and towards the end of the blow-down period, the actual rate of molten metal production averaged over the day will be less than 10ton/h. This must be taken into account when specifying the required output. For example, to obtain an average hourly output of, say, 10ton/h, the cupola should be designed to produce 11 to 12ton/h when operated at the optimum blowing rate.

Table 1 Summarized cupola data – SI units.

1			2	3	4	5		6	7	8	9
Melting rate at various metal: coke ratios under steady operating conditions, t/h			Recommended blowing rate*, m³/min (15°C, 101kPA)	Cross-sectional area of melting zone, m²	Diameter of melting zone, cm	Recommended blower capacity		Approximate well capacity kg/cm height	Total tuyere area, cm²	Number of tuyeres	Approximate bed weight, kg/cm height
Metal:coke ratio						Volume, m³/min	Discharge pressure, kPa				
10:1	8:1	6:1									
1.6	1.3	1.1	18.8	0.164	46	22.7	10.0	6.0	225 - 420	4	0.7
2.2	1.9	1.5	25.5	0.223	53	30.6	10.2	8.0	325 - 550	4	1.0
2.9	2.5	1.9	33.4	0.292	61	40.2	10.5	10.5	420 - 740	4	1.3
3.7	3.1	2.5	42.3	0.370	69	51.0	10.7	13.3	515 - 935	4	1.7
4.5	3.9	3.1	52.1	0.456	76	62.3	11.0	16.3	645 - 1130	4	2.0
5.5	4.7	3.7	63.2	0.552	84	75.9	11.2	19.8	775 - 1390	6	2.5
6.6	5.6	4.4	75.0	0.657	91	90.0	11.5	23.3	935 - 1645	6	2.9
7.7	6.5	5.2	88.1	0.771	99	106	11.7	27.5	1100 - 1935	6	3.5
9.0	7.6	6.1	102.5	0.896	107	123	12.0	31.7	1290 - 2225	6	4.0
10.3	8.7	6.9	117.2	1.026	114	141	12.2	36.7	1450 - 2500	8	4.6
11.7	10.0	7.9	133.7	1.168	122	160	12.7	41.7	1680 - 2900	8	5.2
13.2	11.2	8.9	150.0	1.318	130	130	13.0	46.7	1870 - 3290	8	5.9
14.8	12.6	10.0	168.5	1.478	137	202	13.2	53.3	2100 - 3680	8	6.6
16.5	14.0	11.2	188.3	1.646	145	126	13.7	58.3	2350 - 4100	8	7.4
18.3	15.6	12.4	208.4	1.824	152	251	14.2	65.0	2610 - 4550	8	8.2
22.1	18.8	14.9	252.0	2.206	168	303	14.9	78.3	2870 - 5030	10	10.0
26.5	22.5	17.9	301.6	2.626	183	360	15.7	93.3	3740 - 6580	10	11.8
31.1	26.4	21.0	354.0	3.083	198	425	17.2	110	4390 - 7740	10	13.5
36.1	30.6	24.3	410.6	3.574	213	493	18.7	128	4510 - 9030	10	16.0

*Actual blowing rate specified at 115m³/min per m² to take into account satisfactory cupola operation at values within ± 20 per cent of this figure.

†Recommended blower volume capacity based on recommended blowing rate in column 2 plus 20 per cent; discharge pressure approximately 50 per cent higher than anticipated windbelt pressure.

Blower specification

Although a specific blowing rate of $115m^3/min$ per m^2 has been selected as a basis for fixing the diameter of the cupola relative to its output, in actual operation, the output can be varied by altering the blast rate. Normally, a cupola can be operated within a range of about ± 15–20 per cent of the optimum blast rate without serious consequences (in terms of metal-tapping temperatures) of underblowing or overblowing.

The recommended specification for the blowing equipment given in Table 1 (col. 5) therefore allows the cupola to be operated at a blast rate of about 20 per cent in excess of the optimum when required. The discharge pressure at the outlet of the blower must be sufficient to enable the required volume of air to be delivered against the resistance of the blast main, windbelt, tuyeres and, above all, the stock in the furnace. For a cupola of given diameter, this resistance may vary widely according to the nature of the cupola charge materials, but the values given should be adequate under most melting conditions to ensure that the required volume of air will be delivered to the furnace.

Tuyeres

The function of the tuyeres is to conduct equal quantities of air from the wind-belt into the cupola in order to produce uniform combustion conditions through the coke bed. Opinions concerning the correct tuyere area vary widely, but tests carried out under carefully controlled conditions at the BCIRA experimental cupola plant have shown that, provided blast rate was constant and blast distribution to each tuyere was reasonably uniform, tuyere size had no effect on the melting behaviour of a cold-blast acid-lined cupola. It is generally recommended, however, that the total area of the tuyeres should be one-fourth to one-seventh of the cross-sectional area of the cupola inside the lining. On this basis, tuyere area as a function of cupola diameter is given in Table 1 (col. 7). When tuyere area is within the range shown the tuyeres will be sufficiently large to prevent any serious loss of pressure through them and will not, therefore, have any very drastic throttling effect on the blower. On the other hand, a tuyere area which is too large may give difficulties due to uneven blast distribution, particularly with large-diameter furnaces. The shape of the tuyeres has little influence on the behaviour of the furnace, but simple box-type tuyeres are preferred by many designers because they are easily constructed and permit easy modification of area for controlling air flow if the distribution of air into the tuyeres is poor. By selection of the tuyere area within the range recommended, sufficient surplus area is made available to enable one or more of the tuyeres to be reduced in area to correct poor blast distribution – as indicated by repeated uneven burnout of the lining.

Within limits, the number of tuyeres does not seem to be very critical, but to improve uniformity of air supply to the coke bed the number of tuyeres should increase with the furnace diameter. The recommended number of tuyeres for cupolas of various diameters is given in Table 1 (col. 8).

A means of cutting off the air supply to each individual tuyere is of value. When the air supply to any one tuyere of the cupola is cut off by means of an efficient valve or shutter, any slag which has solidified around the inner end of the tuyere is melted away and, thus, the need for manual poking of the tuyeres can be largely eliminated. If the tuyeres are closed in rotation, the coke bed can be kept clear of serious obstructions and the regularity and uniformity of the melt can be improved.

Recent work at BCIRA and elsewhere has shown that the melting performance of the cupola can be improved by the use of two rows of correctly spaced tuyeres with the blast air divided equally between the two rows. By this means, higher metal-tapping temperature and higher carbon pick-up can be obtained for a given coke consumption. Alternatively, coke consumption can be reduced and melting rate increased while maintaining a given metal temperature.

Where two rows of tuyeres are used, the blast supply to each must be measured and controlled. The two rows should be supplied with air from separate windbelts. A typical arrangement is shown in Fig. 2.

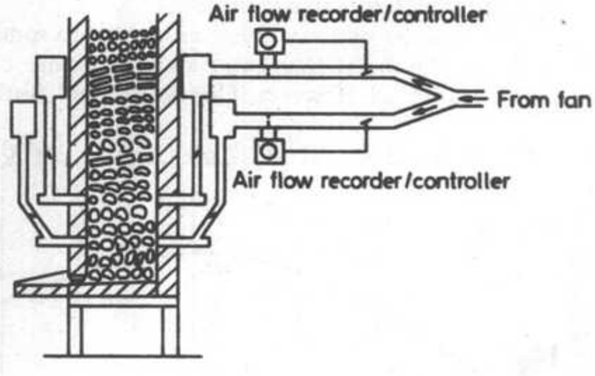

Fig. 2. The divided-blast cupola.

The total area of the tuyeres may correspond to that given in Table 1 (col. 7), but in convertion of existing cupolas, where it may be inconvenient or expensive to alter the size of the lower tuyeres, these may be retained at their original size. The total tuyere area of the upper row may then be the same as, or one-half of, the area of the lower tuyeres.

Shaft height

The function of the shaft – that portion of the furnace from the tuyeres to the charging door – is to accommodate a sufficient volume of metal and coke to absorb the greater portion of the heat from the ascending gases.

A recommended shaft height for cold-blast cupolas is 4.3–6m (14–20ft) see Fig. 3. With less than 4.3m (14ft) the stock is usually subjected to insufficient preheat and there will be a loss in thermal efficiency. A shaft height greater than 6m is not likely to give any worthwhile increase in thermal efficiency, and has the disadvantage that when corrections or changes in the charges are desirable the effect of such corrections will not be observed as quickly in the foundry, owing to the greater number of charges in the stock column.

Lining

The lining of a cupola should be thick enough to withstand the wear and tear of cupola operation, and may be made of firebrick or rammed refractory material.

Where firebricks are employed, a satisfactory method of installing the lining is to use two concentric rings of circle bricks or blocks of appropriate thick-

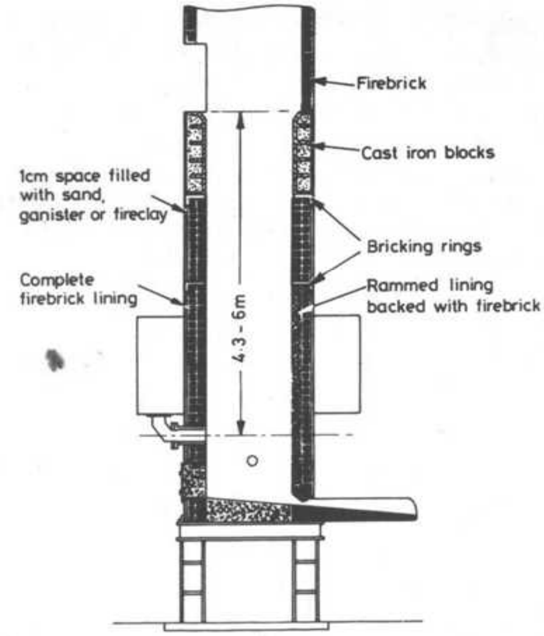

Fig. 3. Lining of cold-blast cupola.

ness correctly radiused for the internal diameter required (Fig. 3). This is economical, as the outer ring can be maintained for a considerable period, only the inner ring requiring replacement. Care should be taken to keep the joints between the bricks as thin as possible, otherwise slag attack at the joints will lead to poor lining life. Each course should be carefully laid and, from upwards of about 1.5 m (5 ft) above the tuyeres, bricking rings – generally in the form of angle-iron shelves – should be provided at intervals of about 1.5–2 m (5–7 ft) to support the lining. It is also advisable to leave a gap of about 1 cm (½in) between the bricks and the shell. This gap is filled with ganister, fireclay or sand and helps to take up expansion of the bricks during the initial heating, and also prevents the occurrence of hot spots on the shell if metal penetrates the joints in the brickwork. If a rammed lining is used, it is desirable to restrict its thickness to about 6 cm (4in) (Fig. 3) and to use an outer ring of firebricks, since a greater thickness of rammed material is difficult (but not impossible) to dry and fire.

In general, the thickness of the lining should not be less than 15cm (6in) except in very small cupolas operating for short periods. For melting periods up to about 6 hours, 23cm (9in) is usually satisfactory. For longer melting periods up to a complete single shift, a satisfactory lining thickness is about 30cm (12in).

The upper part of the shaft may be provided with hollow cast iron blocks to withstand the impact of the materials charged. Above the charging-door, the stack may be lined with a single layer of firebrick circle bricks (Fig.3).

Well depth

The well is that portion of the cupola between the tuyeres and the sand bottom. It serves to collect the metal and slag melted in the upper part of the furnace and permits the two to separate. In estimating the holding capacity of the well it is usual to regard its effective depth as the distance between the sand bottom and the slag hole. The slag hole is generally located at the rear of the cupola directly opposite the taphole, and is usually about 15cm (6in) below the tuyeres.

The method of determining the metal capacity of the well is shown in Fig.4a. Fig.4b. and Tables 1a. & b., give the well capacity per metre and (foot) of depth respectively for cupolas of various diameters. Naturally, if too great a volume of slag is allowed to accumulate in the well, the weight of metal which can be held will be less than that calculated from Figs. 4a & 4b.

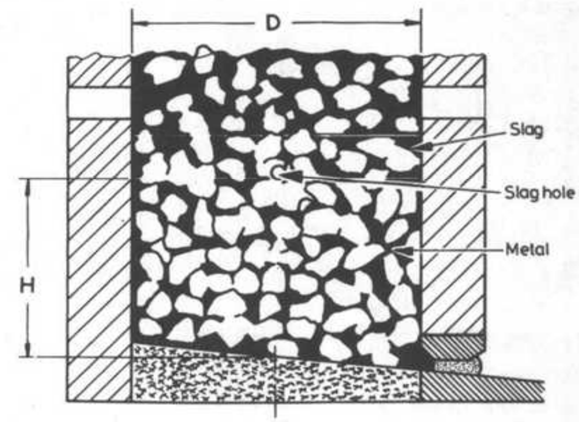

Fig. 4a.

Effect depth of well = Hcm

Volume = $\frac{\pi}{4} D^2 \ cm^3$

Density of molten iron = 0.007 Kg/cm^3.

Coke occupies approx. 50% of well volume,

$\therefore$ **Metal-holding capacity =** $\frac{\pi D^2 H}{4 \times 2} \times 0.007$ **Kg**

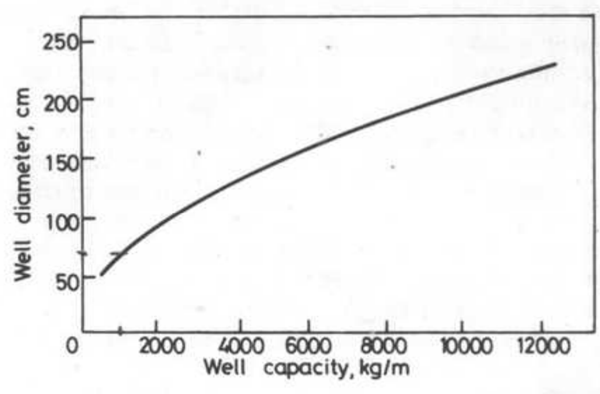

Fig. 4b.

Fig. 4. Metal-holding capacity of cupola well.

When the cupola is tapped intermittently, the well should have sufficient capacity to ensure adequate mixing of the molten metal. When mixtures of pig iron and cast iron scrap are melted, the capacity of the well should be equivalent to the weight of at least two metal charges. If the proportion of steel scrap in the charges is high, and particularly if any considerable quantities of ferro-alloys are used in the charges, the well should hold the equivalent of at least three to four metal charges. On the other hand, a well depth of 1m (3ft) is about the maximum advisable in normal practice to prevent undue temperature loss, and the weight of the metal charge should be selected to comply with the recommendations already outlined relating the well capacity to charge weight while limiting the well depth to within 1m.

It should be appreciated that adequate mixing of the molten metal to provide uniformity of composition will only be obtained if the metal is tapped from a full well.

In continuously tapped cupolas, the depth of the well should be just sufficient to prevent slag entering the tuyeres at the lowest anticipated blast rate to be used.

Practical Aspects of Cupola Operation

In many foundries the cupola is taken for granted, and simple faults in the practice are overlooked owing to familiarity with the plant. This can lead to serious trouble and waste of money.

The present paper deals with some common faults in cupola operation, encountered by the staff of BCIRA.

Lining repair

The first task at the start of the day is to patch the lining, to repair the wear and tear that has taken place during the previous melt. The reasons for this are to ensure that the furnace has the same internal dimensions before each melt since, if the well and melting-zone diameters are allowed to increase, the melting rate per unit area will fall and more coke will be needed to build up the bed to the required height.

Inefficient lining repair can be expected to give the following troubles:

(a) The tuyeres become slagged-over during a melt.

(b) Melting stops altogether owing to a build-up of slag in the furnace.

(c) A hot spot appears on the cupola shell during the melt.

(d) The cupola requires re-bricking after perhaps only one month's use of a new lining.

When such troubles are experienced, most foundrymen blame (incorrectly) the refractory materials, whereas in most cases the basic cause of the trouble is the manner in which the repair work is carried out. It is appreciated that it is difficult for a man to work inside a cupola, particularly a small one, but nevertheless the operator responsible for cupola repair should be encouraged to remember the following points fundamental to efficient repair work:

(i) Cleaning the inside of the cupola.

(ii) Moisture content of the patching material.

(iii) Drying-out the repair work slowly.

These can be remembered as:

Cupola Man's Duties

Cleaning the inside of the cupola Adhering slag and coke should be removed from the lining by chipping, as patching material will not adhere to slag and lumps of coke left on the furnace wall. It is not necessary to remove all the glaze from the lining, provided there is sufficient roughening of the surface to give a key for adhesion of the patching material.

Moisture content of the patching material This should be only sufficient to make the material plastic and workable.

Drying out the repair work slowly The repair should be dried out slowly before the cupola is put into operation. In conditions where the patching material is not dried out, and it is also impossible to clean the furnace thoroughly before application of the patching material, the patching will become detached from the furnace wall early in the melt. This can lead to severe slagging over the tuyeres, and in extreme cases a slag bridge can be formed and melting stopped. In addition, it is obvious that if the patching only remains on the furnace wall for a short time during the melt, erosion of the remaining original lining will commence early in the melt. When the furnace is prepared for the next melt, a thicker layer of patching material will be needed; this will be less well dried-out than on the previous day and is likely to become detached even earlier in the melt. Thus, a chain of circumstances will be set up which can lead to complete removal of a new lining from the melting zone, sometimes in less than a month.

This situation exists in many foundries having only one cupola which is used every day, and where there is insufficient time to clean out the cupola and dry the repair work correctly. In these circumstances poor lining life must be accepted, and it is unfair to blame excessive lining wear on the refractory or on the cupola man who is only doing his best to achieve an impossible task.

The internal diameter of the cupola in the melting zone should always be restored to the specified dimension, and this should be checked with a gauge stick. It is important also to ensure that the patching is applied to give equal wall thickness all round the furnace. This can be checked by measuring the distance from the inside wall of the furnace to the tuyere cover, at diametrically opposed positions in the furnace.

Many cupolas, particularly those of more than 100cm (40 in) diameter, are now repaired by application of monolithic patching material with a pneumatic gun. This reduces patching time, labour, and material costs, and gives consistent and satisfactory results. Points to watch are:

1. Cleaning As for ganister patching, adhering slag and coke on the cupola wall must be removed, but it is not necessary to roughen glazed surfaces as this patching material will readily adhere to such surfaces.

2. Support of newly applied patch A ledge should be provided to support the patch because, until glazing or vitrification takes place, the refractory has low strength. The method of providing a ledge will depend on the extent of the burn-back, but a row of bricks immediately above the tuyeres may be necessary.

3. Control of moisture content To reduce rebound and dust, there must be sufficient moisture in the material, but excessive or varying moisture contents can lead to spalling.

4. Patch application Minimum rebound is achieved by applying the patching material at right angles to the refractory surface, with a progressive circular motion. To prevent spalling of the lining, the patch should be applied to give horizontal layers by shooting the refractory downwards to build up the lining progressively from the tuyeres. A telescopic platform or suitable scaffolding should be provided to enable the operator to do the job properly.

The air pressure and volume should be maintained at the levels recommended by the refractory and the equipment suppliers. Failure to meet these requirements will result in poor lining performance.

Coke bed

A most important part of the successful operation of a cupola is the preparation of the coke bed, since the initial height of the bed above the tuyeres and the degree to which it is burned through before charging commences are vital factors, governing to a large extent the metal temperature and melting rate obtained during the early part of the melt. It is, also, usually impossible to remedy any defect in the bed preparation in less than one hour after the start of the melt.

The importance of these points is so well known that they are frequently taken for granted. In many foundries the preparation of the coke bed is never seen by the supervisory staff and the production of cold metal, which has to be pigged during a considerable time after the start of the melt, is accepted as normal. There is no reason why this should occur, since only four simple points have to be remembered.

I = Ignition
A = Addition
C = Consolidation
M = Measurement

A convenient way to remember these points is:

IACM – I Avoid Cold Metal.

There are many ways of igniting the bed coke. The method shown in Fig. 1, where wood, oily rags and occasionally coal are used, is still employed in many foundries, but has the disadvantage that the labour cost in collection and preparation of the kindling materials is high. It takes considerable time to ignite the bed coke, and smoke is also produced. This contravenes recommendations to local authorities made in 1968 in respect of the Clean Air Act.

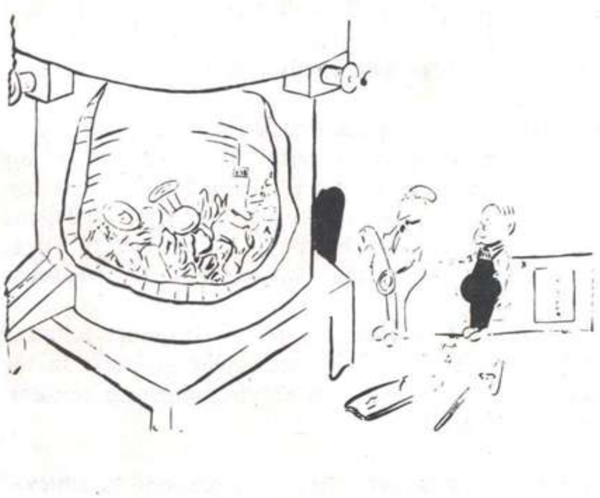

Fig. 1. Igniting the coke-bed, using kindling materials.

A better method, in fact the only acceptable method to comply with Clean Air legislation, is that shown in Fig. 2, where a torch operated with gas and air or oil and air is used. This is simple and clean to operate. Such equipment can be easily installed in most foundries, and if the appropriate fuel and air supplies are not available it is possible to obtain a portable arrangement on a trolley – comprising a fuel tank and a small compressor. Alternatively, an oxy-propane torch, which can be purchased as a complete unit, can be used.

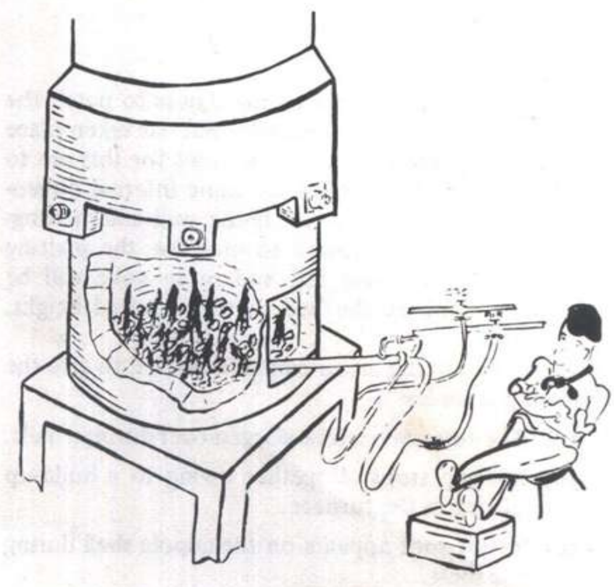

Fig. 2. Igniting the coke-bed, using a torch.

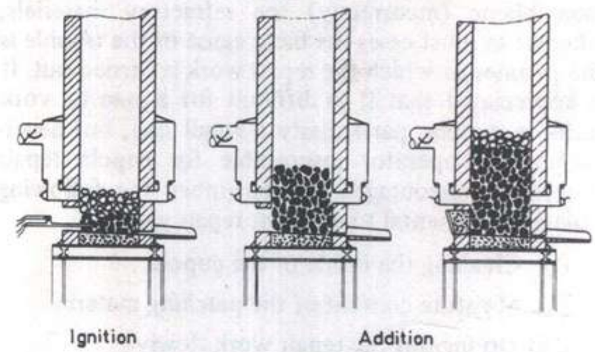

Fig. 3. Cupola coke-bed preparation.

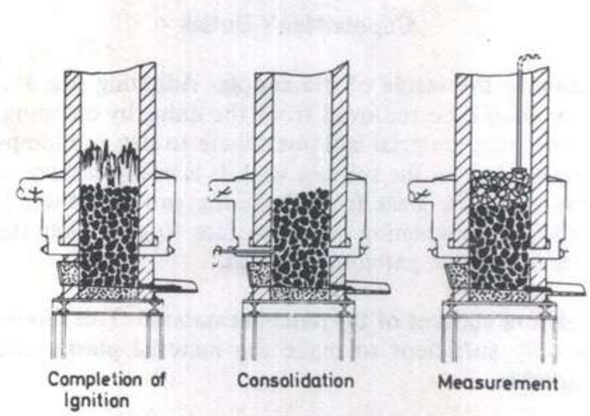

Fig. 4. Cupola coke-bed preparation.

In view of the importance of preparation of the coke bed in the successful operation of the cupola, it is worth while considering the steps required to carry this out correctly. These are shown diagrammatically in Figs. 3 and 4.

The bed is first made up to tuyere level by hand, and the coke ignited by a torch positioned a few centimetres above the sand bottom on a support of coke, with the jet near the centre of the cupola. The coke is then allowed to burn through, and when it has ignited uniformly the tuyere cover plates are opened and coke added through the charging-door as the bed burns through, until the top of the bed is about 30cm (one foot) lower that the level eventually required. When the bed has burnt through evenly to this level the fettling-door is made up.

The tuyere cover plates are then closed, and ignition of the bed completed by blowing for a few minutes using the cupola blower at a gentle blowing rate. It should be emphasized that only a few minutes are required. During this operation a portion of the air passes down through the well and pre-heats the tap-hole. This practice is particularly useful where the time available to prepare the bed is limited. When the fan has been turned off the tuyere cover plates are opened and the bed is poked, through the tuyeres, to close or consolidate any voids which may be present. Failure to carry out this simple operation is a frequent cause of low metal temperature at the beginning of the melt, and it is not unusual for the height of the coke bed above the tuyeres to be reduced by some 30cm by merely closing voids in the bed in this way. Coke is then charged to bring the bed height to its pre-determined level, and a layer of cold coke about 30cm (1 foot) in depth should be required. This permits burning through to a high state of incandescence while charging is proceeding and the production of hot metal at the start of the melt is assisted. It also prevents melting of the charge while the furnace is being filled, which can lead to a hard tap-hole.

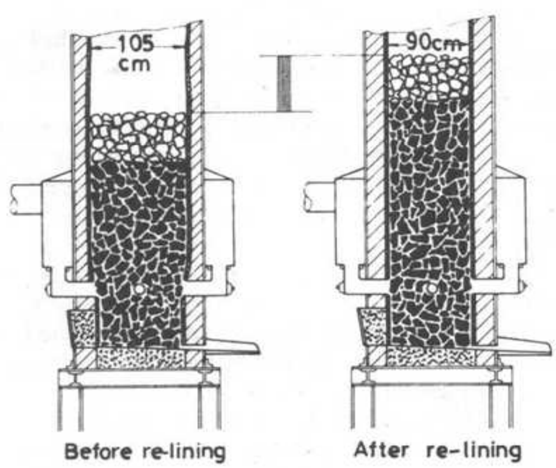

Fig. 5. Why measure coke-bed height?

The height of the coke bed above the tuyeres should always be measured before charging is commenced, and the optimum height has to be determined for each cupola and operating condition. It is unsatisfactory to use a standard weight of coke, since the height of the bed above the tuyeres will vary from day to day according to the internal size of the cupola. This can be illustrated by mentioning a problem in a foundry after a cupola had been relined during a holiday break. At the beginning of the melt there was an unusual delay in metal appearing at the tuyeres, and when metal was tapped it was of low temperature for a considerable time. The practice was to make up the bed by charging a standard weight of coke into the furnace, and the height of the coke bed above the tuyeres before charging was commenced was found to be over 2.5m (8 ft.). Before the holiday period (as shown in Fig. 5) when the cupola had an internal diameter about 1.5cm (6 in.) larger, the height of the bed for the same weight of coke charged was about 60cm (2 ft.) lower. This clearly shows the importance of measuring the height of the coke bed above the tuyeres, even where this may be difficult, as in installations using mechanical charging methods.

Hard tap-holes

A hard tap-hole is caused by blockage of the hole with solidified metal, usually at the start of a melt and occasionally after shutdown of the cupola for a long period, when solidified slag may also be present. It may result from:

1. Incorrect coke-bed preparation
2. Use of contaminated bed coke or kindling materials
3. An excessive length of tap-hole
4. A cold or damp tap-hole
5. Use of unsuitable botting material

Ways of preventing a hard tap-hole are:

Correct coke-bed preparation Ensure that coke in the cupola well is completely ignited, before making-up the fettling-door.

Avoid charging on top of an incandescent coke bed by completing preparation of the bed with a 23-30cm (9 to 12in.) layer of cold coke. This prevents the charge materials from melting prematurely during cupola charging.

Uncontaminated bed coke and kindling materials Use bed coke which does not contain any pieces of tramp iron and castings.

Use smokeless methods of lighting the bed coke, to avoid use of kindling materials containing nails and screws.

Correct tap-hole length For intermittently tapped cupolas, the length of tap-hole should be between 4 and 10cm (1½in. and 4in.) depending on cupola size and tap-hole diameter. For continuously tapped cupolas with large-diameter tap-holes, a greater length is permissable.

With thick cupola linings, reduce the length of the tap-hole by making a recess in the lining and, if necessary, at the exit of the tap-hole from the furnace.

Avoid a cold or damp tap-hole Reduce the volume of refractory to be dried at the tap-hole, by using a tapping brick.

Dry and preheat the tap-hole with a torch flame during furnace preparation.

With intermittently tapped cupolas, completely dry and preheat the tap-hole by leaving it open as the melt commences, until melting actually starts as seen by inspection at the tuyeres. Then partially fill the hole with black sand or a core, before inserting the bott. With continuously tapped cupolas, the tap-hole should be sealed with sand from the inside of the furnace before addition of the coke bed.

Botting material For intermittently tapped cupolas, use a suitable botting material.

Occasionally foundries are visited where this trouble with hard tap-holes is experienced, and invariably the foundry staff do not know how to deal with the problem. In many cases, by the time suitable equipment has been found it is impossible to open the tap-hole and production is lost.

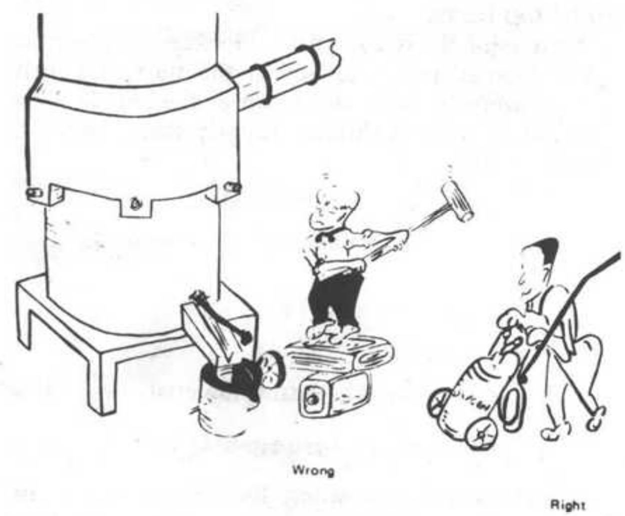

Fig. 6. Equipment for unblocking hard tap-holes.

Fig. 6 shows the essential pieces of equipment needed. These are:

Oxygen cylinder.

Regulator.

Flexible rubber hose.

Metal junction for flexible hose.

Steel tube 10mm o.d., 5mm i.d., 1.2-1.8m long, for intermittently tapped cupolas. A longer tube may be necessary for continuously tapped furnaces.

The procedure to open a hard tap-hole is as follows:

1. Heat the end of the steel tube to red heat, either by inserting it through a tuyere into the hot coke bed, or by using an oxy-acetylene flame.

2. Remove the tube from the tuyere, place it near to the hard tap-hole, and turn on the oxygen supply at low pressure (about 35kPa, 5lbf/in^2).

3. Apply the ignited tube to the hard tap-hole, inserting it cautiously into the hole as the solidified material in the hole melts.

4. Remove the tube when metal begins to flow through the tap-hole.

NOTE To avoid damage, and excessive enlargement of the tap-hole, a controlled lancing action can be achieved by using a thick-walled small-diameter steel tube and minimum oxygen flow.

The following safety precautions should be observed when carrying out this operation:

1. Do not connect the steel tube directly to the flexible hose and oxygen cylinder, since the rubber hose can ignite and burn back to the oxygen cylinder if the lance is dropped accidentally and burning continues. Always connect the steel tube to the flexible rubber hose with a metal adaptor.

2. Oxygen should only be supplied to the steel tube after the tube has been heated, otherwise the oxygen may assist ignition of combustible or smouldering materials in close proximity (e.g. clothing).

3. Two operators should be employed: one to control the oxygen supply at the oxygen cylinder, and the other to deal with clearing the hard tap-hole.

4. Ensure that the steel tube is firmly grasped while it is being supplied with oxygen.

5. Ensure that the operators wear safety glasses and asbestos gloves.

Tap-hole construction and botting clays

Many foundries operating with intermittently tapped cupolas pay insufficient attention to tap-hole design and the botting clay which is used.

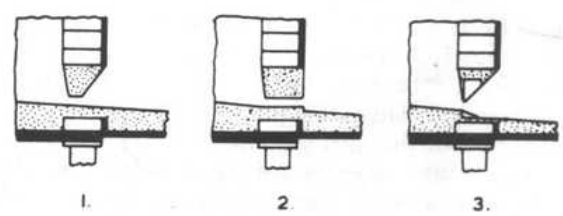

Fig. 7. Alternative shapes of tap-hole.

Several designs of tap-hole are illustrated in Fig.7, and experience has shown that Design 1 is the best to give consistently good results. The hole can usually be made up satisfactorily in ganister or fireclay to give additional strength. With this design it is often possible to tease the dried plug from the hole for tapping; the plug will then be flushed away in one piece by the molten metal.

The second shape of tap-hole, Design 2, is a straight-fronted tap-hole. This requires a very sticky and plastic bott which relies on adhesion to the front of the tap-hole for satisfactory use. On tapping, it is usually necessary for the residual clay plug to be pierced with a pointed bar; once metal is flowing, the heat melts out the remaining fragments of the plug.

The third shape, Design 3, has a deeply-coned front which has almost no parallel portion of tap-hole and is sometimes produced as a firebrick shape. Hard tap-holes rarely occur with this design, but it requires a large and very plastic bott having a low dry-strength.

A simple way to make a tap-hole is to use a brick having the shape of the first tap-hole described. Alternatively, a double-holed brick such as that shown in Fig. 8 can be used. This is probably the best

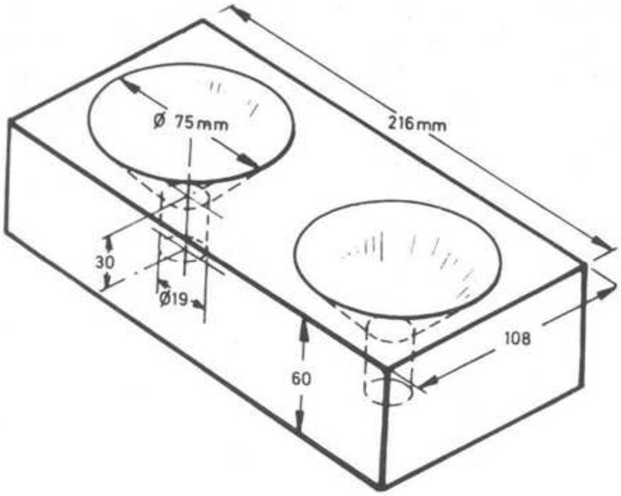

Fig. 8. Double-holed brick suitable for general use. (Front cone angle 80–90°).

design for general use, and has the advantage of requiring no drying; it also heats up rapidly at the start of the melt. The upper hole is also available for use in an emergency.

Casting defects from botting and clay mixtures It is not appreciated by many foundrymen that slag inclusions in castings can result from the use of botts made from clay of low refractoriness, obtained cheaply from local deposits. These arise from an accumulation of fused dross in the ladles and launders, as practically the whole of the clay bott is swept into the ladle every time the cupola is tapped.

The choice of the bott mixture requires care if such defects are to be avoided, and also if a workable material is to be obtained. The mixture must thus meet the following requirements:

The residue should be easily skimmed off the metal surface.

The mixture should not melt on the surface of the molten metal to give a fluid dross.

It should not react with the ladle refractory lining or the metal, or become entrained in the metal to give dross inclusions.

It must be capable of easy moulding by hand into the best shape found by practice to give easy and safe stopping of the metal stream.

It must adhere firmly to the bott stick during application, and leave the stick cleanly by a slight twist.

It must not be so wet as to cause spluttering of the metal.

It must remain firmly in the tap-hole until the next tap is required.

It must be capable of being easily removed.

It must leave a clean hole that does not tend to slag up or erode.

It must not shrink unduly on drying out, and should be permeable to allow escape of steam on drying.

To meet these requirements the mixture must be based on a material having a high fusion temperature, a medium dry-strength, and a low drying contraction. The effect of including red sand or a coaldust of high ash content is to lower the fusion temperature. If, however, the sum of these effects does not reduce the fusion temperature of the mixture to below 1 400°C, there should be no dross or slag defects in the castings arising from the botting clay.

Two recommended mixtures for botting clays, which have been found by experience to be satisfactory in meeting all these requirements, are given below:

Type A – For small botts and frequent tapping –

Fireclay (30-50 per cent on clay grade) 70-88 per cent.

Coaldust (11 per cent ash max., superfine grade) 10-20 per cent, or Sawdust (medium-fine sawings) 2-10 per cent.

Type B – For long collection periods and a large tap-hole –

Fireclay (30-50 per cent on clay grade) 50 per cent.

Coaldust (11 per cent ash max., fine grade) 10-20 per cent, or sawdust (medium-to-coarse sawings) 2-10 per cent.

Black sand (burnt core sand) 20-38 per cent.

Air leaks

The importance of supplying the correct amount of air to the cupola has been discussed elsewhere. Many problems submitted to BCIRA arise from an insufficient amount of air entering the cupola, and this is not always due to the installation of blowing equipment of inadequate capacity. One reason is blockage of the fan inlet with rubbish, but the most common reason is air leaks at the tuyere cover plates, the wind belt and the blast main. There is little doubt that the operating efficiency of most cupolas could be improved by elimination of air leaks at these positions.

Factors Affecting Cupola Performance and their Control

A cupola should provide molten metal at a required rate, at a temperature suitable for pouring sound castings, and the metal should be of a desired composition. These requirements should be realized at the most economical cost.

The factors affecting the performance of the furnace with respect to these requirements and their control are discussed.

Melting rate

The melting rate depends upon the ratio of coke to metal in the charges, and how quickly the coke is burnt. The rate at which the coke is burnt is determined by the blast rate. The relation between the blast rate, the charge-coke quantity and the melting rate may be derived as follows:

If Q = blast rate, m^3/min at s.t.p. (0^oC, 101.3 kPa)

M = quantity of carbon burnt, kg/min

L = quantity of air consumed, m^3 (at s.t.p.) per kg of carbon burnt

C = quantity of carbon burnt, kg per 100 kg of iron melted

S = melting rate, t/h

Then $Q = L \times M$

and $M = \dfrac{S \times 1000}{60} \times \dfrac{C}{100} = \dfrac{S \times C}{6}$

$Q = \dfrac{L \times S \times C}{6}$ equation (1)

In Imperial Units:

If Q = blast rate, ft^3/min (60^oF, 14.7 lbf/in^2 absolute pressure)

M = quantity of carbon burnt (lb) per minute

L = quantity of air consumed per 1 lb of carbon burnt

C = quantity of carbon burnt (lb) per 100 lb iron melted

S = melting rate, t/h

M = quantity oı carbon burnt, kg min.

and $M = \dfrac{S \times 2\,240}{60} \times \dfrac{C}{100}$

therefore: $Q = \dfrac{2\,240}{60} \times S \times \dfrac{C}{100} \times L$

or $Q = 0.373 \times S \times C \times L$ equation (1)

The quantity of air consumed per kg (lb) carbon (L) depends upon the completeness of combustion, i.e. upon the relative proportions of carbon dioxide and carbon monoxide in the gases leaving the furnace. If the carbon should burn to CO_2 only, then 8.93m^3 (151 ft^3) of air would be required to burn 1kg (lb). If it should burn to CO only, then only half of this quantity of air, i.e. 4.47m^3 (75.5 ft^3) would be required. However, in the cupola the coke carbon never burns only to CO_2 or CO, but to a mixture of these gases. The ratio of CO_2:CO in the gases and, therefore, the amount of air needed to burn the carbon is dependent upon a number of factors, but mainly on the ratio of carbon burnt to iron melted (i.e. the charge coke:metal ratio). Typical values for the amount of air required to burn lkg (lb) carbon in terms of the amount of carbon burnt per 100kg (100lb) iron melted are given in Fig. 1.

With the aid of Fig. 1, equation 1 can be used to calculate the blast rate to give a required melting rate as follows:

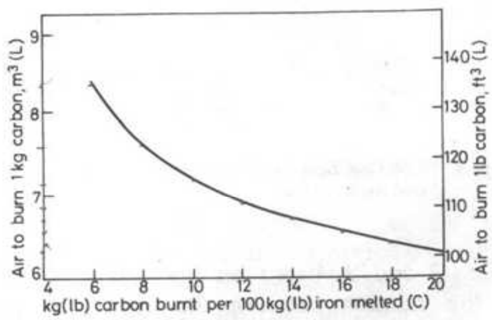

Fig. 1. Air required to burn unit quantity of carbon in the cupola (typical values).

Use of melting-rate formula

Calculate blast rate to give melting rate of 10 t/h.

given coke charge = 15%

carbon content of coke = 91%

carbon pick-up by metal = 0.4%

Step 1: Evaluate 'C'

C = carbon burnt kg (lb) per 100kg (100lb) iron melted;

$= 15 \times \dfrac{91}{100} - 0.4 = 13.25$

Step 2: Evaluate 'L'

From Fig. 1, when C = 13.25, L = 3.1m^3 (109 ft^3).

Step 3: Use melting-rate Formula 1 to calculate required blast rate:

$Q = \dfrac{L \times S \times C}{6} = \dfrac{6.8 \times 10 \times 13.25}{6}$

$= 150\ m^3/min.$

$Q = 0.373 \times S \times C \times L$

$= 0.373 \times 10 \times 13.25 \times 109$

$= 5\ 390\ ft^3/min$

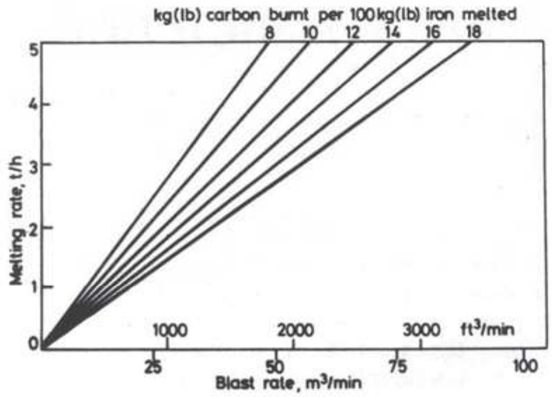

Fig. 2. Relation between blast rate, carbon:iron ratio and melting rate.

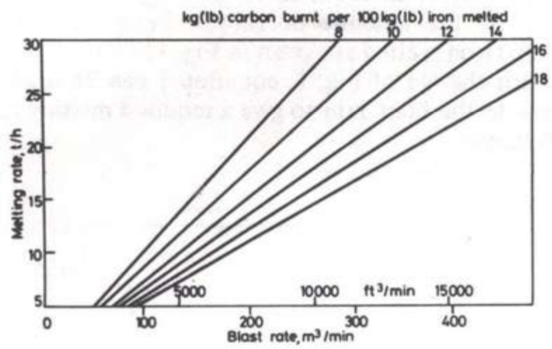

Fig. 3. Relation between blast rate, carbon:iron ratio and melting rate.

Figs. 2 and 3, based on Formula 1 and Fig. 1, give the relation between the blast rate, melting rate and quantity of carbon burnt per 100kg (lb) of iron melted, for melting rates below and above 5 t/h respectively.

Metal temperature

It has been shown how the melting rate is dependent upon the blast rate and the coke charge (or, more correctly, on the amount of carbon burnt per 100kg (lb) of iron melted). However, when operating a cupola, metal is required not only at a definite rate but also at a temperature suitable for the production of sound castings. It is obviously desirable to correlate these factors (i.e. blast rate and coke charge) with the metal temperature.

For any given cupola, the relations between the blast rate, coke charge, the melting rate and the metal temperature can be expressed in the form of a cupola behaviour or 'net' diagram. Such a diagram obtained for a 76cm diameter cold-blast cupola[1] is shown in Fig. 4.

This diagram shows:

1. At a fixed charge coke:metal ratio (or more correctly, C-burnt:iron ratio), increasing the blast rate increases both the melting rate and the metal temperature until an optimum value is reached, whereafter further increase in the blast rate causes the metal temperature to fall.

2. At constant blast rate, increasing the coke charge reduces the melting rate and increases the metal temperature.

3. To increase the metal temperature but maintain a constant melting rate, both coke charge and blast rate must be increased together.

It should be noted that at each charge coke quantity, there is an optimum blast rate at which a maximum metal temperature is obtained. The value of the optimum blast rate varies to some extent with the charge coke quantity but is approximately 100-120 m^3/m^2min (350-400 ft^3/ft^2min). The curves in the region of the optimum are flat and, for design purposes, a blast rate of $115m^3/m^2min$ ($375ft^3/ft^2min$) may be taken as a close approximation to the optimum over the entire range of charge coke quantities.

The net diagram in Fig. 4 is only strictly valid in a quantitative sense for the cupola for which it was obtained. It should not be used to predict metal temperature for any other cupola, since this will depend upon such design factors as cupola diameter, well depth, shaft height, etc. and on operational factors such as the nature of the charge materials used and the size and quality of the coke. The quantity of coke used to provide a required metal tapping temperature must be decided on the basis of experience. However, a knowledge of the cupola behaviour diagram provides an immediate indication of the direction in which control should be exercised to alter operational conditions – to achieve the desired effect on metal temperature or melting rate or both.

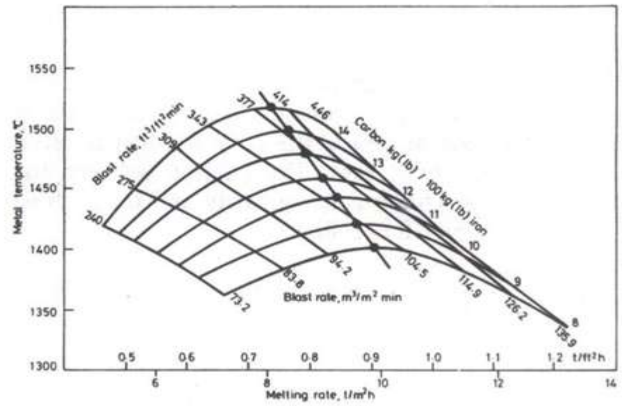

Fig. 4. Cupola net diagram (Patterson Siepman Pacyna) diameter 76 cm (30 in.).

Metal composition

The factors affecting the melting rate and the metal temperature have now been considered, but the cupola is also required to provide metal of desired composition. The composition of the metal produced from the cupola is determined predominantly by the composition of the charge mixture. Charge mixture control and the selection of materials are considered elsewhere in the booklet.

All that need be emphasized at this stage, therefore, is that carbon pick-up and melting losses of silicon and manganese are dependent on the temperature of the metal tapped and, therefore, the closer the control of this, the more consistent will be the quality of the iron produced.

Control

It has been shown that cupola performance is affected by the blast rate and the coke:metal ratio, and the importance of charge mixture selection and control has been mentioned. It follows, therefore, that for consistent control of cupola operation, it is necessary:

1. To control the blast rate to one row of tuyeres, in only one row is used, or to both rows of tuyeres if two rows are used.

2. To control the weighing of the metallic charge ingredients.

3. To control the coke:metal ratio.

Blast rate

Many cupolas are provided with a guage which indicates the pressure of air in the windbelt, and the blast supply is regulated according to the reading obtained. However, the pressure of air in the windbelt is only an indication of the head required to drive a certain volume of air into the cupola; it gives no information concerning the rate at which air is supplied.

The relation between the blast rate and the blast pressure not only varies widely for different cupolas, but can also vary considerably during the operation of any particular furnace. For example, for a constant rate of air supply, the pressure increases if the tuyeres slag-over very badly, or if the packing density of the charges increases. In practice, increased blast pressure due to these causes is usually accompanied by a reduction in the rate of air supplied by the fan, but the furnaceman often mistakenly assumes that the increase in the blast pressure is indicative of an increased blast rate. He then closes the blast control valve to restore the blast pressure to its initial value, thus reducing the blast rate, and so helps to drive the furnace further out of control.

On the other hand, the pressure falls if charges 'scaffold' or if less densely packed charge materials are used. In this case, the fall in pressure is usually accompanied by an increase in the blast rate. The natural tendency is for the furnaceman to open the blast control valve, again to the detriment of good control.

The blast rate can only be accurately controlled by the provision of an air-flow meter. This is usually based on the use of an orifice plate or venturi tube, inserted in the blast main or in the inlet ducting to the fan. The rate of air flow is proportional to the differential head across the measuring element, and is usually recorded on a chart calibrated in terms of the flow rate (m^3/min, ft^3/min). With the aid of such an instrument the furnaceman can, by suitable adjustment of the blast control valve, maintain the blast rate constant at the level necessary to given the required melting rate and metal temperature.

Instruments and control gear are available to control the blast rate automatically, and these are particularly valuable on longer melts and where a high degree of reproducibility of melting conditions is required from day to day. Their use is also strongly recommended for cupolas designed or converted for operation with two rows of tuyeres with the blast divided equally between them.

Weighing

The most important factors governing the composition of the metal tapped from a cupola are the composition and relative amounts of the individual components of the metal mixture charged. To obtain consistency of metal composition at the spout it is essential, therefore, that each component should be carefully weighed. Ferro-alloy additions should be weighed on separate, accurate scales with a range appropriate to the quantities used.

The metal temperature, the melting rate and, to a certain extent, the metal composition are dependent upon the coke-to-metal ratio. It is, therefore, also important for good operational control to weigh the coke charges.

Finally, even if the blast rate is controlled and the charge materials are accurately weighed, the metal composition can still vary widely at the spout, particularly when melting high-steel charges and when considerable quantities of ferro-alloys are used in the charges. An essential element in control of metal composition is, therefore, the provision of adequate mixing capacity in the cupola well of intermittently tapped cupolas, or in the receiver for continuously tapped cupolas.

REFERENCE

1. Patterson (W), Siepmann (H) & Pacyna (H): Giesserei Tech. Wiss. Beihefte, 1961, v.13, Oct., pp 239 - 252.
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Development of the Divided-Blast Cupola

Development work

The divided-blast cupola was developed after an investigation carried out a BCIRA showed that the efficiency of a cupola could be improved by providing it with two rows of tuyeres and correctly proportioning the blast supply between them. The divided-bast cupola is illustrated schematically in Fig. 1.

The cupola used for the development work was lined to an internal diameter of 76cm. It was equipped with two rows of tuyeres, each of which was supplied with a measured and controlled quantity of air. For this purpose two wind-belts were provided, each with its own separate blast main in which the blast rate was measured and controlled automatically (Figs. 2 & 3).

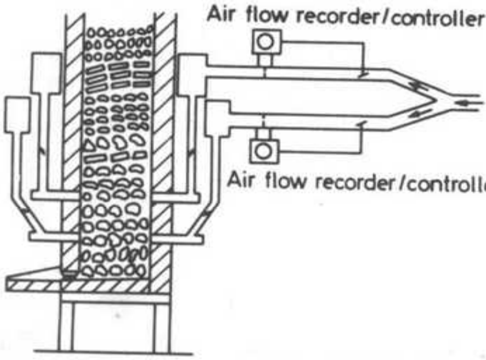


Fig. 1. The divided-blast cupola.

The results of tests to determine the effect of varying the blast distribution to the two rows of tuyeres are shown in Fig. 4. For this purpose, three different sets of operating conditions were used. In each case, as the proportion of the blast supplied to the upper tuyeres was increased, the metal temperature increased until the quantity of blast supplied to the upper tuyeres attained 50 per cent of the total blast rate. Further increasing the amount of blast supplied to the upper tuyeres resulted in a decrease in the temperature. This showed that optimum results were obtained with the blast air equally divided between the rows of tuyeres.

Fig. 5 shows the effect of varying the distance between the two rows of tuyeres on the metal temperature and the melting rate at various coke charge quantities. The blast rate was constant at 45m^3/min. (1 600 ft^3/min). When two rows of tuyeres were used the blast was equally divided between them. At each coke charge the maximum metal temperature was obtained when the distance between the two rows of tuyeres was about 1 m. Compared with operation with only one row of tuyeres, the use of two rows at optimum spacing resulted in an increase of 45-50 °C in the tapping temperature of the metal, and only a very slight reduction in the melting rate.

Fig. 6 shows the relation between the charge coke quantity and the metal temperature for operation with one row of tuyeres and two rows of tuyeres,

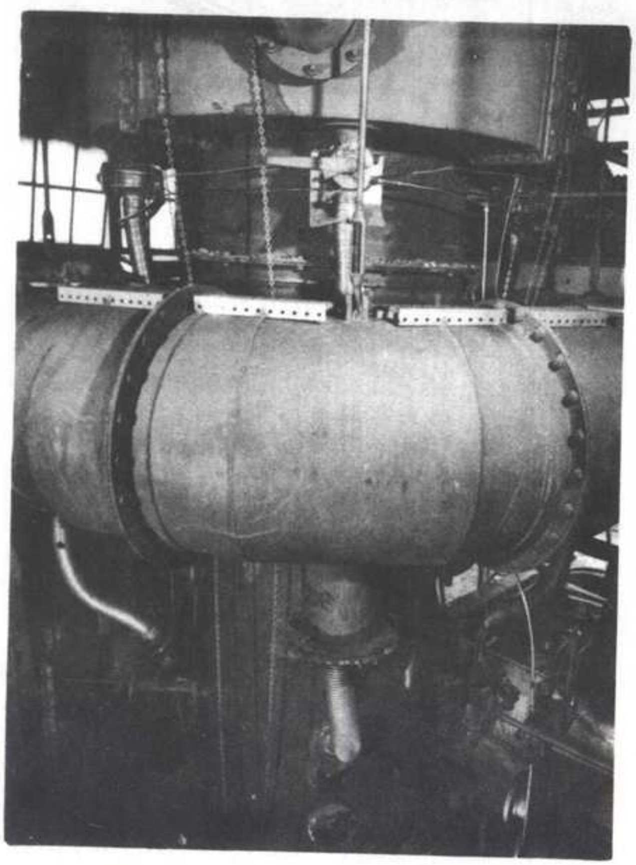


Fig. 2. View of cupola showing upper and lower wind-belt with shut-off valves in down comer pipes from upper wind-belt.

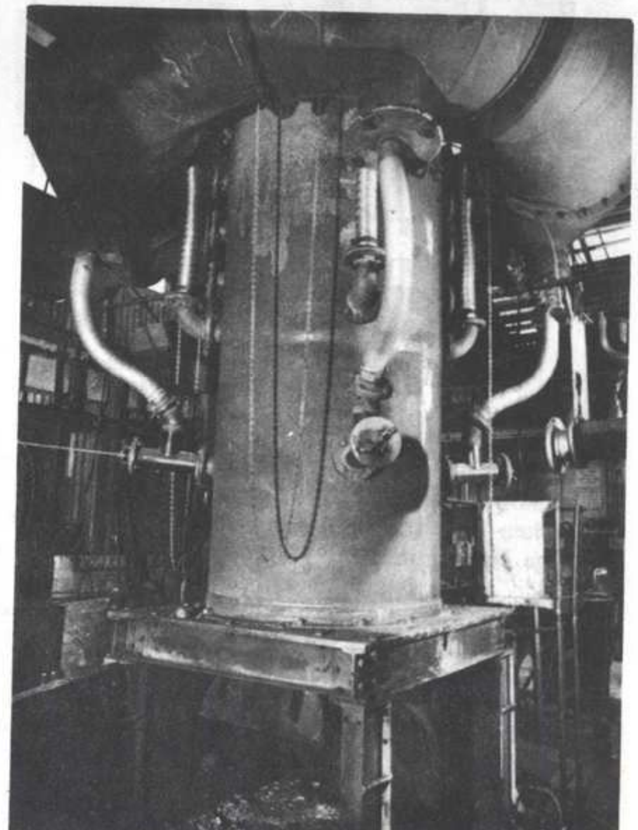
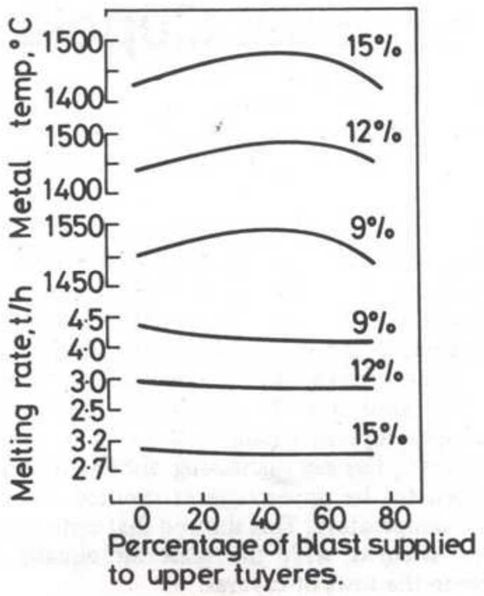


Fig. 3. View of cupola showing lower and upper tuyeres and alternative positions of upper tuyeres.



Coke 9%, blast rate 45m^3/min. (1600ft^3/min.) tuyeres 76cm (30in.) apart
Coke 12%, blast rate 37m^3/min. (1300ft^3/min.) tuyeres 46cm (18in.) apart
Coke 15%, blast rate 45m^3/min. (1600ft^3/min.) tuyeres 137cm (54in.) apart

Fig. 4. Effect of blast distribution to two rows of tuyeres on furnace performances.

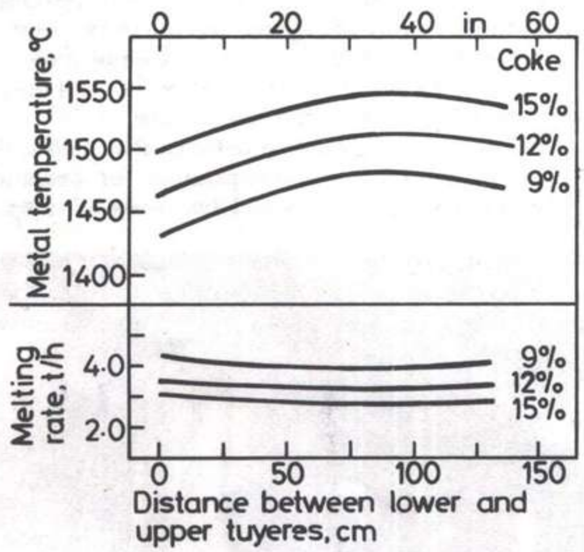


Fig. 5. Effect of tuyere spacing of metal temperature and melting rate for various charge coke quantities.

having optimum spacing between the rows and with the blast equally divided between the rows.

When operating with one row of tuyeres, a coke charge consumption of 15 per cent was required to provide a tapping temperature of 1 500 °C. The melting rate obtained at a blast rate of 45m^3/min (1 600 ft^3/min) was 3.05 t/h.

When operating with two rows of tuyeres, a temperature of 1 500 °C was obtained at a charge coke consumption of 10.8 per cent. At a blast rate of 45m^3/min (1 600 ft^3/min), the melting rate obtained was 3.63 t/h.

Thus, operation with two rows of tuyeres could allow charge coke consumption to be reduced by 28 per cent and the melting rate to be increased by 19 per cent compared with operation with one row of tuyeres, while permitting a tapping temperature of

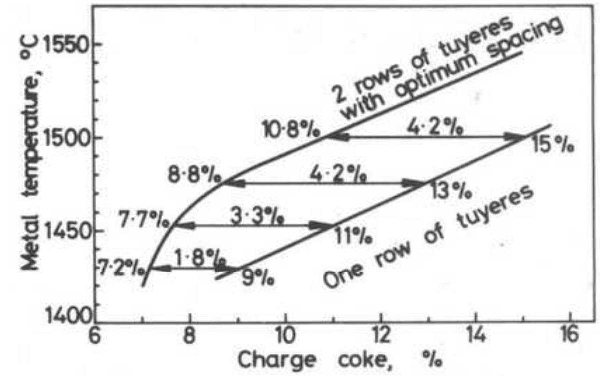


Fig. 6. Coke savings possible by operating with two rows of tuyeres with correctly proportioned blast supply.

1 500°C to be obtained.

The actual coke savings and increases in melting rate are shown in Fig. 7.

Similarly, 13% charge coke could be reduced to 8.8% with divided-blast operation;
11% charge coke could be reduced to 7.7% with divided-blast operation;
9% charge coke could be reduced to 7.2% with divided-blast operation;

to provide equivalent levels of metal temperature. The corresponding increases in the melting rate were 23%, 20% and 11%.

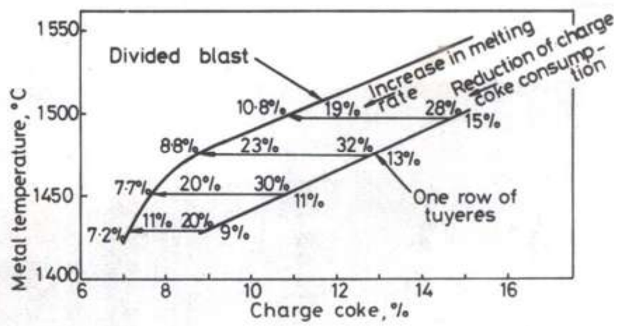


Fig. 7. How metal temperature can be increased or how coke consumption may be reduced and melting rate increased by divided-blast operation.

When operating with divided-blast without reducing the coke charge, the metal temperature was increased by approximately 45-50 °C, the pick-up of carbon increased by about 0.2 per cent, and there was little or no change in the melting loss of silicon. When the coke charge was reduced so that a metal temperature similar to that obtained with one row of tuyeres was maintained, carbon pick-up increased slightly – by approximately 0.06 per cent – and the melting loss of silicon increased by approximately 0.18 per cent.

With divided-blast operation, the lining burn-out extended further upwards in the furnace but was not so deep as when operating with one row of tuyeres (Fig. 8). This was a clear indication that the use of two rows of tuyeres extended the operating height of the coke bed, and this is believed to be the major reason for the improved performance rather than any substantial combustion of carbon monoxide by the

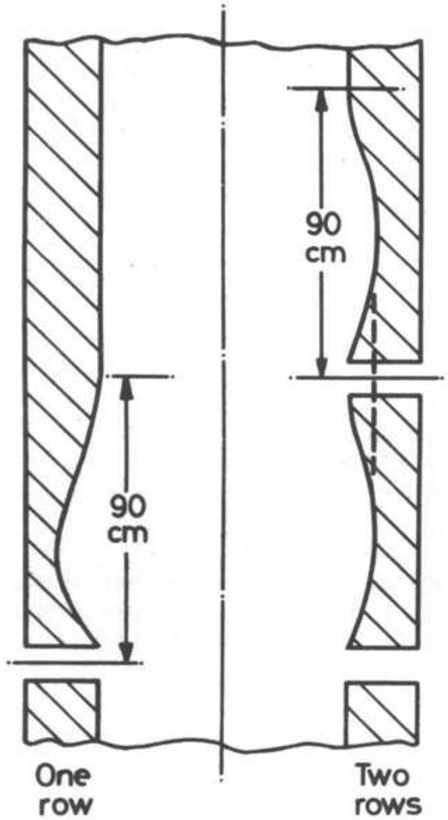


Fig. 8. Pattern of lining burn-out when operating with one, and two, rows of tuyeres.

air entering the upper row of tuyeres. In fact, inspection through the sight glasses of the upper tuyeres confirms that melting occurs above those tuyeres and that the coke bed height has been increased. At the commencement of a melt it is necessary, therefore, to measure and adjust the coke bed to a given height above the top row of tuyeres. Since the upper tuyeres are 1 m above the lower tuyeres, which would normally be the conventional tuyeres, the total coke-bed height above the tap-hole will be increased by a metre upon conversion to divided-blast operation.

Industrial application

Almost a hundred cupola plants are now operating in the UK with the divided-blast system, and many others are being installed throughout the world. These cupolas range in size from 73 to 229 cm internal diameter, and melt from 2.5 t/h to 40 t/h. In practice, it is usual to utilize the existing blowing equipment and to modify the blast mains and wind-belts as shown in Fig. 9.

In general, the results obtained from industrial cupolas have confirmed those obtained at BCIRA from experimental work. Foundries have reported that when advantage has been taken of divided-blast operation to reduce charge coke consumption, this has amounted to between 20 and 35 per cent with, in some cases, an accompanying increase in tapping temperature.

When divided-blast has been introduced without reducing charge-coke consumption, increases in tapping temperatures of between 40 and 80 °C have been reported, although in some cases other altered features of design and operating practice accompany-

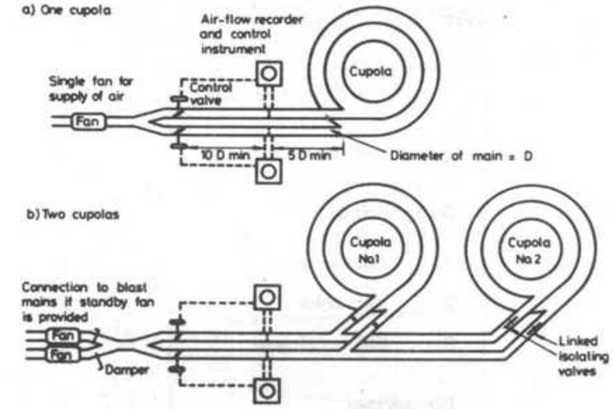


Fig. 9. Layout of blowers, mains and control instruments for operation with divided-blast.

ing the change-over to divided-blast may have contributed to the improvement in performance. The increase in carbon pick-up accompanying the increased tapping temperature has enabled some foundries to reduce their metallic charge cost. For example, one foundry has found it possible to replace 10 per cent of pig iron with steel scrap, with also a small saving of charge coke.

Many foundries have taken advantage of the increased melting rate made possible by the conversion of the cupolas. The reduced coke charge has enabled output rates to be increased by up to 25 per cent without increasing the normal blast rate.

The overall savings that can be achieved by the use of divided-blast operation can be substantial with medium to large cupolas, particularly with extended melting campaigns. A considerable financial saving is achieved by the reduced charge coke consumption, but this is slightly reduced by the additional bed-coke requirement and the increased melting loss of silicon.

At one foundry melting 96 tonnes of metal per day over an eight hour shift the costs involved were as follows:

1. Charge coke saving
(13.1% to 8.9%) = £4.03 t/day
coke valued at £69/t = £278/day
(£66 720/year)

2. Extra coke required by bed = 0.64 t/day
= £44/day
(£10 560/year)

3. Extra 0.2% silicon in charge = 0.192 t/day
45/50% FeSi valued
at £230/t = £71/day
(£17 040/year)

Net saving per day = 278 – (44 + 71)
= £163

Savings per year = £39 120

TABLE 1 SAVINGS FROM DIVIDED-BLAST OPERATION

Cupola output, t/h:		12		3	
Melting period, h		8	2	8	2
Daily savings					
1. Charge coke	£	+ 278	+ 69	+ 69	+ 17
2. Bed coke	£	− 44	− 44	− 11	− 11
3. Silicon charge	£	− 71	− 18	− 18	− 5
Net savings	£	163	7	40	1
Annual net savings	£	39 100	1 700	9 600	240

The cost of converting these cupolas to divided-blast was approximately £14 000 and, since there are no recurring fuel, oxygen or other charges, this is obviously a profitable conversion.

The change in overall savings with the length of the melting campaign and the output of the cupola are shown in Table 1. It can be seen that if the above cupola was operated for only two hours a day instead of eight, the annual savings would be reduced from £39 100 to only £1 700. On the other hand, if the cupola output was reduced from 12 to 3 t/h while still retaining an eight-hour melting period, savings would be reduced from £39 100 to £9 600.

The costs given in Table 1 assume that the charge level of silicon has to be increased by 0.2 per cent to compensate for the higher silicon loss with divided-blast. In practice, it has been found that this is the maximum additional loss of silicon, and many foundries have found no need to increase the charged silicon levels. Under these circumstances the savings resulting from divided-blast operation would be much greater than those shown in Table 1.

Divided hot-blast cupolas

Following the successful development of the divided-blast cupola operated with cold blast, many foundries have shown interest in the application of divided-blast to hot-blast cupolas, particularly as a means of increasing the output of existing furnaces being already operated to the limit of the blast-heating equipment.

Unfortunately, the conversion of many existing hot-blast plants to divided-blast operation presents major design and constructional problems. An essential requirement of divided-blast operation, if optimum results are to be consistently obtained, is that the flow of blast to each row of tuyeres should be measured and controlled. At many hot-blast plants the blast-heating equipment is located too near the cupolas to allow the hot-blast outlet to be divided between two separate mains. Furthermore, the mains and the wind-belts must be larger than those for cold-blast practice, owing to the need for thermal insulation, so problems arise in accommodating them within the space usually available. Additionally, to measure and control the supply of blast in each main the measuring elements (orifice plates or venturis) and the blast control valves would have to be located on the hot-air side of the blast-heating system.

It was felt that, at plants where such difficulties arise, the cupolas could be more readily converted and problems of blast rate measurement and control simplified by supplying the upper row of tuyeres with cold-blast. To determine if normal hot-blast performance could be equalled or improved and the melting rate increased by suppling cold-blast to the upper row of tuyeres, partially to replace or supplement the supply of hot-blast to the lower row of tuyeres, a series of melts was carried out at the BCIRA cupola plant.

The first three melts shown in Fig. 10 were carried out to determine the effects of varying the distribution of the blast to the lower and the upper tuyeres, the total blast rate being maintained at a constant value of $42.5 m^3/min$ ($1500 ft^3/min$). The coke charge in all three melts was 11 per cent by weight of the metal.

In the first melt the total blast quantity, at a temperature of 500 °C, was supplied to the lower row of tuyeres. In the second melt, 75 per cent of the total blast quantity at a temperature of 500 °C was supplied to the lower tuyeres and 25 per cent, as cold-blast, to the upper tuyeres. In the third melt, the blast was divided equally between the lower and upper rows of tuyeres.

By reducing the quantity of hot-blast to the lower tuyeres by 25 per cent and replacing it with a corresponding quantity of cold-blast to the upper tuyeres, the tapping temperature of the metal increased by approximately 20 °C but the melting rate decreased from approximately 4.35 t/h to 4.02 t/h. The increased tapping temperature was accompanied by a slight increase in the carbon and silicon contents of the metal.

When the quantity of hot-blast supplied to the lower tuyeres was further reduced, to 50 per cent of the total blast quantity, and that of cold-blast to the upper tuyeres increased to 50 per cent, the tapping temperature of the metal decreased but was still maintained at the level obtained when the same total quantity of blast was supplied at a temperature of 500 °C to the lower tuyeres alone. There was little further change in the melting rate or the combustion ratio. The carbon and silicon contents decreased slightly, to approximately the same levels as were obtained with normal hot-blast operation.

The results of the first three melts showed that when 75 per cent of the total blast quantity was supplied as hot-blast to the lower tuyeres and 25 per cent as cold-blast to the upper tuyeres, the tapping temperature of the metal was approximately 20 °C

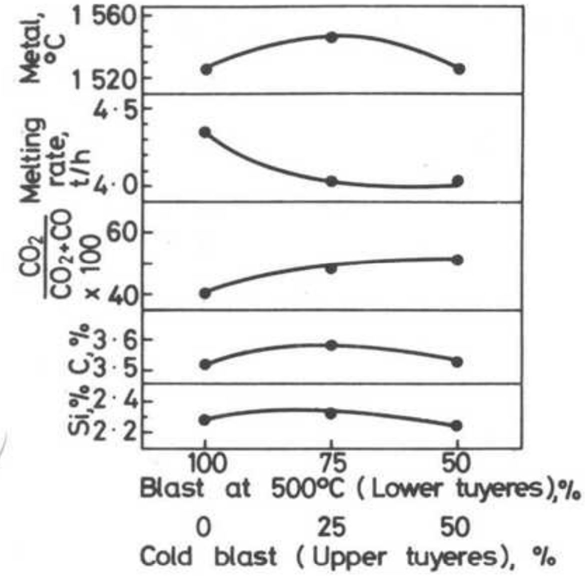


Fig.10. Effect of varying the distribution of hot and cold air to lower and upper tuyeres. (Coke charge 11%, total blast rate, 42.5 m^3/min) (1 500 ft^3/min.).

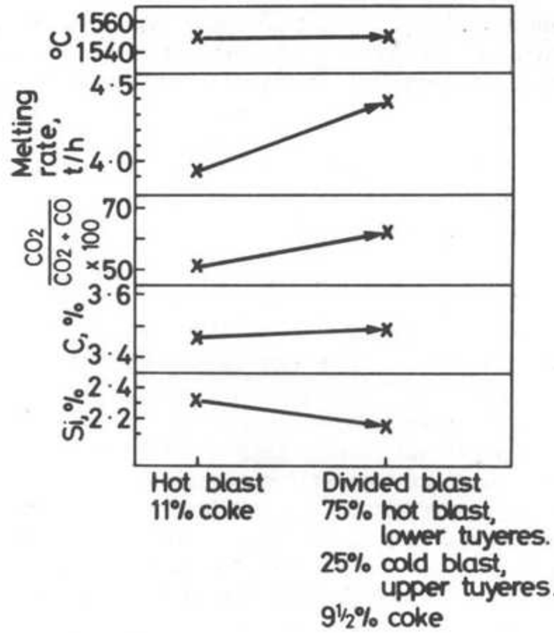


Fig.11. Comparison of cupola performance when operated with 1: hot blast, 500°C, 11% coke, blast rate, 42.5 m^3/min (1 500 ft^3/min) and 2: divided-blast, 9½% coke, total blast rate 42.5 m^3/min (1 500 ft^3/min).

higher than that obtained when the entire blast quantity was supplied to the lower tuyeres. From the knows characteristics of the BCIRA cupola, it was estimated that it should be possible to reduce the charge coke quantity by 1½ per cent by weight of the metal, i.e. from 11 per cent to 9½ per cent, when using divided-blast in the proportions of 75 per cent and 25 per cent to the lower and upper tuyeres respectively, and yet maintain the same metal temperature as that obtained with normal hot-blast operation. At the same time, at constant blast rate, the reduction in the coke charge should provide an increased melting rate. Two further melts were therefore carried out to verify these suppositions.

One melt was carried out using a coke charge of 11 per cent by weight of the metal, the blast rate was 42.5m^3/min (1 500 ft^3/min) and the entire blast quantity, preheated to 500 °C, was supplied to the lower row of tuyeres. In the other melt the coke charge was 9½ per cent by weight of the metal; the total blast rate was 42.5m^3/min (1 500 ft^3/min) of which 75 per cent was supplied as hot-blast at 500 °C to the lower tuyeres and 25 per cent as cold-blast to the upper tuyeres.

Fig. 11 shows that the tapping temperature in both melts was identical. The melting rate increased from 3.93 t/h when using normal hot-blast with 11 per cent coke, to 4.38 t/h when using divided-blast with 9½ per cent coke.

In the hot-blast melts discussed so far the total blast supply rate was maintained at a constant value. Under these circumstances it was found that the maximum tapping temperature was obtained when the blast was divided in the proportions of 75 per cent (hot-blast) to the lower tuyeres and 25 per cent (cold-blast) to the upper tuyeres, and the melting rate could be increased slightly by reducing coke consumption. However, in practice, where an increased rate of output is required from an existing hot-blast cupola, cold blast could be used, not to replace part of the hot-blast supply, but to supplement it. To demonstrate how the melting rate could be increased in this way, a further melt was carried out. The operating conditions were identical to those used in the melt when the cupola was operated with hot-blast in the normal way with one row of tuyeres, except that an additional quantity of air, in the form of cold-blast, equivalent to 25 per cent of the total blast quantity, was supplied to the upper row of tuyeres.

The results were:

1. The melting rate increased by 22 per cent, from 4.35 to 5.30 t/h.
2. The tapping temperature of the metal increased from 1 525 °C to 1 565 °C.
3. The combustion ratio, or efficiency, and the stack-gas temperature both increased.
4. The metal composition was not significantly affected except for a slight increase of the silicon content.

In cold-blast cupola practice, optimum results were obtained with the divided-blast cupola when the blast was divided equally between the two rows of tuyeres. When, in hot-blast cupola practice, the quantity of hot-blast supplied to the lower row of tuyeres is reduced and replaced by an equal quantity of cold-blast to the upper row of tuyeres, the sensible-heat content of the total blast supply is reduced. Consequently, the improvement in performance from using two rows of tuyeres in this manner cannot be expected to be as great as that obtained in cold-blast practice, or as that which might be anticipated if hot-blast was supplied to both rows of tuyeres.

Several foundries in the UK have converted hot-blast cupolas to divided-blast operation by supplying cold air to the top row of tuyeres and, although some savings in charge coke consumption have been obtained, the greatest benefit has been derived from the increased output resulting from the conversion.

At present only one cupola has been installed with a hot-blast supply to both rows of tuyeres. The cupola, at Dalton Foundry in the USA, has operated at a blast temperature of 450-500 °C and optimum conditions have been established with the blast divided, 75 per cent to the lower row and 25 per cent to the upper row of tuyeres. Coke consumption has decreased by 22 per cent, with no reduction in metal tapping temperature, and the melting rate has increased from 26 to 36 t/h.

Modified and Special Cupola Operating Techniques

The following aspects of modifed and special cupola operation are discussed:

1. Basic cupola operation
2. Water-cooling
3. Hot-blast
4. Supplementary fuels in the cupola
5. The cokeless cupola
6. The use of calcium carbide

1. Basic cupola operation

A limitation of acid-cupola melting is that it is not possible to reduce the sulphur content of the metal during melting, as the metal always absorbs sulphur from the coke. To enable irons of low sulphur content to be produced (directly from the cupola) the slag must be basic, containing a high ratio of lime to silica, and this requirement prohibits the use of acid refractories for the cupola lining. When operating with a basic slag, cupolas are lined with magnesite, dolomite or carbon refractories.

The use of basic linings in cupolas creates a number of difficulties due to the high rate of wear of the material, its expense, and the general difficulties of carrying out lining repairs. The solution to these problems that has generally been adopted is to water-cool the melting zone, which may be completely unlined, or provided internally with a thin refractory lining which soon stablilizes itself against slag attack due to the action of the cooling-water.

A slag becomes basic as the sum of the lime (CaO) and magnesia (MgO) exceeds the sum of silica (SiO_2) and alumina (Al_2O_3). The basicity of a slag is given by the following ratio:

$$\frac{\mathbf{CaO\ \% + MgO\ \%}}{\mathbf{SiO_2\ \% + Al_2O_3\ \%}}$$

Slags can be classified according to the degree of basicity as follows:

Mild basicity:	1 – 2
Moderate basicity:	2 – 3
High basicity:	3

High slag basicity will give irons of high total carbon and low sulphur contents and high silicon melting loss, whereas the reverse occurs with low slag basicity.

The fluxes added to basic cupolas include limestone, dolomite, dolomitic limestone, and fluorspar. Additions of fluorspar in the range 0.5 – 3.5 per cent ensure a slag of good fluidity.

The excess lime available in a basic slag enables desulphurization to take place according to the following equation:

$$\mathbf{C + CaO + FeS > CaS + Fe + CO}$$

The objects of basic melting have primarily been concerned with obtaining irons of high carbon and low sulphur content. Often such characteristics have been required for the production of nodular (SG) iron, but plants of this type have also been used for the production of other types of iron using high percentages of steel scrap, e.g. ingot mould iron and synthetic or refined pig iron. Basic melting has also found application in instances where only a relatively mild degree of basicity has been required to limit sulphur pick-up to a slight extent, for example in foundries producing light castings in phosphoric irons. In such cases, the use of the water-cooled cupola operated with mildly basic slags has permitted the use of 100 per cent cast iron scrap charges and allowed carbon and sulphur specifications to be met.

The use of basic slags has obvious advantages, but there are also the following disadvantages:

1. Silicon losses are greater than in an acid-slag-operated cupola. The higher the slag basicity, the greater the silicon loss. The oxidation of silicon can be expressed by the following equation:

   $$\mathbf{Si_{(m)} + SiO_{2(s)} \rightleftharpoons 2FeO_{(s)} + 2Fe_{(m)}}$$

   Using an acid slag this reaction cannot proceed as far, owing to the high silica content of the slag, and this explains the lower silicon loss obtained.
2. Refractory costs are higher, particularly in the case of lined cupolas.
3. Cost of fluxing material is higher.
4. Metal analysis is more difficult to control than for acid-cupola operation. Where the production of relatively critical engineering castings is required, it is essential to provide adequate mixing facilities in order to control compositional variations. In practice, this means that the metal may be tapped from the cupola into a heated receiver, preferably of the electric induction type.

2. Water-cooling

Water-cooling of the cupola is used fundamentally as a means of minimizing the comsumption of refractories in the melting zone, and is generally adopted for the following reasons:

(a) To extend the duration of a melt.

(b) To reduce the labour and time required for repairs to the lining.

(c) To limit the consumption of expensive refractory materials, particularly basic refractories.

(d) To enable the internal diameter of the cupola to be increased by reducing the lining thickness or removing the lining completely in the melting zone, so that a higher melting-rate may be obtained.

The methods of water-cooling fall into two categories, internal cooling and external cooling. In the former method the water is circulated through a number of jackets or through banks of steel tubing set in the furnace lining in the melting zone. External cooling is accomplished by covering the outside of the cupola shell with a sheet of water, usually by means of water sprays. The different types of water-cooling are shown diagrammatically in Fig. 1. Where a spray of water is applied to the cupola shell and a refractory lining of normal thickness is used in the cupola, this type of water-cooling is normally only adopted as a precautionary measure in case the lining should burn back to the shell. This type of water-cooling is usually only applied where it is required to extend the duration of a melt, or where the refractory lining has been reduced in thickness in order to obtain a higher output and, in these circumstances, it is required to prevent burn-back to the shell.

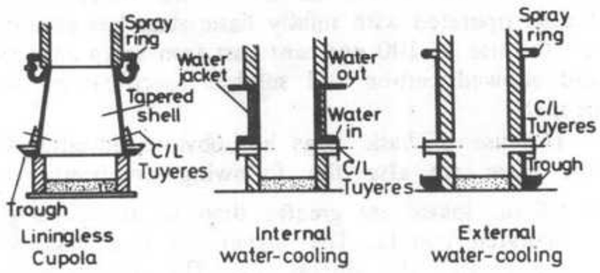


Fig. 1. Methods of water-cooling cupolas.

Operation of this type of water-cooling in conjunction with a fully lined cupola does not significantly affect metal temperature. A decided disadvantage of full water-cooling, where the cupola operates with little or no refractory lining, is the effect it has of reducing the metal temperature or, alternatively, of increasing the coke consumption to obtain a desired metal temperature. This effect becomes increasingly serious as the cupola diameter decreases. For this reason it is not generally advisable to apply full water-cooling to cupolas of less than 1 metre internal diameter, particularly if a high tapping temperature is required.

A further difficulty often experienced with the use of water-cooled and, particularly, liningless cupolas is the formation of a scull or ring of partly fused metal and slag which generally occurs above the cupola melting zone at the extremity of the water-cooled shell or cooling-tank. It is considered that this scull forms because some of the air entering the furnace does not react immediately with the coke, but passes upwards along the relatively cool surface of the water-cooled melting zone, this being the path of low resistance to air flow. Just above the top of the coolers, where there is a refractory lining in the shaft, this air becomes hot enough to ignite the coke and partly melt some of the metal. The scull is often difficult to dislodge and, in extreme cases, can continue to build up until it interferes seriously with the operation of the furnace. The difficulties encountered owing to the formation of the scull have been largely overcome by installing projecting water-cooled tuyeres with high air velocities and, in some cases, a conical-shaped shell.

Where the only benefit to be derived from the installation of water-cooling is economy in acid refractory materials and in labour costs for patching, it is unlikely that the saving will balance the relatively high cost of the water-cooling equipment, the additional maintenance it is likely to demand and the cost of the extra fuel required to compensate for the heat losses to the cooling-water. For these reasons it is unusual and generally inadvisable for water-cooling to be applied to cupolas working on short heats of less than about 8 hours unless the furnace is operated with basic slags, when metallurgical considerations override those of fuel economy and thermal efficiency. Even for longer heats, especially when high tapping temperatures are required, liningless operation is not generally advisable in acid practice and it is preferable to water-cool the cupola shell or install internal coolers using a normal lining in front of the cooled surface. The water-cooling is then purely a precautionary measure in case the lining burns back to the shell or coolers.

3. Hot-blast operation

When the cupola blast is pre-heated, certain advantages are obtained in respect of cupola operation. The effects of pre-heated blast on the metal temperature and composition when melting a given charge mixture are illustrated in Fig. 2, which summarizes the results of experiments conducted at BCIRA's cupola plant.

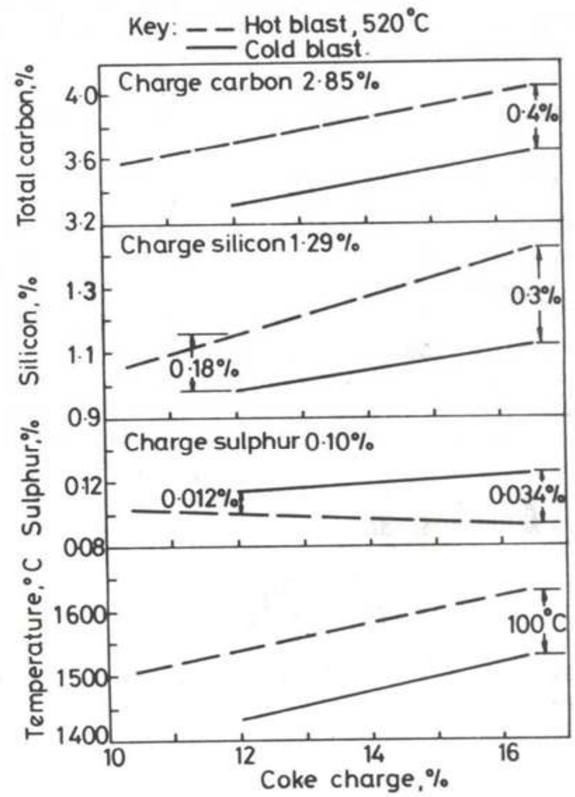


Fig. 2. Effect of hot-blast on metal temperature and composition.

At a given coke charge, heating the blast to 520 °C increased the metal temperature by 100 °C and increased the carbon content of the metal tapped by 0.4 per cent. The silicon content of the metal was increased and the sulphur pick-up reduced. With cold-blast operation, a coke charge of 16 per cent was required to obtain a metal temperature of 1 520 °C. When the blast was pre-heated to 520 °C the same metal temperature could be obtained by using a coke charge of 10.7 per cent. More important, however, is the fact that if the charge coke is maintained at that used for cold-blast operation, the effect of hot-blast in increasing the carbon pick-up enables pig iron to be replaced by steel scrap in the furnace charge. In addition, the reduced sulphur content of the metal enables a higher proportion of cast iron scrap to be incorporated in the charge.

The advantages of hot-blast may be summarized as follows:

(a) Reduced coke consumption.

(b) Increased metal temperature.

(c) Increased melting rate.

(d) Reduced sulphur pick-up.

(e) Reduced melting losses.

(f) Increased carbon pick-up and, hence, the ability to substitute steel scrap for pig iron.

It should be noted that it is not possible to obtain all these advantages at the same time. For instance, an increase in the proportion of steel scrap may require an increase in the proportion of coke for recarburization. This, in turn, is accompanied by a reduction in melting rate and an increase in the pick-up of sulphur.

The blast may be pre-heated by means of recuperative systems which extract heat from the waste gases, or by independently-fired oil or gas heaters. A blast temperature of 500 °C is the usual temperature employed in most designs of hot-blast installations. The economic benefits diminish when the blast temperature exceeds 500 °C since recuperators for operation at higher temperatures become unduly expensive. The capital cost of fully recuperative hot-blast equipment is much higher than for an independently fired heater, and can only be justified where the output is sufficiently high, probably not less that 150 t/week. Experience has indicated that with independently fired blast heaters the maintenance requirement is low, and amounts to little more than periodic cleaning of the recuperator elements, combustion chamber and burners. With fully recuperative systems, however, much more extensive maintenance is essential for satisfactory operation of the plant. An independently fired blast heater can also result in more consistent blast temperatures than a recuperative system, and this can have an important bearing on technical control and supervision and the resultant quality of the iron produced. Against the advantage of the independently fired unit must be set the extra fuel cost.

The capital cost of a hot-blast cupola plant is invariably much higher than that of a cold-blast cupola plant, owing to the cost of the recuperator or independently fired heater and the need to install more expensive fume-cleaning equipment. The capital cost of a hot-blast plant is unlikely to be justified on the basis of reduced coke consumption alone, and it only becomes economic when advantage is taken of the possibility of melting increased proportions of steel or cast iron scrap. In these circumstances, little or no economic benefit in coke consumption may be achieved.

4. Supplementary fuels in the cupola

General experience For some years, it appears, many cupolas in Russia and Eastern Europe have been operated with the use of natural gas as a supplementary fuel partially to replace the use of coke in cupolas. Although work has been undertaken in other countries, the system has not found wide application outside Eastern Europe. In coke-gas cupolas, the gas is burnt with air in combustion chambers attached to the cupola and the combustion products are introduced into the furnace at some distance, usually varying between about 1 and 2 metres above the tuyeres.

Over the past five years or so, some half-dozen cupola plants in Britain have been converted for coke-gas operation. Details of the operating results obtained at three of these plants are summarized in the table below.

In each of the three foundries, therefore, the total thermal input per tonne of iron melted was reduced and, since the cost of gas per GJ was less than that of coke, significant savings in fuel costs were obtained. At the relatively low charge-coke quantities used at the three foundries, however, the tapping temperature of the metal is relatively low, being within the range of approximately 1 380-1 420 °C.

	COKE OPERATION				COKE-GAS OPERATION				
Plant No.	Cupola dia, mm	Coke %	Coke, GJ/t	Melting rate, t/h	Coke %	Coke, GJ/t	Gas, GJ/t	Total, GJ/t	Melting rate, t/h
1	610	12.9	3.90	1.83	7.1	2.14	1.24	3.38	2.34
2	760	10.4	3.14	3.30	5.5	1.66	1.04	2.70	4.0
3	914	12.5	3.78	5.59	7.5	2.27	1.14	3.41	7.0

Research in Germany Dahlmann, Schock & Orths have reported the results of an extensive series of systematic tests carried out in Germany on a cupola of 800 mm internal diameter provided with four tuyeres and six gas-burners. Tests were first carried out operating the cupola in the conventional way with a coke charge of 15 per cent. In the following tests, the amount of charge coke was reduced incrementally and the quantity of gas increased so that the coke and gas quantities were equivalent thermally to that provided by 15 per cent coke in normal operation.

The results obtained, summarized in Fig. 3, show that as coke was replaced by its thermal equivalent of gas, the tapping temperature of the metal decreased only slightly until the coke charge was reduced below 9 per cent, and then very rapidly.

The authors stated their conclusions as follows:

1. If a temperature of 1 480 °C is considered adequate, then from an initial coke charge of 15 per cent coke consumption can be reduced by 40 per cent, to 9 per cent by weight of the metal, by replacement with natural gas without the temperature falling short of that required.

2. The average natural-gas input was equivalent to about 1.96 GJ per tonne.

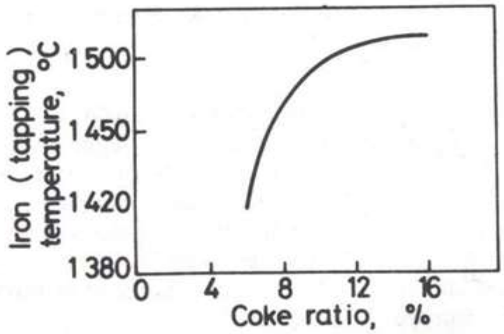


Fig. 3. **Iron (tapping) temperature as a function of coke ratio – supplementary gas firing.**

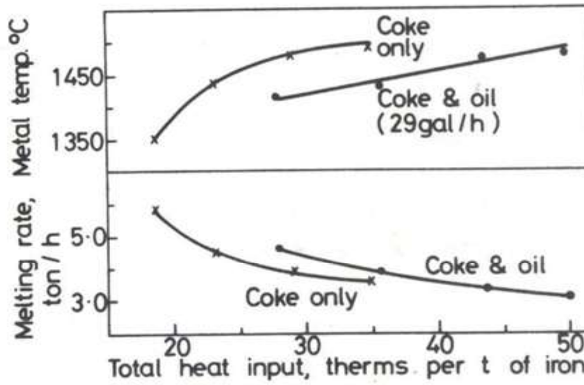


Fig. 4. **Relation between metal temperature melting rate, and total heat input for operation with and without oil.**

3. The carbon content of iron decreased from 3.6 per cent to 3.4-3.5 per cent, the silicon content from 1.83 to 1.75 per cent, and the sulphur content fell by about 0.013 per cent, with the change-over from 15 per cent coke to 9 per cent coke plus natural gas.

The results of this work show one similarity with those obtained at the three British foundries referred to earlier. At the low charge-coke quantities used at these foundries, i.e. 5.5 to 7.5 per cent, the tapping temperatures are relatively low and of the order to be anticipated from the results of the German workers. However, the gas consumption reported by the three foundries is lower than that quoted in the German work.

Work at BCIRA with oil Tests have been carried out at the 760mm-diameter cupola at BCIRA to examine the possibility of using oil as a supplementary fuel in the cupola. These tests were conducted at charge coke quantities of 6 to 12 per cent by weight of the metal, both with and without the use of oil as a supplementary fuel. The oil-firing rate was approximately 132 l/h (29 gal/h). The total air-input rate was constant at 42.8 m^3/min (s.t.p.). At a coke charge of 6 per cent the use of oil substantially increased the metal temperature, but at coke charges of 8 per cent and above, oil-firing slightly reduced the metal temperature. However, at a coke charge of 6 per cent, even with oil-firing, the tapping temperature of the metal was relatively low at about 1 415 °C. The oil consumption was equivalent to about 1.05 GJ per tonne. Thus the performance of the cupola, operated with oil, was equivalent to that of the three coke-gas cupolas already discussed. In terms of the results with oil at the BCIRA cupola plant, it is therefore not clear how the coke consumption could have been reduced at these plants without a marked fall in the tapping temperature of the metal.

In Fig. 4, the metal temperature and the melting rate are related to the total thermal input per tonne of iron melted both for operation with coke as the only fuel and for operation with coke and oil. It can be seen that the total thermal input required to produce a given metal temperature was higher for coke-oil operation. This was true even when oil was used, at a coke charge of less than 8 per cent.

Trials with gas as a supplementary fuel Trials have also been carried out by BCIRA, in collaboration with a Member foundry, without and with the use of coke-oven gas as a supplementary fuel. In this case, high tapping temperatures were required, the charges consisted predominantly of steel scrap with ferro-alloy additions, the metal produced being treated with calcium carbide and a carburizing agent in a porous-plug ladle for the production of a synthetic pig iron.

These attempts to replace part of the coke with an approximate thermal equivalent of gas were not successful, as there was a marked reduction in the temperature of the metal tapped from the cupola, a smaller pick-up of carbon, and an increased melting loss of silicon. Somewhat more encouraging results on the use of gas as a supplementary fuel have been reported from France. However, since the work on hydrocarbon fuels to supplement the use of coke on cupolas was carried out the situation concerning the availability and prices of these fuels in the UK and other countries has altered considerably. This, particularly with the development of the divided-blast cupola, has arrested interest in the possibilities of using gas or oil as a means of reducing coke consumption and costs in cupolas in the United Kingdom.

5. The cokeless cupola

The salient features of the cokeless cupola developed at the foundry of Hayes Shell-Cast Ltd is shown in Figure 5.

In this furnace the charge materials and refractory bed are supported by a water-cooled grate instead of a coke bed. The high-intensity burners can be fired by gas or light oil, but oil has been shown to produce higher metal tapping temperatures. To offset the loss of carbon and produce iron of sufficiently high carbon content, a carburizer is injected into the furnace just below the burner level but above slag level. The fuel costs for the cokeless cupola are low, but are offset by the high cost of the refractory bed and carburizing material. The fuels, gas or light oil, have low sulphur contents and result in tapped metal having a sulphur content as low as 0.02-0.03 per cent – which is advantageous as regards the production of nodular (SG) iron..

An important advantage of the cokeless cupola is the reduced emissions, and tests so far carried out show that a simple wet arrester is sufficient to reduce the emissions to a level complying with the most stringent regulations now in force.

6. The use of calcium carbide

The eutectic grade of calcium carbide, containing about 72 per cent calcium carbide and corresponding to the calcium carbide/lime eutectic, melts at about 1 630 °C and, having a lower melting-point than

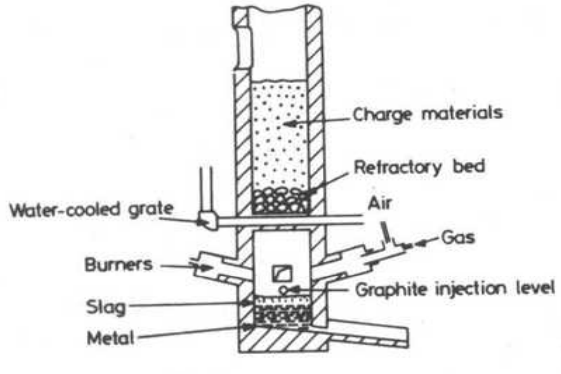


Fig. 5. Diagrammatic arrangement of gas-fired cupola.

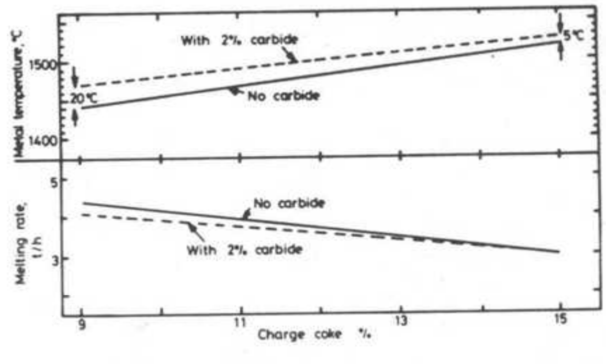


Fig. 6. Effect of 2% addition of low-melting-point calcium carbide.

commercial carbide, is the grade suitable for cupola use.

Calcium carbide may be added to cupolas as a supplement to limestone or dolomite. During combustion it combines with oxygen to form calcium oxide and carbon dioxide. The calcium oxide enters the slag and can have a desulphurizing action when the slag basicity is increased. It is claimed also that, as the reaction is strongly exothermic, increased tapping temperatures and increased carbon pick-up can be obtained. Its use has also been advocated when using high proportions of scrap, high-steel charges, or poor-quality coke, and when producing nodular (SG) iron from acid cupolas. It has also been claimed that it can be of particular benefit in providing proper slag control and better temperatures at the start of a cupola heat.

The results of an investigation carried out on the BCIRA cupola plant are shown in Fig. 6. The results show the effect of a 2 per cent addition of calcium carbide to the charges, on metal temperature and melting rate – compared with normal operation. The addition of 2 per cent of carbide increased the metal temperature, when using a coke charge of 16.5 per cent, by only 5 °C and had no significant effect on the melting rate. At a coke charge of 10 per cent the carbide addition raised the metal temperature by 20 °C and reduced the melting rate slightly, by 2.6 per cent. The addition of the carbide had no significant effect on the carbon, silicon, manganese or phosphorus contents of the iron produced; the sulphur content was reduced by approximately 0.01 per cent. On the basis of these results it was concluded that the magnitude of the benefits obtained by the use of the low-melting-point carbide was too small in relation to its cost to justify its use.

The situation at present is that some foundries claim that expenditure on calcium carbide can be justified in terms of technical and economic considerations, but others have abandoned its use after exploratory trials. Carbide can now be obtained prepacked in steel drums, and also plastic bags for direct addition to the cupola.

Oxygen in the Cupola

The beneficial thermal and metallurgical effects of using oxygen in the cupola have been known for many years, but until fairly recently it was never very widely used because the benefits could seldom justify its cost. In the few instances where it was employed, this was usually for remedial periods at the start of a melt or following stoppages for the quick attainment or recovery of metal temperature.

In recent years the prices of coke, pig iron and ferro-alloys have all increased very markedly, but that of oxygen has risen at a less rapid rate. Consequently, there has been a growing interest in its use. Whereas a few years ago the continuous use of oxygen would have been uneconomic, it can now be justified in many foundries.

Benefits of using oxygen

Oxygen may be used continuously during the melt, or just intermittently.

Continuous use. *Compared with normal operation the continuous use of oxygen:*

1. Provides a higher metal temperature, a higher carbon pick-up and a lower melting loss of silicon for the same coke consumption. The higher carbon pick-up obtained permits metallic charge costs to be reduced, since a proportion of the pig iron in the charge mix can be replaced by less expensive cast iron or steel scrap. The reduced melting loss of silicon conserves the use, and reduces the cost, of silicon additions.
2. Allows coke consumption, and therefore the fuel cost, to be reduced for a given tapping temperature but with no increase in carbon pick-up and no reduction in the melting loss of silicon.
3. Allows a higher rate of output to be obtained from a given cupola.
4. Provides a more rapid attainment of a desired tapping temperature at the start of a melt or following shut-down periods.

Intermittent use. *Oxygen may be used intermittently:*

1. To attain a required tapping temperature more rapidly at the start of a melt or following a shut-down, so saving the amount of metal pigged or otherwise reducing the incidence of casting defects caused by the use of cold metal.
2. To achieve a higher rate of output over short periods.

Methods of using oxygen

There are three ways in which oxygen may be introduced into the cupola, as shown in Fig. 1.

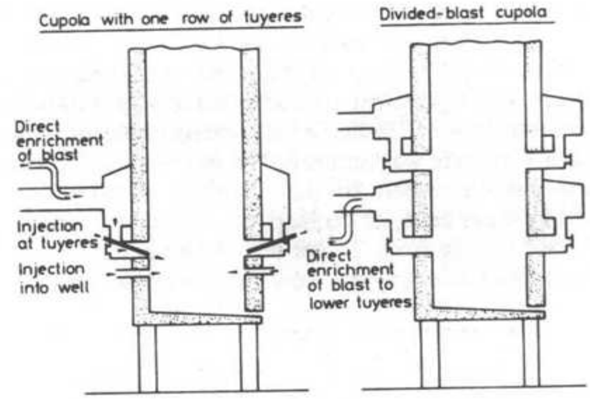


Fig. 1. Methods of inducing oxygen into the cupola

Direct enrichment of the blast. This is the simplest method, involving the least amount of special equipment and maintenance. The oxygen is fed into the blast main, where it mixes with the blast air before it enters the cupola through the tuyeres.

Injection into the well. The oxygen is injected into the coke bed beneath the tuyeres through water-cooled injectors set in the lining. The injectors are supplied from a ring main, the number of injectors depending on the furnace size. Oxygen when used in this way is considerably more effective than when it is used for direct enrichment of the blast supply.

However, the injection of oxygen into the well has been confined to continuously tapped cupolas since, with intermittent tapping, there is a risk that slag or metal may rise to the level of the injectors. Even with continuous tapping this danger is not completely eliminated.

Injection at the tuyeres. The oxygen is introduced into the cupola through injectors inserted in the tuyeres. For its effect, this method lies between direct blast enrichment and well injection. The injectors are subject to heat radiated from the coke during blast-off periods, and are generally in the form of stainless-steel tubes.

Effects of oxygen in conventional and divided-blast cupola operation.

An important development in cupola practice in recent years has been the introduction of the divided-blast cupola in which the blast is supplied to two rows of correctly spaced tuyeres and is divided about equally between them. Nearly 100 plants in the UK, and others abroad, have been converted or newly installed to operate with the divided-blast system, with significant improvement in efficiency and economy of operation.

An investigation was undertaken at BCIRA to determine the effects of using oxygen in conventional and divided-blast operation, with the principal object of evaluating the most technically and economically acceptable method of operating a cold-blast cupola of the several methods available. For this purpose oxygen was used in the following ways in both conventional and divided-blast operation.

1. To enrich the blast supply directly.
2. Injected through lances inserted in the tuyeres.

3. Injected into the well at various distances beneath the tuyeres.

Test conditions. Comparative tests were made over a range of charge-coke quantities. The tests were carried out in a 76cm internal diameter cold-blast cupola at a constant equivalent blast rate of $45m^3$/min (1 600 ft^3/min). This means that in the melts without oxygen the actual blast rate used was $45m^3$/min. In the melts in which oxygen was used the total amount of oxygen supplied in the form of blast air and additional oxygen was equivalent to that provided by a normal blast rate of $45m^3$/min. In these melts, therefore, the actual blast rate was reduced. The quantity of oxygen used was equivalent to that required to enrich the blast by 4 per cent, i.e. to increase the oxygen content of the blast air from 21 per cent to 25 per cent. The actual blast and oxygen flow rates were as follows:

quantity of oxygen required at 10.8 per cent coke would be $28.4m^3$/t, which at an assumed price of £6.00/$100m^3$ would cost £1.70. Under these circumstances, therefore, the use of oxygen would yield a net saving of £1.66/t of iron melted.

For a tapping temperature of 1 475 °C, however, injection of oxygen at the tuyeres would allow the consumption of charge coke to be reduced from 12.0 per cent to 9.3 per cent, providing a saving in coke cost of £2.16/t. The cost of oxygen (consumption $26.2m^3$/t) at £6.00/$100m^3$ would be £1.57. The use of oxygen in this case would yield a net saving of £0.59/t.

In the above examples the price of oxygen has been assumed to be £6.00/$100m^3$. The price which a foundry pays for oxygen, however, depends upon a number of factors, but mainly on the quantity

	Melts without oxygen	Melts with oxygen
Blast rate – m^3/min (ft^3/min)	45 (1 600)	36 (1 276)
O_2 in blast – m^3/min (ft^3/min)	9.5 (336)	7.6 (268)
N_2 in blast – m^3/min (ft^3/min)	36 (1 264)	28.5 (1 008)
O_2 added – m^3/min (ft^3/min)	0 0	1.9 (68)
Total O_2 supply – m^3/min (ft^3/min)	9.5 (336)	9.5 (336)
O_2 content of blast % by vol.	21	25

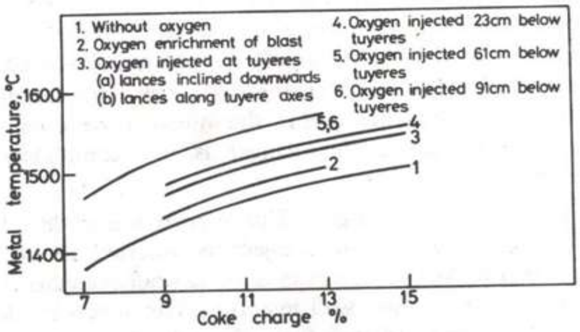


Fig. 2. Conventional operation – Relation between the coke charge and metal temperature showing effect of various methods of using oxygen

Results

Conventional operation: Fig. 2 shows the relations between the coke charge and the metal temperature for operation without and with the use of oxygen in the different ways described earlier. At any given coke charge, 4 per cent enrichment of the blast increased the metal temperature by 15 °C, injection at the tuyeres increased it by 40 °C, injection into the well 23cm below the tuyeres by 50 °C and injection 61cm and 91cm below the tuyeres by 85 °C. To enable oxygen to be injected as far as 61cm and 91cm below the tuyeres, the injectors were retained in a fixed position and the tuyeres raised.

The curves in Fig. 2 show that the extent to which coke consumption may be reduced by the use of any of the oxygen processes depends upon the tapping temperature required. For example, without oxygen, a coke charge of 15 per cent was required to provide a tapping temperature of 1 500 °C. By injecting oxygen at the tuyeres, this temperature could be obtained at a coke charge of 10.8 per cent. If coke is assumed to cost £80.00/t, this would represent a saving in fuel cost of £3.36/t of iron melted. The

purchased. Larger foundries can generally obtain oxygen at a lower price than smaller foundries. The current price of oxygen may be expected to be within the approximate range of £3-£8/$100m^3$, although this probably represents the extreme range, few foundries being able to purchase oxygen at the lowest price quoted and few expected to pay the highest. In addition to the cost of the oxygen purchased, the supply company imposes a rental charge for the liquid oxygen storage vessel and evaporator. For example, for a 4 t/h cupola plant this charge would be of the order of about £4 000 per annum and due account must be taken of this charge in computing the likely benefit to be derived from the use of oxygen.

Divided-blast operation: Fig. 3 shows the relations between the coke charge and the metal tapping temperature for divided-blast operation without and with oxygen. In the divided-blast cupola the best results were obtained when the oxygen was supplied to the lower row of tuyeres only. Neither tuyere nor well injection gave any further improvement in the results obtained by this simplest method of using the oxygen.

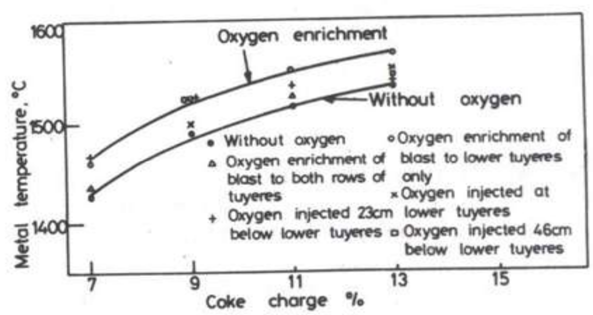


Fig. 3. Divided-blast operation – Relation between the coke charge and metal temperature showing effect of various methods of using oxygen

At a given coke charge the use of oxygen in an amount equivalent to enriching the total blast supply by 4 per cent, but by adding this oxygen only to the blast supplied to the lower row of tuyeres, increased the tapping temperature of the metal by about 35 °C.

Comparison of conventional and divided-blast operation.

Fig. 4 shows the results obtained in both conventional and divided-blast operation. This shows that at a given charge-coke consumption, divided-blast operation without oxygen provided a higher tapping temperature than that obtained in conventional operation with either 4 per cent oxygen enrichment of the blast or even 4 per cent oxygen injection at the tuyeres. It provided a similar temperature to that obtained by injecting 4 per cent oxygen into the well 23cm below the tuyeres in conventional operation.

The injection of oxygen 61-91cm beneath the tuyeres in conventional operation gave results similar to those obtained by the direct enrichment of the blast to the lower row of tuyeres in divided-blast operation.

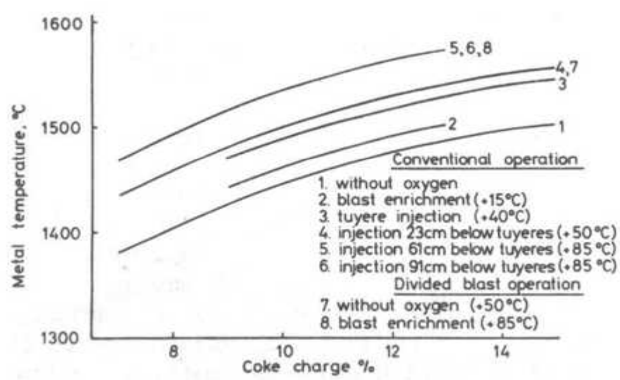


Fig. 4. Conventional and divided-blast operation – Relation between coke charge and metal temperature, showing effect of various methods of using oxygen

Economic appraisal of processes

An economic appraisal of the various processes investigated led to the conclusion, as might be expected from the results discussed above, that in general, divided-blast operation offers the most favourable means of reducing melting costs. Once the capital cost of providing the divided-blast system has been written off, there is no continuing cost as with oxygen. Some foundries have been able to recover the capital expenditure involved in converting to divided-blast operation in a period of months rather than years.

Further improvement can be obtained by using oxygen in the divided-blast cupola, but the economic justification for its use depends upon the melting conditions required and the price at which oxygen can be purchased. For example, Fig. 4 shows that a tapping temperature of 1 550 °C would require a coke charge of 15 per cent, but with 4 per cent enrichment of the blast this could be reduced to 11 per cent, yielding a saving in coke cost of £3.20/t (coke £80.00/t). The cost of oxygen (at £6.00/100m^3) would amount to £1.72/t. In this case, the net saving in cost would amount to £1.48/t.

On the other hand, for a tapping temperature of 1 500 °C, coke consumption could be reduced from 10 per cent without oxygen to 8.3 per cent with oxygen enrichment of the blast. Under these conditions the reduction in coke cost would be £1.36/t and the cost of oxygen £1.48/t. In these circumstances therefore the use of oxygen would provide no overall saving in fuel cost.

There are cases, however, where the installation of the divided-blast cupola may not be practicable, or desirable; for example, the cupola may be too short in the shaft, the daily operating period may be too short and the output too low to justify the capital expenditure. Alternatively, conversion may be difficult or impossible; for example, in the case of fully water-cooled cupolas it may be necessary to redesign and replace the entire water-cooled section.

If there is constraint to operate with one row of tuyeres, then, as shown earlier, oxygen may be used most effectively by injecting it into the cupola well. However, this process unfortunately presents a number of problems which have severely curtailed its use;

1. Its application has been confined to continuously tapped cupolas.
2. Difficulty is often encountered in accommodating the injectors at a suitable and safe distance below the tuyeres.
3. The injectors are subject to attack by metal and slag droplets and require occasional repair and replacement.
4. The injection of oxygen into the well gives rise to areas of localized lining wear in the region of the injectors.
5. For a given tapping temperature, the carbon and silicon contents of the metal produced from a given metal charge mix are lower than for other methods of operation.

On the other hand, direct enrichment of the blast is the simplest method of using oxygen, but it is also the least effective. As a compromise, the injection of oxygen at the tuyeres avoids most of the problems of well injection, is considerably more effective than direct blast enrichment and offers good prospects of economic benefit. This process is finding increasing application in foundries.

Effect of oxygen on melting rate

The tests described were all carried out at a constant equivalent blast rate of 45m^3/min (1 600 ft^3/min). At a given coke charge the method of operation, i.e. with (or without) oxygen, divided-blast, or both, had no effect of practical significance on the melting rate. However, if the coke charge was reduced to provide a given metal temperature, the melting rate increased according to the method of operation employed – as shown overleaf.

The increased melting rate obtained with the use of oxygen, divided-blast or both together can provide economic benefit by the reduction of fixed costs, avoidance of overtime working, and increased profits on the greater output.

Effect of supplementing the blast supply with oxygen

If an increased melting rate is required together with an increased tapping temperature, or if an increase in the melting rate greater than that obtained by reducing the coke charge at a constant equivalent blast rate is needed, oxygen can be used to augment the normal

Effect of method of operation on melting rate for tapping temperature of 1 500 °C. Equivalent blast rate $45 m^3/min$ (1 600 ft^3/min).

	Coke charge %	Melting rate t/h	% increase
Conventional operation			
Without oxygen	15	3.20	–
4% oxygen enrichment of blast	13	3.50	12
4% oxygen injected at tuyeres	10.8	4.00	27
4% oxygen injected 61cm below tuyeres	8.3	4.70	46
Divided-blast operation			
Without oxygen	10.0	4.20	33
4% oxygen enrichment of total blast supply. Oxygen supplied to lower tuyeres.	8.3	4.70	46

blast supply, i.e. the equivalent blast rate can be increased by the use of oxygen.

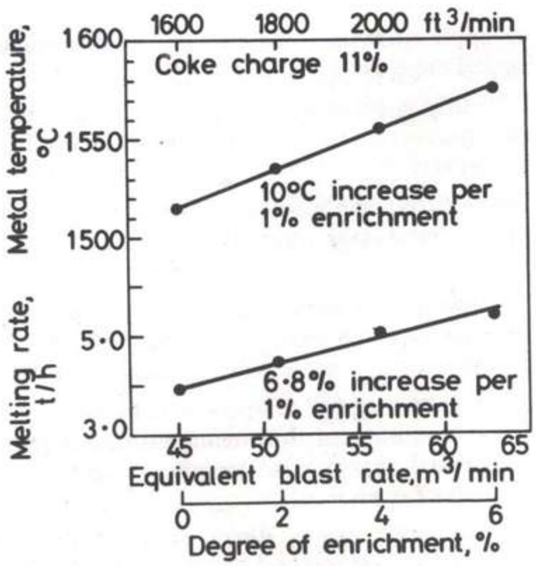


Fig. 5. Divided-blast – Effect of increasing degree of oxygen enrichment by supplementing the blast supply.

Fig. 5 shows the effect of increasing the supply of oxygen at a constant blast rate of $45 m^3/min$ (1 600 ft^3/min) in divided-blast operation. This shows that as oxygen is added to the normal blast supply, for each 1 per cent enrichment of the blast the metal temperature increased by 10 per cent and the melting rate by 6.8 per cent. By this means the melting rate of an existing cupola can be increased substantially above the normal optimum without any ill-effects of overblowing, which would occur when attempting to do so by blowing an excessive amount of blast air.

A number of foundries have found that the reduced costs and increased profits arising from the additional output obtained in this way have completely justified the use of oxygen.

Summary

It has been shown that, in the conventional cupola, oxygen can be used most effectively by injecting into the well through water-cooled lances. However, this process gives rise to problems such as injector maintenance, localized burn-out of the lining, the need for a water supply, and the difficulty of accommodating the lances at a suitable distance beneath the tuyeres. Its use is also limited to continuously tapped cupolas. The simpler process of blast enrichment is also the least effective. As a compromise which is likely to be most acceptable, the injection of oxygen at the tuyeres would appear to offer a solution with good possibilities of economic benefit.

However, in seeking a permanent improvement in efficiency and operating conditions, it is preferable to convert a cupola to divided-blast operation if the operating period is long enough and the output sufficiently high to justify the cost of conversion. Further benefit may then be obtained, if required, by the use of oxygen in the divided-blast cupola, but the economic justification for its use will depend upon the melting conditions required and the costs at which oxygen can be purchased.

In addition to its continuous use, oxygen can be, and is, used only intermittently in many foundries. The application of oxygen 10-15 minutes before tapping increases the temperature of the first metal tapped and also increases the rate of recovery of metal temperature after any prolonged stoppages. The facility of using oxygen for the rapid attainment or recovery of metal temperature reduces the quantity of cold metal pigged, or otherwise reduces the incidence of castings scrapped as a result of using cold metal.

Apart from its continuous use, therefore, the availability of oxygen affords a useful control tool to remedy poor melting conditions which may arise unavoidably, and it can also allow output rates to be increased considerably when required.

On the other hand, oxygen should not be regarded as an inexpensive commodity, and should not be made too readily available as an easy remedy to overcome persistent malpractices.

Cupola Charge Calculations and Selection of Materials

The principal objective of good cupola operation is the economical production of iron having the composition required for the types of casting to be made. To meet this objective, it is desirable to know the chemical composition of all raw materials entering the cupola, to have these in the correct proportions, and to have knowledge of the composition changes that will occur in the cupola during melting.

Raw materials available for cupola melting

The types of raw materials for use as melting stock in the cupola may be classified according to their carbon contents, as shown in Table 1. There are high-carbon materials, medium-carbon materials, and low-carbon materials. A fourth group consists of ferro-alloys and similar materials.

Table 1. Classification of charge materials.

High-carbon	– Pig iron – Refined iron
Medium-carbon	– Returned scrap – Bought cast iron scrap – Cast iron turnings and borings
Low-carbon	– Steel scrap
Alloys	– Dilute, e.g. silvery pig iron – Concentrated, e.g. ferro-silicon – Briquettes

High-carbon materials

The main source of high-carbon material for cupola-melting is pig iron, a metallic product obtained from the reduction of iron ores in the blast furnace. The alloy elements in pig iron such as carbon, silicon, sulphur and, to a certain extent, manganese, can be controlled by the method of operation of the blast furnace to produce a range of pig irons of different compositions suitable for the cupola-melting of most grades of cast irons. Phosphorus, on the other hand, cannot be controlled, since all phosphates in the ore are reduced to phosphorus which is dissolved in the liquid metal. The phosphorus content of pig iron is therefore determined by the type of iron ore used in the blast furnace, and pig irons are normally classified according to their phosphorus contents – as shown in Table 2. The four classifications are:

Hematite 0.05% phosphorus

Low-phosphorus 0.08–0.3%

Medium-phosphorus 0.3–0.7%

High-phosphorus 0.7–1.2%

Each of these types of pig iron is available with a range of carbon and silicon contents, and the iron-founder is able to select the particular analysis most suitable for his needs. Normally, if a high-carbon pig iron is required, the silicon content tends to be at the lower end of the silicon range, and vice versa. In all types the sulphur content is low, at 0.05 per cent maximum. The composition of a pig iron should normally be known with certainty, since a certificate of analysis is nearly always available with each delivery.

Pig iron generally constitutes between 10 and 30 per cent of the cupola charge, the exact amount depending upon the type of iron to be produced. The main benefits from using pig iron are:

1. To provide carbon in conjunction with other metallic materials so that, with the carbon pick-up obtained in the cupola, metal of the correct carbon content can be tapped.

2. To provide, as far as possible, the necessary amount of silicon in the charge, avoiding the need for excessive use of ferrosilicon.

3. To reduce the charged sulphur content and prevent the sulphur content of the tapped metal from rising to a dangerous level.

4. To ensure that the required maximum phosphorus content of the charge is not exceeded, taking into account the phosphorus contents of the returned and bought cast iron scrap that may also be in the charge.

The only disadvantage of using pig iron is that it is expensive, and it should therefore be used as economically as possible.

Refined irons are the second source of high-carbon materials. They are generally produced in cupolas or electric furnaces from charges containing 50 per cent or more of steel scrap, the remainder of the charge consisting of either pig iron or suitable cast iron scrap. The composition of these irons tends to be similar to that of low-phosphorus pig iron in that the sulphur content is less than 0.05 per cent and the phosphorus content is between 0.1 and 0.2 per cent. They can differ slightly from blast-furnace low-phosphorus pig irons in that carbon contents as low as 2.6 per cent can be obtained, and they may be alloyed with other elements, such as chromium and nickel. Refined irons tend to be as expensive as blast-furnace pig irons and, thus, should be used as economically as possible. However, by the use of refined irons, some foundries having only modest technical control are able to produce high-duty irons. As with pig iron, a certificate of analysis is nearly always available from the supplier with each delivery.

Medium-carbon materials

Cast iron scrap provides a wide range of medium-

Table 2. Pig iron classification.

	T.C.%	Si%	Mn%	S%	P%
Hematite	3.7–4.5	0.5–3.5	0.5–1.2	0.05max	0.05max
Low-phosphorus	3.8–4.2	1.0–4.5	0.5–2.0	0.05max	0.08–0.3
Medium-phosphorus	3.5–4.0	2.0–3.5	0.8–1.0	0.05max	0.3–0.7
High-phosphorus	3.3–3.8	2.0–4.5	0.6–1.2	0.05max	0.7–1.2

Table 3. Bought cast iron scrap.

Type of scrap	C%	Si%	Mn%	S%	P%
Light-section scrap	3.2–3.4	2.5–3.0	0.5–0.7	0.10–0.15	1.0–1.2
Textile & machinery scrap	3.1–3.3	1.8–2.2	0.5–0.7	0.10–0.15	0.7–1.0
Automobile engine	3.1–3.3	2.0–2.2	0.5–0.8	0.08–0.15	0.2
Railway chair	2.8–3.3	1.5–2.5	up to 0.5	up to 0.25	1.0–1.5
Ingot mould	3.5–3.8	1.4–1.8	0.5–1.0	0.08	0.1max
Blackheart malleable scrap	2.2–3.0	1.3–1.6	0.3–0.6	0.08–0.18	0.06
Whiteheart malleable scrap	0.2–2.5	0.3–0.8	0.2–0.3	0.15–0.25	0.06

carbon materials. The best supply of cast iron scrap is, undoubtedly, the foundry's own returned scrap – since its composition is known and should be reasonably consistent. This material should be utilized in the furnace charges to its fullest extent, which is the rate at which it becomes available. Very often, however, even when the returned scrap is fully utilized, further scrap has to be bought; this is available in several fairly readily identifiable types shown in Table 3.

The light-section scrap has an average thickness of up to 6mm, and consists of rainwater gutters and pipes, radiator castings, stove plates, etc. Care should be taken in using this scrap because of its high phosphorus content. The textile and machinery scrap consists mainly of disused textile machinery in the North of the country, and engineering machinery in the Midlands and South.

Automobile engine scrap is readily identified by appearance, and is often received in the form of complete engines and gearboxes. It is, therefore, more suitable for use in large-diameter cupolas unless time is taken to break the engines for use in smaller cupolas. Where the engines are charged complete, allowance must be made for the fact that they may contain as much as 25 per cent of steel components. In addition, this scrap may also be contaminated with non-ferrous materials such as aluminium in pistons and inlet manifolds and copper in water-cooling connections.

Two types of malleable scrap are available, but generally not in large quantities. This cannot easily be segregated, and tends to be mixed with other types of scrap.

Railway chairs are easily identified, and their size and shape make them very suitable for cupola-melting. Care should be taken, however, in the use of such material, as both sulphur and phosphorus contents are frequently high. For example, the phosphorus content has been found to be 1.8 to 2.0 per cent.

Ingot-mould scrap can be recognized by its extremely thick section with flat parallel surfaces, one of which is usually crazed.

A further source of cast iron scrap is the borings and turnings arising from the machining of iron castings. This is a cheap form of scrap, best utilized in electric furnaces, although it has been used in cupola charges with varying degrees of success. When used in cupolas, it must either be packed in canisters or, preferably, briquetted and should not exceed 20 per cent of the cupola charge. Owing to the high surface area of swarf, oxidation losses and sulphur pick-up tend to be higher than normal.

Low-carbon materials

Steel scrap is the major source of low-carbon materials for cupola-melting. Apart from the carbon content, silicon is also low, and the sulphur and phosphorus contents are normally about 0.05 per cent. There are many types of steel scrap, and care should be taken in selection; for example, by making sure that the supply of steel for grey and malleable iron production is not contaminated with alloy elements such as chromium, nickel, tungsten, etc. which might be injurious to the metal produced.

Steel of thin section in the form of sheet, wire, bales, etc. should be avoided where possible, since it is liable to be severely oxidized in the cupola and tends to restrict carbon pick-up and also increases the losses of silicon and manganese. On the other hand, steel scrap of heavy section should also be avoided where possible, as it may descend to the cupola tuyere zone before it has completely melted.

Ideally, therefore, steel scrap should be reasonably free from rust or scale and should not be less than 6mm in thickness or greater than 75mm in thickness, although it is often necessary to compromise to some extent between desirability, availability, and of course the price.

Steel scrap is normally one of the cheapest materials available for cupola-melting and should be used, therefore, to its maximum practical extent in cupola charges. The proportion of steel scrap that can be used depends on the other materials making up the cupola charge. For example, where the associated pig iron is a blast furnace product of high carbon and high silicon content, the most desirable percentage of steel that can be used will be higher than that in circumstances where the associated pig iron is of the lower-carbon type like a refined iron.

Alloys

Alloys and ferro-alloys, constituting the fourth group of charge materials, may be used as a portion of the cupola charge for regulation of the silicon and manganese levels in the iron as well as for the addition of such elements as nickel, chromium, molybdenum, copper, etc. which are sometimes used to modify the properties of the iron.

Alloys used for addition to the charge may vary from relatively dilute ferro-alloys such as silvery pig iron containing 10 to 14 per cent silicon, or manganese pig irons, to the highly concentrated materials such as ferro-alloys containing 75-80 per cent of alloy material. Selection of a particular alloy material is mainly dependent upon cost and convenience.

Silvery pig irons are available as cast pigs and, with small cupolas, should be broken to permit accurate weight adjustment. The concentrated ferro-alloys are usually cast in large slabs and are sold in different size ranges according to application. In such cases the ferro-alloy used for cupola addition must be sized small enough to permit the accurate weighing but large enough to minimize segregation as the charge settles in the cupola.

The addition of alloying materials in briquetted form is the most popular method of addition since it eliminates the weighing of small quantities of material. Each briquette contains a fixed weight of the alloy, normally 1 kg, and they are often notched to permit breaking in half if smaller additions are required.

The size of all materials entering the cupola should not exceed one third of the cupola diameter in any dimension. In the case of bars or rails, this refers to their length, while for flat plates the diagonal is the dimension to be considered. In instances where flat steel plates are charged, they should not be used in such proportions as would substantially restrict the upward passage of gas in the cupola shaft.

Composition changes during melting

When calculating the percentages of the various types of metallic materials to be used in cupola charges, it is necessary to know what changes in composition will occur during melting. Table 4 gives an approximate indication of the changes to be expected with normal acid cold-blast cupola operation. Carbon will nearly always increase, since the metal will dissolve some of the coke carbon as it drops through the melting-zone into the cupola well. There are many factors that affect the amount of carbon pick-up during melting, such as the initial carbon content of the charge, the silicon and phosphorus contents of the iron at the spout, the amount of coke used in the charge, the method of tapping the metal, the slag basicity, and the metal temperature. For continuously tapped front-slagged cupolas, the equation:

$$\text{TC\% at spout} = 2.4 + \frac{\text{TC\% in charge}}{2} - \left(\frac{\begin{matrix}(\text{Si\%} + \text{P\%})\\ (\text{at spout})\end{matrix}}{4}\right)$$

has been found to give good agreement for the tapped carbon content. No such equation has been derived for intermittently tapped cupolas, but generally the carbon content can be expected to be higher than that given by this equation.

In cupola-melting there is nearly always oxidation of silicon, and this normally amounts to between 10 and 30 per cent of the charged level of silicon. This figure will vary, depending upon melting technique; for example, the loss is decreased by high tapping temperatures, whilst it is generally increased by increasing the percentage of steel in the charge.

As with silicon, manganese is also lost during melting, owing to oxidation. This loss is generally around 20 to 30 per cent of the charged manganese level, although it will vary according to melting technique.

The sulphur content of iron is always increased in acid-cupola operation. The increase is dependent upon many factors, such as the quantity of coke in the charge, slag basicity, the quantity of steel in the charge, and the sulphur content of the coke. No reliable method of predicting the tapped sulphur content is generally available but, in practice, sulphur pick-up has been found to be as low as 10 per cent and as high as 80 to 90 per cent of the charged sulphur level. For calculating furnace charges, therefore, some prior knowledge of the magnitude of sulphur pick-up under similar operating conditions must be available.

Phosphorus tapped is generally considered to be the same as phosphorus charged. A small increase can sometimes be detected, but normally this is so small as to be insignificant.

Typical cupola charges

In the UK, iron castings are specified to be made in a particular grade of iron, and these grades are defined in terms of the tensile strength of irons when cast as 30mm (1.2 in) diameter bars. For example, a Grade 220 iron has a tensile strength of 220 N/mm^2 and a Grade 260 iron has a tensile strength of 260 N/mm^2. Metal composition largely determines tensile strength, and Greenhill[1] has shown the range of compositions required to produce the particular grades of grey iron. This report also containes typical charge mixtures used for the production of the various grades of iron. A grade 17 iron is shown to have the following composition:

TC	**3.0–3.2%**
Si	**1.6–1.9%**
Mn	**0.6–0.8%**
S	**0.15% maximum**
P	**0.3% maximum**

A typical cupola charge to produce this iron is given as:

25%	**low-phosphorus pig iron**
25%	**steel scrap**
35%	**grade 17 foundry scrap**
15%	**automotive cast iron scrap**

This charge mixture takes into account the expected changes in composition likely to occur during melting.

Cupola-charge calculation

How can the charge make-up be calculated? Table 5 shows a simple case of a charge consisting of 50 per cent pig iron having 2.0 per cent silicon, and 50 per cent steel scrap with 0.1 per cent silicon. By adding an equal quantity of steel the 2 per cent of silicon in the pig iron will be diluted to 1.0 per cent, so that the pig iron contributes 1.0 per cent silicon to the final charge composition. Likewise, the silicon in the steel scrap can be considered to be diluted by the pig iron to 0.05 per cent silicon and hence, the steel contributes 0.05 per cent silicon to the final charge compo-

Table 4. Composition changes during cold-blast cupola operation.

Carbon	– Gain	: dependent on charged C, Si, P etc.
Silicon	– Loss	: 10–30% of charge content
Manganese	– Loss	: 20–30% of charge content
Sulphur	– Gain	: variable, 10–60% of charge content
Phosphorus	– No change	

Table 5. Charge calculation for silicon (1).

	%Si	Contribution to charge.	
50% Pig iron	2.0	2.0 x	= 1.00%
50% Steel scrap	0.1	0.1 x	= 0.05%
		Total	1.05%

Table 6. Charge calculation for silicon (2).

	%Si	Contribution to charge.	
65% Pig iron	1.8	1.8 x	= 1.17%
35% Steel scrap	0.10	0.10 x	= 0.04%
		Total	1.21%

sition. The actual calculation subconsciously carried out is to multiply the silicon content by the percentage of that component in the total charge and divide by the total, that is 100 per cent. By addition of the two values, the final charge composition is found to be 1.05 per cent silicon. This same calculation is carried out for other elements.

Charge calculation will be more difficult if the charge consists of 65 per cent pig iron containing 1.8 per cent silicon, and 35 per cent steel scrap with 0.10 per cent silicon as shown in Table 6, but the same method of calculation can be used. When the charge mixture is made up to five components and similar calculations for the carbon, silicon, manganese, sulphur and phosphorus have to be made, calculation becomes more difficult and tedious but can be assisted by use of tables compiled for such calculations (Tables 7 & 8). When the percentage of the element in the material and the percentage of material in the charge are known, Table 7 can be used to determine the percentage of element contributed by that material to the charge. Hence, 65 per cent of pig iron having a silicon content of 1.8 per cent will contribute 1.17 per cent of silicon to the final charge composition, as shown in the table. Likewise, 35 per cent of steel scrap having a silicon content of 0.10 per cent can be found to contribute 0.4 per cent of silicon. Thus the final charge composition is 1.21 per cent silicon.

After this demonstration of how this table can be used to calculate a simple charge composition, the charge calculation for a Grade 17 iron can be attempted. For the purpose of this charge calculation, it will be assumed that the composition of the iron required by the foundry is carbon 3.1 per cent, silicon 1.75 per cent, manganese 0.70 per cent, sulphur less than 0.15 per cent and phosphorus less than 0.3 per cent, and the typical cupola charge make-up suggested earlier will be used. The analysis of the materials available for the cupola charge is shown in Table 9. Two pig irons are available and the engine scrap is the source of low-phosphorus cast iron scrap. To start, pig iron A has been selected in preference to pig iron B, and the charge calculation sheet is given in Table 10. The amount of carbon, silicon, sulphur, manganese and phosphorus contributed by each material in the charge to the final charge composition has been calculated by the use of Table 7 and, by simple addition, the total charge composition has been determined.

The charge composition must be corrected for changes in composition that will occur during melting. In this case, a silicon loss of 15 per cent of the charged level has been assumed and a 25 per cent loss of manganese. No change in phosphorus content is expected. After determination of the tapped silicon and phosphorus contents and knowing the charged carbon level, the equation given earlier can be used to determine the tapped carbon content for a continuously tapped cupola. Previous experience of operating the cupola is the only method of determining sulphur pick-up and, in this case, it has been estimated to be 0.04 per cent. The tapped iron is to be treated in the ladle with 0.25 per cent silicon as ferro-silicon.

Comparison of the calculated final composition with the target figures shows that the carbon content is 0.19 per cent too high, the silicon and manganese contents are lower than desired – but can be corrected by the addition of ferro-alloys, and the sulphur and phosphorus contents are below the permitted maxima. To reduce the carbon content, a lower-carbon pig iron such as pig iron B available can be selected and Table 11 shows the charge calculation sheet when this pig iron is used. The amount of carbon contributed by the 25 per cent pig iron has fallen from 0.93 per cent when pig iron A was used to 0.75 per cent for pig iron B. The rest of the calculation is carried out as in the previous example to give the final composition. Although the carbon figure is still higher than the

Table 7. Calculation for main charge.

Percentage of element in material used	Percentage of total charge: 5	10	15	20	25	30	35	40	45	50	55	60	65	70	75	80
	Percentage of element contributed to composition of charge															
0.1		0.01	0.02	0.02	0.03	0.03	0.04	0.04	0.05	0.05	0.06	0.06	0.07	0.07	0.08	0.08
0.2	0.01	0.02	0.03	0.04	0.05	0.06	0.07	0.08	0.09	0.10	0.11	0.12	0.13	0.14	0.15	0.16
0.3		0.03	0.05	0.06	0.08	0.09	0.11	0.12	0.14	0.15	0.17	0.18	0.20	0.21	0.23	0.24
0.4	0.02	0.04	0.06	0.08	0.10	0.12	0.14	0.16	0.18	0.20	0.22	0.24	0.26	0.28	0.30	0.32
0.5		0.05	0.08	0.10	0.13	0.15	0.18	0.20	0.23	0.25	0.28	0.30	0.33	0.35	0.38	0.40
0.6	0.03	0.06	0.09	0.12	0.15	0.18	0.21	0.24	0.27	0.30	0.33	0.36	0.39	0.42	0.45	0.48
0.7		0.07	0.11	0.14	0.18	0.21	0.25	0.28	0.32	0.35	0.39	0.42	0.46	0.49	0.53	0.56
0.8	0.04	0.08	0.12	0.16	0.20	0.24	0.28	0.32	0.36	0.40	0.44	0.48	0.52	0.56	0.60	0.64
0.9		0.09	0.14	0.18	0.23	0.27	0.32	0.36	0.41	0.45	0.50	0.54	0.59	0.63	0.68	0.72
1.0	0.05	0.10	0.15	0.20	0.25	0.30	0.35	0.40	0.45	0.50	0.55	0.60	0.65	0.70	0.75	0.80
1.1		0.11	0.17	0.22	0.28	0.33	0.39	0.44	0.50	0.55	0.61	0.66	0.72	0.77	0.83	0.88
1.2	0.06	0.12	0.18	0.24	0.30	0.36	0.42	0.48	0.54	0.60	0.66	0.72	0.78	0.84	0.90	0.96
1.3		0.13	0.20	0.26	0.33	0.39	0.46	0.52	0.59	0.65	0.72	0.78	0.85	0.91	0.98	1.04
1.4	0.07	0.14	0.21	0.28	0.35	0.42	0.49	0.56	0.63	0.70	0.77	0.84	0.91	0.98	1.05	1.12
1.5		0.15	0.23	0.30	0.38	0.45	0.53	0.60	0.68	0.75	0.83	0.90	0.98	1.05	1.13	1.20
1.6	0.08	0.16	0.24	0.32	0.40	0.48	0.56	0.64	0.72	0.80	0.88	0.96	1.04	1.12	1.20	1.28
1.7		0.17	0.26	0.34	0.43	0.51	0.60	0.68	0.77	0.85	0.94	1.02	1.11	1.19	1.28	1.36
1.8	0.09	0.18	0.27	0.36	0.45	0.54	0.63	0.72	0.81	0.90	0.99	1.08	1.17	1.26	1.35	1.44
1.9		0.19	0.29	0.38	0.48	0.57	0.67	0.76	0.86	0.95	1.05	1.14	1.24	1.33	1.43	1.52
2.0	0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80	0.90	1.00	1.10	1.20	1.30	1.40	1.50	1.60
2.1		0.21	0.32	0.42	0.53	0.63	0.74	0.84	0.95	1.05	1.16	1.26	1.37	1.47	1.58	1.68
2.2	0.11	0.22	0.33	0.44	0.55	0.66	0.77	0.88	0.99	1.10	1.21	1.32	1.43	1.54	1.65	1.76
2.3		0.23	0.35	0.46	0.58	0.69	0.81	0.92	1.04	1.15	1.27	1.38	1.50	1.61	1.73	1.84
2.4	0.12	0.24	0.36	0.48	0.60	0.72	0.84	0.96	1.08	1.20	1.32	1.44	1.56	1.68	1.80	1.92
2.5		0.25	0.38	0.50	0.63	0.75	0.88	1.00	1.13	1.25	1.38	1.50	1.63	1.75	1.88	2.00
2.6	0.13	0.26	0.39	0.52	0.65	0.78	0.91	1.04	1.17	1.30	1.43	1.56	1.69	1.82	1.95	2.08
2.7		0.27	0.41	0.54	0.68	0.81	0.95	1.08	1.22	1.35	1.49	1.62	1.76	1.89	2.03	2.16
2.8	0.14	0.28	0.42	0.56	0.70	0.84	0.98	1.12	1.26	1.40	1.54	1.68	1.82	1.96	2.10	2.24
2.9		0.29	0.44	0.58	0.73	0.87	1.02	1.16	1.31	1.45	1.60	1.74	1.89	2.03	2.18	2.32
3.0	0.15	0.30	0.45	0.60	0.75	0.90	1.05	1.20	1.35	1.50	1.65	1.80	1.95	2.10	2.25	2.40
3.1		0.31	0.47	0.62	0.78	0.93	1.09	1.24	1.40	1.55	1.71	1.86	2.02	2.17	2.33	2.48
3.2	0.16	0.32	0.48	0.64	0.80	0.96	1.12	1.28	1.44	1.60	1.76	1.92	2.08	2.24	2.40	2.56
3.3		0.33	0.49	0.66	0.83	0.99	1.16	1.32	1.49	1.65	1.82	1.98	2.15	2.31	2.48	2.64
3.4	0.17	0.34	0.51	0.68	0.85	1.02	1.19	1.36	1.53	1.70	1.87	2.04	2.21	2.38	2.55	2.72
3.5		0.35	0.53	0.70	0.88	1.05	1.23	1.40	1.58	1.75	1.93	2.10	2.28	2.45	2.63	2.80
3.6	0.18	0.36	0.54	0.72	0.90	1.08	1.26	1.44	1.62	1.80	1.98	2.16	2.34	2.52	2.70	2.88
3.7		0.37	0.56	0.74	0.93	1.11	1.30	1.48	1.67	1.85	2.04	2.22	2.41	2.59	2.78	2.96
3.8	0.19	0.38	0.57	0.76	0.95	1.14	1.33	1.52	1.71	1.90	2.09	2.28	2.47	2.66	2.85	3.04
3.9		0.39	0.59	0.78	0.98	1.17	1.37	1.56	1.76	1.95	2.15	2.34	2.54	2.73	2.93	3.12
4.0	0.20	0.40	0.60	0.80	1.00	1.20	1.40	1.60	1.80	2.00	2.20	2.40	2.60	2.80	3.00	3.20
4.1		0.41	0.62	0.82	1.03	1.23	1.44	1.64	1.85	2.05	2.26	2.46	2.67	2.87	3.08	3.28
4.2	0.21	0.42	0.63	0.84	1.05	1.26	1.47	1.68	1.89	2.10	2.31	2.52	2.73	2.94	3.15	3.36
4.3		0.43	0.65	0.86	1.08	1.29	1.51	1.72	1.94	2.15	2.37	2.58	2.80	3.01	3.23	3.44
4.4	0.22	0.44	0.66	0.88	1.10	1.32	1.54	1.76	1.98	2.20	2.42	2.64	2.86	3.08	3.30	3.52
4.5		0.45	0.68	0.90	1.13	1.35	1.58	1.80	2.03	2.25	2.48	2.70	2.93	3.15	3.38	3.60
4.6	0.23	0.46	0.69	0.92	1.15	1.38	1.61	1.84	2.07	2.30	2.53	2.76	2.99	3.22	3.45	3.68
4.7		0.47	0.71	0.94	1.18	1.41	1.65	1.88	2.12	2.35	2.59	2.82	3.06	3.29	3.53	3.76
4.8	0.24	0.48	0.72	0.96	1.20	1.44	1.68	1.92	2.16	2.40	2.64	2.88	3.12	3.36	3.60	3.84
4.9		0.49	0.74	0.98	1.23	1.47	1.72	1.96	2.21	2.45	2.70	2.94	3.19	3.43	3.68	3.92
5.0	0.25	0.50	0.75	1.00	1.25	1.50	1.75	2.00	2.25	2.50	2.75	3.00	3.25	3.50	3.75	4.00

Reproduced from the report of the Institute of British Foundrymen, Sub-Committee T.S. 27.[5]

Table 8. Calculation of increments from alloying materials.

Weight of alloying material per tonne, kg	Percentage of element present in alloying material										
	40	45	50	60	65	70	75	80	85	92	100
	Percentage increase of added elements										
1	0.04	0.05	0.05	0.06	0.07	0.07	0.08	0.08	0.09	0.09	0.1
2	0.08	0.09	0.10	0.12	0.13	0.14	0.15	0.16	0.17	0.18	0.2
3	0.12	0.14	0.15	0.18	0.20	0.21	0.23	0.24	0.26	0.28	0.3
4	0.16	0.18	0.20	0.24	0.26	0.28	0.30	0.32	0.34	0.37	0.4
5	0.20	0.23	0.25	0.30	0.33	0.35	0.38	0.40	0.43	0.46	0.5
6	0.24	0.27	0.30	0.36	0.39	0.42	0.45	0.48	0.51	0.55	0.6
7	0.28	0.32	0.35	0.42	0.46	0.49	0.53	0.56	0.60	0.64	0.7
8	0.32	0.36	0.40	0.48	0.52	0.56	0.60	0.64	0.68	0.74	0.8
9	0.36	0.41	0.45	0.54	0.59	0.63	0.68	0.72	0.77	0.83	0.9
10	0.40	0.45	0.50	0.60	0.65	0.70	0.75	0.80	0.85	0.92	1.0
11	0.44	0.50	0.55	0.66	0.72	0.77	0.83	0.88	0.94	1.01	1.1
12	0.48	0.54	0.60	0.72	0.78	0.84	0.90	0.96	1.02	1.10	1.2
13	0.52	0.59	0.65	0.78	0.85	0.91	0.98	1.04	1.11	1.20	1.3
14	0.56	0.63	0.70	0.84	0.91	0.98	1.05	1.12	1.19	1.29	1.4
15	0.60	0.68	0.75	0.90	0.98	1.05	1.13	1.20	1.28	1.38	1.5
16	0.64	0.72	0.80	0.96	1.04	1.12	1.20	1.28	1.36	1.47	1.6
17	0.68	0.77	0.85	1.02	1.11	1.19	1.28	1.36	1.45	1.56	1.7
18	0.72	0.81	0.90	1.08	1.17	1.26	1.35	1.44	1.53	1.66	1.8
19	0.76	0.86	0.95	1.14	1.24	1.33	1.43	1.52	1.62	1.75	1.9
20	0.80	0.90	1.00	1.20	1.30	1.40	1.50	1.60	1.70	1.84	2.0
21	0.84	0.95	1.05	1.26	1.37	1.47	1.58	1.68	1.79	1.93	2.1
22	0.88	0.99	1.10	1.32	1.43	1.54	1.65	1.76	1.87	2.02	2.2
23	0.92	1.04	1.15	1.38	1.50	1.61	1.73	1.84	1.96	2.12	2.3
24	0.96	1.08	1.20	1.44	1.56	1.68	1.80	1.92	2.04	2.21	2.4
25	1.00	1.13	1.25	1.50	1.63	1.75	1.88	2.00	2.13	2.30	2.5
26	1.04	1.17	1.30	1.56	1.69	1.82	1.95	2.08	2.21	2.39	2.6
27	1.08	1.22	1.35	1.62	1.76	1.89	2.03	2.16	2.30	2.48	2.7
28	1.12	1.26	1.40	1.68	1.82	1.96	2.10	2.24	2.38	2.58	2.8
29	1.16	1.31	1.45	1.74	1.89	2.03	2.18	2.32	2.47	2.75	2.9
30	1.20	1.35	1.50	1.80	1.95	2.10	2.25	2.40	2.55	3.76	3.0
31	1.24	1.40	1.55	1.86	2.02	2.17	2.33	2.48	2.64	2.85	3.1
32	1.28	1.44	1.60	1.92	2.08	2.24	2.40	2.56	2.72	2.94	3.2
33	1.32	1.42	1.65	1.98	2.15	2.31	2.48	2.64	2.81	3.04	3.3
34	1.36	1.53	1.70	2.04	2.21	2.38	2.55	2.72	2.89	3.13	3.4
35	1.40	1.58	1.75	2.10	2.28	2.45	2.63	2.80	2.96	3.22	3.5
36	1.44	1.62	1.80	2.16	2.34	2.52	2.70	2.88	3.06	3.31	3.6
37	1.48	1.67	1.85	2.22	2.41	2.59	2.78	2.96	3.15	3.40	3.7
38	1.52	1.71	1.90	2.28	2.47	2.66	2.85	3.04	3.23	3.50	3.8
39	1.56	1.76	1.95	2.34	2.54	2.73	2.93	3.12	3.32	3.59	3.9
40	1.60	1.80	2.00	2.40	2.60	2.80	3.00	3.20	3.40	3.68	4.0
41	1.64	1.85	2.05	2.46	2.67	2.87	3.08	3.28	3.49	3.77	4.1
42	1.68	1.89	2.10	2.52	2.73	2.94	3.15	3.36	3.57	3.86	4.2
43	1.72	1.94	2.15	2.58	2.80	3.01	3.23	3.44	3.66	3.96	4.3
44	1.76	1.98	2.20	2.64	2.86	3.08	3.30	3.52	3.74	4.05	4.4
45	1.80	2.03	2.25	2.70	2.93	3.15	3.38	3.60	3.83	4.14	4.5
46	1.84	2.07	2.30	2.76	2.99	3.22	3.45	3.68	3.91	4.23	4.6
47	1.88	2.12	2.35	2.82	3.06	3.29	3.53	3.76	3.40	4.32	4.7
48	1.92	2.16	2.40	2.88	3.12	3.36	3.60	3.84	4.08	4.42	4.8
49	1.96	2.21	2.45	2.94	3.19	3.43	3.68	3.92	4.17	4.51	4.9
50	2.00	2.25	2.50	3.00	3.25	3.50	3.75	4.00	4.25	4.60	5.0

After the report of the Institute of British Foundrymen, Sub-Committee T.S. 27.[5]

target figure, it is within the range of 3.0–3.2 per cent acceptable for Grade 17 iron. Once again, silicon and manganese can be corrected by the addition of ferro-alloys. and the sulphur and phosphorus contents do not exceed the maximum amounts required.

The alternative method of reducing the tapped carbon content without using the alternative lower-carbon pig iron would be to change the proportions of materials used in the charge. For example, suppose the percentage of pig iron A used in the first charge calculation was reduced from 25 per cent to 20 per cent and the percentage of steel increased from 20 per cent to 25 per cent. By doing this, there could be an added benefit of reducing the metallic charge costs. Table 12 shows the charge calculation sheet for the reduced pig iron charge of 20 per cent. In this calculation, 0.30 per cent of silicon and manganese is added as ferro-alloys or briquettes to the charge and, assuming similar losses or gains as in the previous calculation, a final composition is obtained in the ladle of carbon 3.17 per cent, silicon 1.75 per cent, manganese 0.70 per cent, sulphur 0.130 per cent, and phosphorus 0.11 per cent. The silicon, manganese, sulphur and phosphorus are as required, and although the carbon is still a little high it is within the acceptable range for Grade 17 iron. If the calculation was taken a stage further, and 20 per cent of pig iron B was used instead of pig iron A, the tapped carbon content could be reduced to 3.10 per cent.

It is apparent, therefore, that to obtain iron of the composition required at the spout using the materials available, the components of the charge and their proportions must be adjusted from the typical cupola charge make-up originally suggested.

The proportions of materials having been fixed, the total charge weight must be determined and the weights of the individual components calculated. The charge weight should normally be approximately one tenth of the weight of metal melted in one hour of continuous blowing; that is, 10 charges per hour into the cupola. If the melting rate is 10 t/h, a charge weight of one tonne can be used. For the charge in Table 12 the weights of the individual components would be:

200kg low-phosphorus pig iron
150kg engine scrap
350kg Grade 17 scrap
300kg steel scrap.

In order to calculate the amounts of ferro-alloys required to make the 0.3 per cent additions of silicon and manganese, Table 8 can be used. As shown by this table, 3kg of pure silicon and 3kg of pure manganese would be required for an addition of 0.3 per cent of each of these elements. If silicon and manganese briquettes, each containing 1kg of the appropriate element are used, then 3 of these briquettes per charge would be required. If the addition was to be made during a 50 per cent ferro-alloy, then 6kg of the alloy material would have to be added with each charge.

Least-cost charge

There are many ways in which a cupola charge can be made up to give metal of the required composition at the cupola spout. For example, if there are eight different charge materials in a stockyard and five are required in the charge, there are 56 different charge mixtures which will give the correct composition at the spout. There will, however, be only one that has the lowest cost and gives the biggest profit per tonne of castings. It is an almost impossible task for a metallurgist to calculate this least-cost charge, so BCIRA has developed and provides to foundries a computer program that will rapidly do this, provided that the appropriate information is available.

There are three ways in which the service can be employed:

1. To make better use of materials already in stockyard.
2. To show how to use alternative raw materials not normally purchased by the foundry.
3. To determine the extent to which certain restraints reduce the savings which can be achieved.

A number of foundries have used the service, and savings in charge costs have been achieved in each case. In one foundry a saving of £2.20 per tonne was obtained by better use of materials already in the stockyard, and when alternative raw materials were considered the savings were increased to £3.53 per tonne.

It was also evident that further reduction in charge cost was precluded by the requirement that the sulphur content of the charge should not exceed 0.05 per cent. The third form of the BCIRA service was used, therefore, to determine the extent to which further reductions in charge cost could be obtained if this constraint was relaxed. It was shown that if the maximum sulphur content of the charge was allowed to rise to 0.052 per cent, a further saving of £0.96 per tonne could be obtained.

A final point is that, having calculated the exact weights of the various raw materials required in a charge, it is important that they should be weighed accurately. The molten metal tapped from a cupola cannot be expected to be any more uniform in composition that the accuracy with which the charge components are weighed.

Reference

1. **Greenhill (J.M.) Production of Grey Cast Iron to meet the Requirements of British Standard 1452: 1961. Foundry Trade Journal, March 1970. (BCIRA External Report 585).**

Table 9. Available charge materials.

Material	TC %	Si %	Mn %	P %	S %
Pig iron A	3.7	2.5	1.0	0,15	0.03
Pig iron B	3.0	2.9	0.9	0.10	0.04
Engine scrap	3.2	2.2	0.8	0.15	0.15
Grade 17 scrap	3.1	1.7	0.7	0.10	0.13
Steel scrap	0.1	0.1	0.2	0.05	0.05

Table 10. Charge calculation sheet 1.

Material	Additions %	TC%	Si%	Mn%	S5	P%
Pig iron A	25	0.93	0.63	0.25	0.008	0.05
Engine scrap	15	0.48	0.33	0.12	0.023	0.03
Grade 17 scrap	35	1.09	0.60	0.25	0.046	0.04
Steel scrap	25	0.03	0.03	0.05	0.013	0.01
Ferro-alloys						
Charge composition		2.53	1.59	0.67	0.090	0.13
Changes on melting		+0.76	−0.22	−0.17	+0.040	nil
Composition as tapped		3.29	1.37	0.50	0.130	0.13
Ladle addition			+0.25			
Final composition		3.29	1.62	0.50	0.130	0.13

Table 11. Charge calculation sheet 2.

Material	Additions %	TC%	Si%	Mn%	S%	P%
Pig iron B	25	0.75	0.73	0.23	0.010	0.03
Engine scrap	15	0.48	0.33	0.12	0.023	0.03
Grade 17 scrap	35	1.09	0.60	0.25	0.046	0.04
Steel scrap	25	0.03	0.03	0.05	0.013	0.01
Ferro-alloys						
Charge composition		2.35	1.69	0.65	0.092	0.11
Changes on melting		+0.84	−0.25	−0.16	0.04	nil
Composition as tapped		3.19	1.44	0.49	0.132	0.11
Ladle addition			0.25			
Final composition		3.19	1.69	0.49	0.132	0.11

Table 12. Charge calculation sheet 3.

Material	Additions %	TC%	Si%	Mn%	S%	P%
Pig iron A	20	0.74	0.50	0.20	0.006	0.03
Engine scrap	15	0.48	0.33	0.12	0.023	0.03
Grade 17 scrap	35	1.09	0.60	0.25	0.046	0.04
Steel scrap	30	0.03	0.03	0.05	0.015	0.01
Ferro-alloys			0.30	0.30		
Charge composition		2.34	1.76	0.93	0.090	0.11
Changes on melting		+0.83	−0.26	−0.23	0.040	nil
Composition as tapped		3.17	1.50	0.70	0.130	0.11
Ladle addition			0.25			
Final composition		3.17	1.75	0.70	0.130	0.11

Materials Handling and Stockyard Layout

By virtue of the weight and variety of materials involved, the problems associated with cupola charging and with stockyard layout are essentially those of materials handling. As in many other instances, we have resorted to mechanized methods of moving materials as a means of overcoming the shortage and high cost of labour. Although many foundries still rely heavily on manual effort, the application of mechanical charging methods has steadily increased. Great improvement in the design and reliability of such equipment has been achieved, and there are now a number of well-established and well-tried systems available to the industry.

Cupola charging and stockyard layout are closely connected, so it is impossible to consider one without the other if any improvement in working efficiency is to be achieved. In formulating any plan of improvement, it is important to recognize the following basic factors:

1. The provision of space for the storage of an adequate supply of raw materials, the extent of which must bear some relation to the daily consumption and the frequency with which replacements can be obtained.
2. The maintenance of material stocks as near as possible to the cupolas in order to facilitate their subsequent reclamation.
3. The incorporation of good handling facilities in order to reduce manual effort.
4. The layout and equipment used must conform to the technical aspects of the melting unit, i.e. melting rate, number of furnaces to be served, make-up of charges, etc.
5. Due recognition must be given to the limitations imposed by the site conditions and, where possible, the plan should take advantage of the nature of the site.
6. The capital cost involved by the replanning must be capable of being off-set in a reasonable number of years by the labour saved. In this connection due regard must be paid to the anticipated future molten-metal requirements of the foundry.

Whilst certain well-defined principles have been established, their application is not always a straightforward task and it is invariably necessary to consider each foundry on its own merits.

Labour utilization

Initially, it is useful to consider the conventional method of platform charging, as this still forms the normal procedure for many ironfoundries in this country. By doing so, a clearer picture of the value of mechanical equipment and the possibilities of improvements may be obtained.

In such cases, it is usual to provide metal for a restricted period towards the end of the day in order to allow as much time as possible for moulding. Whilst the tonnages to be melted may vary from 2 to 25 tonnes, the melting period seldom exceeds three hours. Such melting arrangements have, to a large extent, dictated the method of charging, it being usual for labour to be employed for several hours prior to the melt for the collection or reclamation of charge materials from the stockyard by means of wheelbarrows. While there are one or two variants, the materials are elevated for storage on a platform at charging-sill height by means of a lift hoist or hoist block. During the melting period the materials are made up into charges and hand-fed to the furnace. This general procedure is adopted for a wide range of melting rates, the only change being in the number of men involved in the operation.

The system is completely flexible, as it is applicable to a wide range of throughput and distance between stockyard and melting plant. It is, however, dependent on manual effort and, for high throughputs, a large number of men can be involved because of the double handling. For convenience, the labour utilization can be considered in two parts; first, that involved on the platform for making up the charges and for hand-feeding them into the furnace, and second, that employed for reclaiming materials from the stockyard to the cupola platform. As far as the actual charging operation itself is concerned, it is convenient to make comparisons on the basis of the tonnes of metal charged per man hour. In effect this figure is the average melting rate divided by the number of men employed for charging.

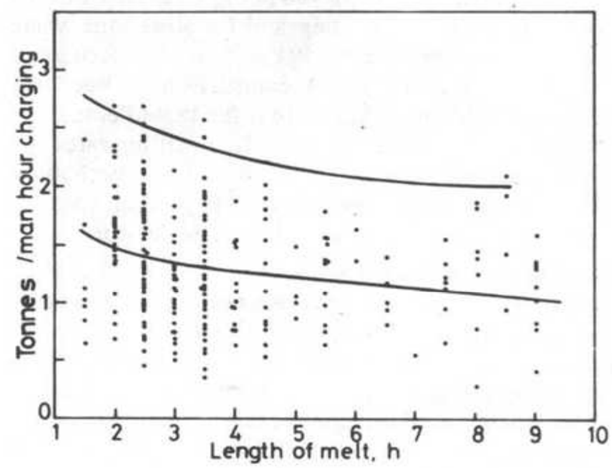


Fig. 1 Charging performance of cupola operators.

Fig. 1 shows the values in terms of tonnes per man hour charged, for some 400 foundries, plotted against melting period. For any particular melting period the values are distributed normally about a mean figure. The mean values represent the most frequently occurring value, and their variation with melting period is shown by the lower line in the graph. Ninety-five per cent confidence limits may be set up about this mean line to indicate the maximum rate of charging which might be expected to arise, statistically, values for charging rates in excess of the upper line not being expected to arise more that one in twenty times. The upper limit may be used as a yard-stick against which operator performance may be assessed.

A number of interesting points arise from this analysis. First, it will be seen that the theoretical maximum rate of charging falls with the increase in the melting period, i.e. 2.7 tonnes per man hour for short melts to 2.0 tonnes per man hour for long melts. Second, the higher values, namely those near the 95 per cent confidence limit, are more usually associated with good handling facilities, whilst those nearer the mean value or below it have poor handling facilties. In other words, values between the mean line and the upper limit represent varying degrees of efficiency.

The handling facilities on the charging-platform of many foundries are extremely poor, with the result that excessive labour has to be employed. Stocks of materials are usually held in indiscriminate piles on the platform, and the weigh scale is normally set about 15cm above the platform. Three, or even four, separate lifts may be involved with this procedure, one from stocks into a barrow, a second from barrow to weigh scale, and a third from weigh scale into the furnace. Improvements in such platform situations are fairly readily achieved.

The most effective system is to use a dial weigh scale and tipping-skip suspended from a light trolley carried on a length of monorail supported overhead and running between stocks of material and the cupola sill. The charges can be made up as the various materials are reclaimed, the operator finally tipping the whole charge directly into the furnace. With a system of this kind the operators will have no difficulty in achieving the maximum charging rate represented by the 95 per cent confidence limit, i.e. 2.7 tonnes per hour for short melts and 2.0 tonnes per hour for longer melts of seven or eight hours' duration.

Data of the kind displayed in Fig.1 can be produced for both platform charging and for situations where the operators feed the bucket or skip of a mechanical charger at ground level. A comparison between the two sets of figures indicates that the labour employed is normally the same for equivalent melting rates and melting periods. In effect, this implies that mechanical charging units themselves do not, in general, ensure a saving in the amount of actual charging labour.

Reclamation of materials

The second aspect of labour utilization is that concerned with the reclamation of materials from the stockyard to the cupola platform of the skip bucket of a charging-machine. This is demonstrated in Fig. 2, which shows the relation between the man-hours expended reclaiming materials for each melt and the amount of metal melted, for three groups of foundries.

The first group covers foundries, using the platform-charging method, which melt every day for three or four hours. The second group also covers foundries using the platform-charging system, but which only melt on one or two days of each week. The handling problem is no greater than for the first group, and the greater number of man-hours spent reclaiming materials is due to the fact that the labour involved tends to use the time which is available rather than the time required.

The third group covers foundries melting every day and which have cupolas equipped with some form

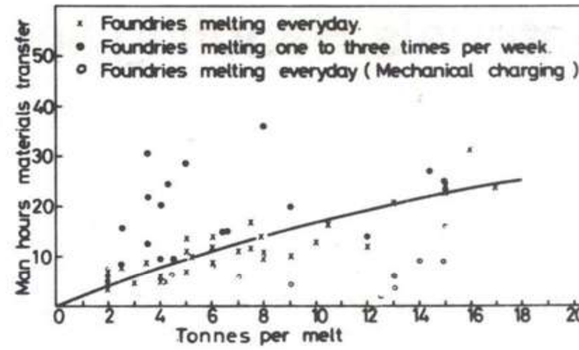


Fig. 2 Rate of working-man-hours per tonne of material transferred.

of charging machine. The lower number of man-hours spent reclaiming materials by foundries using mechanical-charging equipment arises largely from the fact that most of the reclamation takes place during the melting period.

The implication is that the real value of a mechanical-charging machine is only achieved if the stocks of material are held near to the charging unit. The further the storage area is from the cupola, the more labour has to be expended for reclamation; then, for small tonnages, the full value of the charging machine may in fact be lost.

Although the main advantage of mechanical-charging units is the reduction in labour force which they offer, there are additional aspects to be noted:

1. With a mechanical-charging machine the operators work at ground level and are therefore not subject to the heat and fume which can cause unpleasant conditions when charging from a cupola platform.
2. Generally, it is possible to exercise greater supervision over the charging procedure when this takes place at ground level.
3. While the installation of a mechanical-charging machine can be expensive, its cost has to be considered in relation to the cost of the alternative: of installing a lift hoist, and of erecting a platform of suitable size and strength to store sufficient materials for a day's melt.

Inclined skip-hoist and drop-bottom bucket-hoist charging machines

Of the various types of mechanical-charging equipment available, the inclined skip-hoist or inclined drop-bottom bucket hoist chargers have found the most favour. These units have been manufactured for many years, and provide a reliable and well-tried system catering for the needs of 99 out of 100 foundries. The equipment is simple to operate and maintain, and the skip or bucket type charger is relatively inexpensive to install. With cupolas of normal height, say 6 – 7½m (20 to 25 ft), the cycle time is of the order of two minutes, so that a rate of feeding of 15 charges per hour can be attained with charge coke being loaded separately.

Skip hoist chargers are employed for a wide range of melting rate but, in general, they are more suitable for the smaller furnaces of, say, less than 90 cm (36 inches) diameter and a melting rate of 5 tonnes per hour. One reason for this limitation is the tendency for the charge to 'ramp' on deposition in the furnace.

This ramping can be appreciable in large-diameter furnaces, and undesirable if the charge contains a wide variety of constituents and the metal has to be held to close limits of composition. Experience indicates that the charges tend to spread more evenly if the level of the stock is kept a short distance below the charging-sill. For this reason the charging-sill on furnaces charged in this way should be 60 – 90 cm (2 – 3 feet) higher than would be the case for hand-charging.

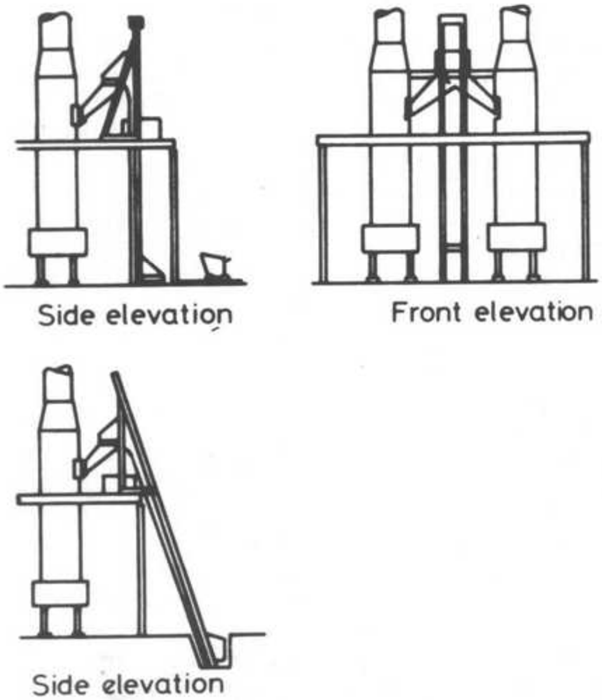


Fig. 3 Cupola charger with bifurcated discharge chutes.

Until recently, it was common practice to install a skip-hoist charger between a pair of furnaces (Fig. 3), the skip bucket tipping its contents into a double-trouser-leg-type of chute called a breeches chute. A flap gate in the chute directs the charge material into whichever furnace is in use. Although this arrangement provides the cheaper installation, the modern tendency is to construct the charger so that it can be swung from one furnace to the other. The breeches-chute system has three important disadvantages:

a. There is a high risk that the charge may 'scaffold' in the breeches chute.

b. The job of repairing one furnace while the other is in operation is both dusty and noisy for the man inside the furnace.

c. There is always a risk that the flap gate on the chute has not been fastened and that charge material may be diverted into the furnace which is being repaired.

In the case of cupolas with melting rates above 4 to 5 tonnes per hour, it is more usual to use an inclined drop-bottom bucket charger (Fig. 4). Again, the unit can be designed to serve two furnaces, a swivelling mechanism being incorporated to allow the charger to be moved from one furnace to the other.

Several recent cupola installations employ a vibratory conveyor for feeding charge materials into the cupola, with the object of reducing the size of the charge opening in the cupola shaft. The volume of air

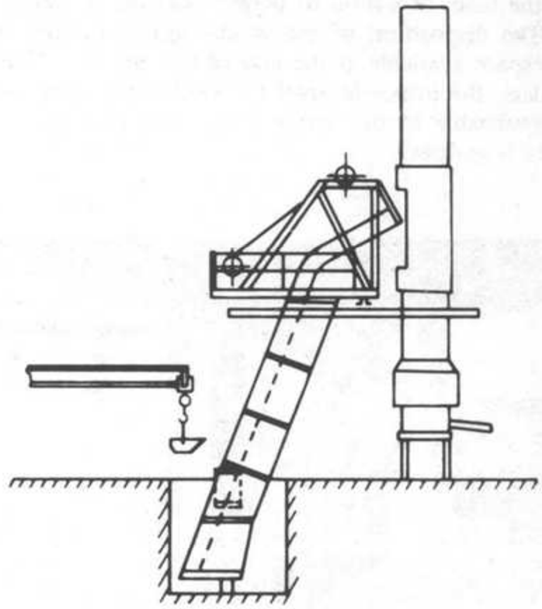


Fig. 4 Inclined bucket charger.

drawn into the cupola through the charge door is related to the size of this opening. For some designs of cupola emission-control equipment the amount of air admitted in this way may effect the size and cost of the installation. This is particularly the case where furnaces have a melting rate of more than 10 tonnes per hour.

Stockyard layout

The advantage of charging-machines is only obtained if the stocks of charge materials are held as near as possible to the cupolas, so that reclamation can be carried out at the same time as the charges are being made up.

Dial weigh scale and tipping skip

For daily throughputs up to 20 to 25 tonnes per day and melting rates up to 4 to 5 tonnes per hour, a simple but effective stockyard layout can be developed on the basis of the suspended dial weigher and tipping system of charge make-up. A typical example is shown in Fig. 5, where the stock bunkers are arranged on either side of a central pathway above which is fixed a monorail. This monorail carries a light trolley from which is suspended a dial weigh scale and tipping skip. The operator pushes the trolley along the central pathway, weighing the appropriate portion of material from each bunker. The bunkers are made from reinforced concrete or, preferably, from old railway sleepers, and are arranged to accept tipping lorries, metallic materials being on one side of the central pathway and coke on the other – the latter being roofed over to provide protection from the weather (Fig. 6).

When the charge is complete, the operator moves to the cupola and tips the charge into the skip bucket of the inclined skip charger. With such arrangement, it is well within the capability of one operator to sustain a melting rate of 2 tonnes per hour for seven or eight hours. When the melting rate exceeds 2 tonnes per hour, it is necessary to introduce additional operators, i.e. two operators for melting rates of 2 to 4 tonnes per hour. It may then be advisable to provide two suspended tipping skips and arrange the monorail in the form of a loop to permit one-way movement.

The disposition of the stock bunkers depends on the space available at the rear of the furnaces. Nevertheless, the principle involved remains the same, and is applicable to the majority of small and medium-sized foundries.

Fig. 5 Dial weigh scale and tipping skip.

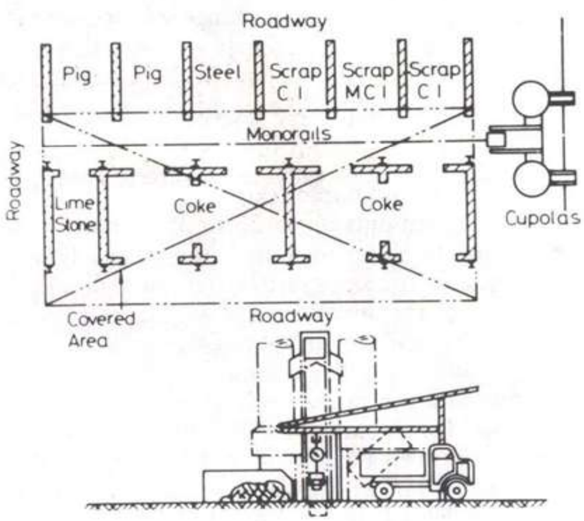


Fig. 6 Stockyard with charge make-up using dial weigh scale and tipping skip.

The chief limitation to this system is the quantity of materials which can be stored within reach of the monorail. If materials are delivered by tipping lorry it is not possible to store at a height greater than 60 – 90cm (2 – 3 feet) above floor level. This means that a 3 x 3m (10 x 10ft) bunker will be capable of storing 15 to 20 tons of pig iron or 10 to 12 tons of cast iron scrap. The storage of coke also presents a problem, and it is generally difficult to store more than 25 to 30 tonnes in a bunker which is replenished by a tipping lorry.

As the daily throughput increases, it becomes increasingly difficult to store all the charge materials near enough to the furnaces, and modification is required to avoid the operators' having to walk considerable distances to make up and reclaim charge materials.

However, the monorail system of charge make-up can be adopted even for high throughputs, by using the bunkers near the monorail only for the storage of day-to-day stocks. It is then necessary to provide another open area for main storage, and to develop a convenient system of transferring materials from this main storage area to the day-to-day storage bunkers. While any open ground can be used as the main storage area, and lorries or front-loaders utilized for the transfer of material, such a system can be labour-intensive. For this reason the layout of the stockyard for most large foundries has been based on the use of an overhead gantry crane.

Overhead gantry cranes

In such cases the gantry covers sufficient floor space to accommodate both main stocks and day-to-day stock bunkers, the crane with magnet attachment being used to transfer materials from main stocks to the local stocks. Charge make-up from the local day-to-day stock bunkers is then effected by the dial weigher tipping-skip system. Fig. 7 illustrates a system where the gantry crane covers the stockyard, servicing two pairs of cupolas. Whereas the manual handling of pig iron and scrap can only be carried out at about 2 to 3 tonnes per man-hour, a magnet gantry crane can unload and transfer metallic materials at a rate of 10 to 15 tonnes per hour. Additionally, a magnet crane enables the bunkers to be placed close together, and allows them to be filled to a far greater height than is possible when unloading by hand or by tipping lorries. In fact, the height is only limited by the strength of the bunker walls.

The principle embodied in using a magnet crane in this way can be applied to the platform charging method, where the site conditions permit and a substantial platform is already in existence, and it is not worth while going to the extra expense of installing a mechanical charger. An example of this situation is illustrated in Fig. 8. The grantry crane is installed to cover the rear portion of the cupola plaform, and is used to unload lorries into the main storage area at ground level and for transferring materials into day-to-day stock bunkers stored on the platform. The charge make-up and feeding of the furnace is effected by means of monorail with suspended dial weigher and tipping skip (Fig. 9). For the same throughput, such an arrangement requires no more labour than is necessary for a system based on a gantry crane and mechanical-charging machine.

The major problem is in handling coke and limestone to the charging platform. A grab bucket can be used on the crane, but this does not generally find favour for handling coke. The alternative is to fill large skips by hand at ground level and then elevate them to the platform by means of the gantry crane.

When a crane is used in this way it will be realized that the gantry has to be set at a much greater height than when servicing a mechanical-charging machine.

Mobile crane

The installation of a gantry crane does, however, entail the provision of a rectangular area of level ground at the rear of the furnaces, and sometimes the nature of the site does not permit this. In this circum-

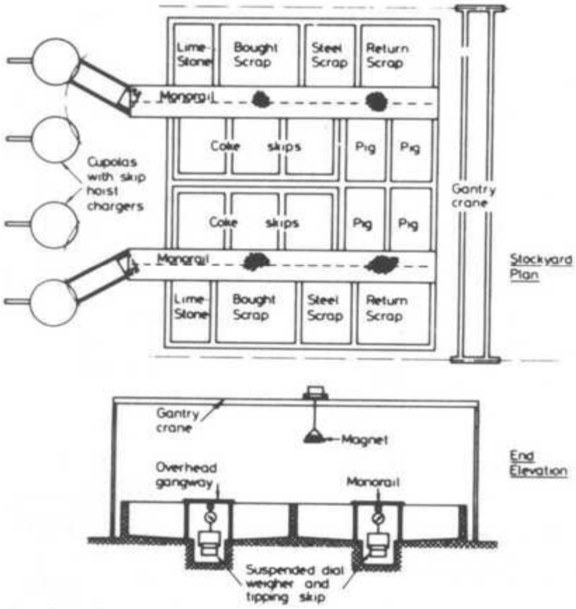


Fig. 7 **Stockyard with charge make-up by dial weigher and tipping skip and material transfer by gantry crane.**

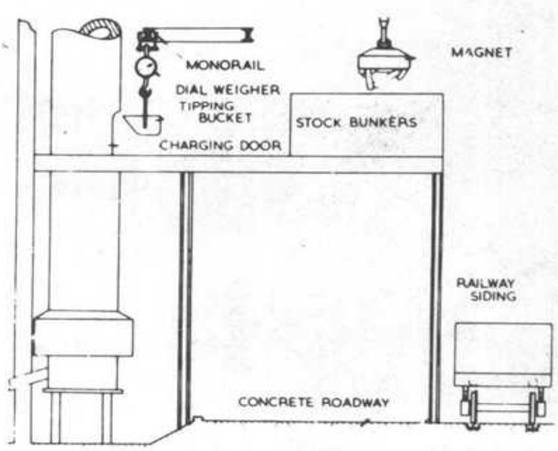


Fig. 8 **Magnet crane applied to platform charging.**

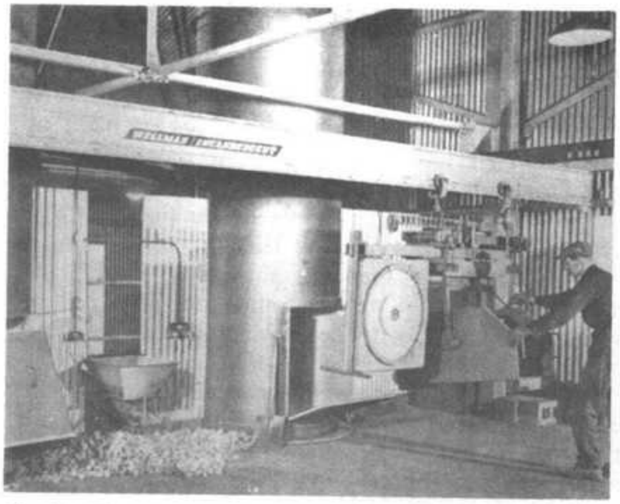


Fig. 9 **Charge make-up by dial weigher and tipping skip at platform level.**

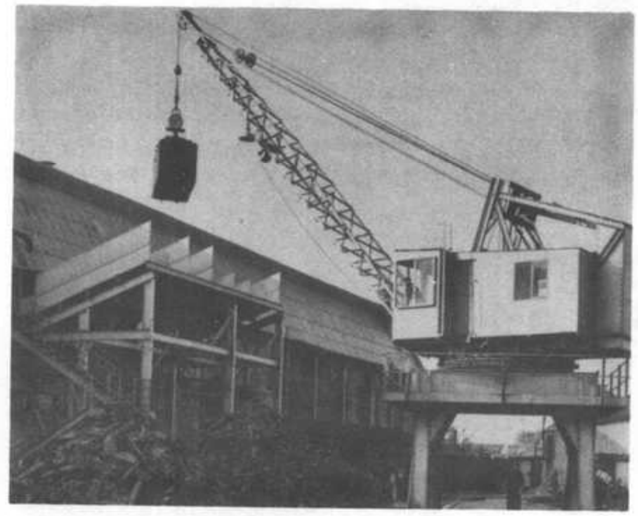


Fig.10 **Use of portal crane for material transfer.**

stance, use can often be made of a mobile jib-type crane, since this is capable of being used in stockyard areas of irregular shape. Fig. 10 illustrates the use of a portal jib crane to unload railway wagons and to elevate materials to day-to-day storage bunkers located on the cupola charging-platform. Portal cranes of this type run on railway lines, and therefore tend to be restricted in the area they cover. The use of a portal crane is particularly applicable in the case illustrated because of the triangular nature of the site and the entry of materials by railway wagon.

A diesel-operated mobile jib crane is more versatile, since it can be used almost anywhere in the yard and can, moreover, enter the foundry buildings – provided they are of reasonable height. It can be fitted with a magnet or grab-bucket, but the general speed of travel is lower than with a gantry crane.

The use of a mobile magnet crane is illustrated in an example which is also of particular interest in that it uses a stationery weigh hopper instead of a travelling dial weigher and tipping skip for making up the charges. A photograph of the mobile crane is shown in Fig. 11, and the plan and elevation of the charge make-up arrangements in this stockyard are shown in Figs. 12 & 13. Weigh hoppers of this type are being used to an increasing extent by large foundries. They consist essentially of a drop-bottom bucket which forms an integral part of a weighing mechanism and which is set immediately above the bucket of an inclined charging-machine. The charges are made up and weighed in the weigh hopper, the bottom doors of which are then opened to release the charge into the skip or bucket of the charging machine.

In this example the site conditions at the rear of the furnaces did not permit the use of a gantry crane and charge materials were therefore distributed over a wide area by tipping lorry. The foundry decided to use a mobile magnet crane to fill three-sided bins with the materials used in the cupola charge. The magnet was then released and the crane employed to transport the bins and place them on a sloping ramp surrounding a weigh hopper located a few metres above ground level. Arrangements were made to allow the skip bucket of the charging machine to descend below ground level so that it could accept the charges released from the weigh hopper.

Each of the three-sided bins holds about 2 tonnes of pig iron or scrap. The mobile crane is used also to replenish a coke hopper located above the weigh hopper. A drop-bottom bucket holding about 750kg

Fig.11 Use of mobile crane for material transfer.

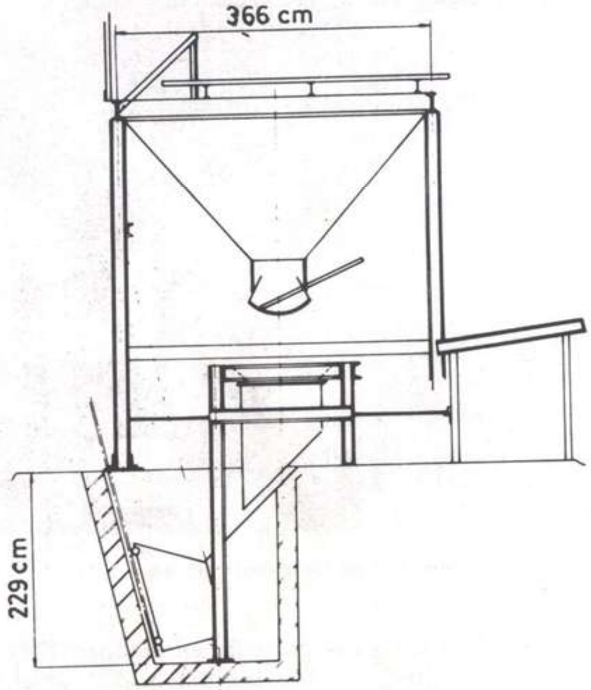


Fig.12 Weigh hopper used with mobile-crane material transfer.

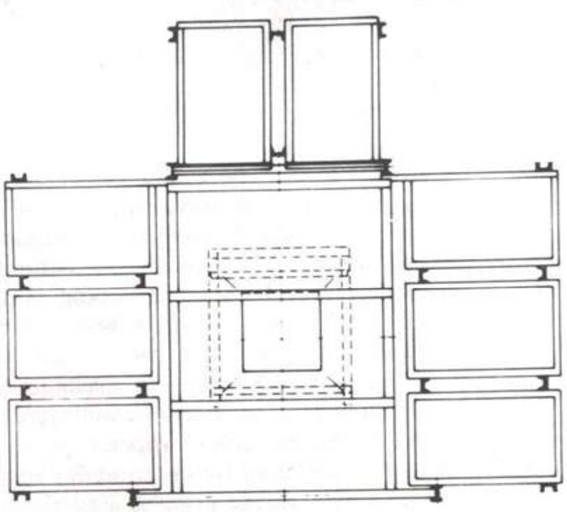


Fig.13 Plan of bunker arrangement used with mobile-crane material transfer.

of coke is filled by hand in the stockyard and then placed over the coke hoppers by the mobile jib crane. Two men are employed, on the platform surrounding the weigh hopper making up charges. They are engaged in this way for four hours each day for 30 tonnes of metal. The crane operator is only employed for half a day, reclaiming materials. The labour expenditure is thus about 13 man-hours per melt.

Before the installation, the cupolas were charged by hand from a conventional platform, four men being employed over the whole day collecting material from the stockyard and charging the furnace. The daily melt involved about 30 to 32 man-hours.

Use of magnets for charge make-up

In recent years there has been an increasing tendency to use gantry cranes with magnets for charge make-up as well as for the reclamation of metallic materials. An example of this type of installation is shown in Fig. 14. The most popular arrangement used, both in this country and abroad, is to employ a weigh hopper with bottom discharge gate set at floor level, as illustrated in Figs. 15, 16 & 17. An inclined bucket charger is employed for the actual charging of the furnace, and the design is such that when it is in its resting position, the drop-bottom bucket passes beneath the weigh hopper. The charges are made up in the weigh hopper by means of a gantry crane travelling over the stock bunkers. The magnet has a variable flux control which enables the material to be dropped off piece by piece. The rate of handling is very much higher than for manual charging, and melting rates of 15 to 20 tonnes per hour can be sustained by two men, one in the crane and the other check-weighing at the weigh hopper.

Although the check-weigh man can shovel in the coke and limestone, most charging systems of this type incorporate hoppers for the storage of these materials. These hoppers may be replenished by grab-bucket or skip carried on the gantry crane or, preferably, by belt inclined conveyor from floor level (Fig. 18). They are fitted with vibratory chutes to enable the coke and limestone to be fed into the weigh hopper.

The following two examples illustrate variations of the magnet system of charge make-up. The first installation (Figs. 19 & 20) is novel in that the weigh hopper forms an integral part of the magnet gantry crane.

The crane cab is placed central to the crane span and set much lower than normal. On lengthwise travel the cab moves up and down a pathway on either side of which are placed the stock bunkers. The magnet is used in cross travel to transfer materials from the bunkers into the weigh hopper, the crane travel and charge make-up being under the control of the operator in the cab. When the charge make-up is complete, the cab moves forward to the cupolas and the bottom gate of the weigh hopper is opened to allow the charge to be deposited in the skip of an inclined skip hoist charging machine. A second operator can be employed to feed coke and limestone direct to the skip charger or, alternatively, hoppers with vibratory feeder chutes can be used to feed the materials into the weigh hopper.

For small to medium throughputs, the magnet can also be used to replenish the stock bunker, but for larger tonnages it is necessary to install a conventional magnet crane at a higher level above the charging crane.

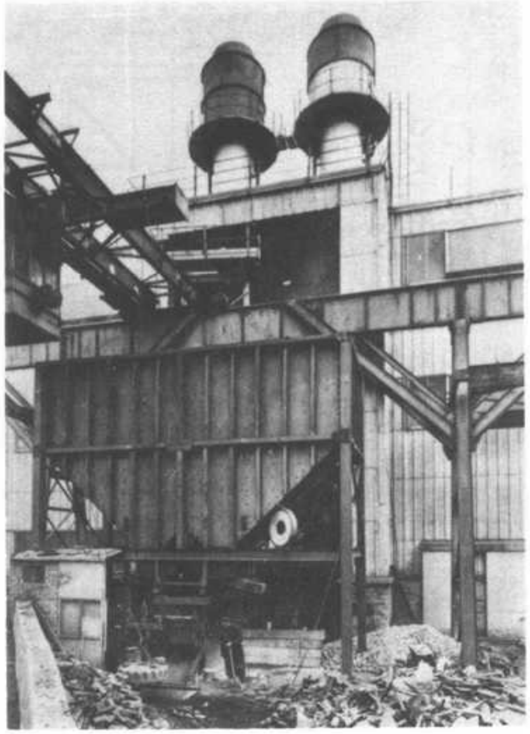


Fig.14 Use of gantry crane with magnet.

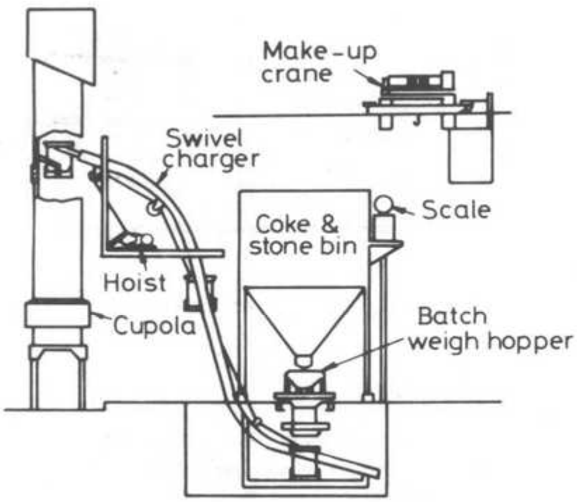


Fig.15 Make-up crane and weigh hopper used with swivel bucket charger.

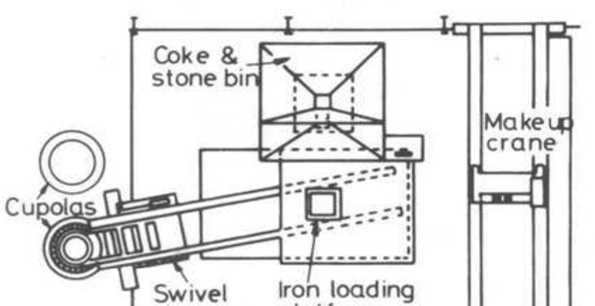


Fig.16 Make-up crane and weigh hopper used with swivel bucket charger.

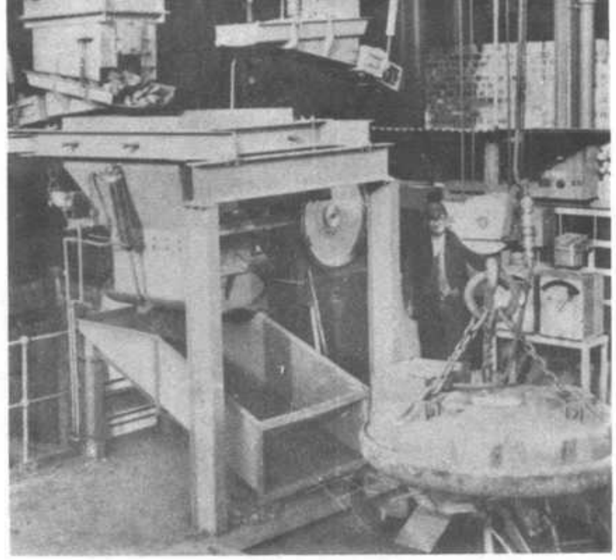


Fig.17 Make-up crane and weigh hopper used with swivel bucket charger.

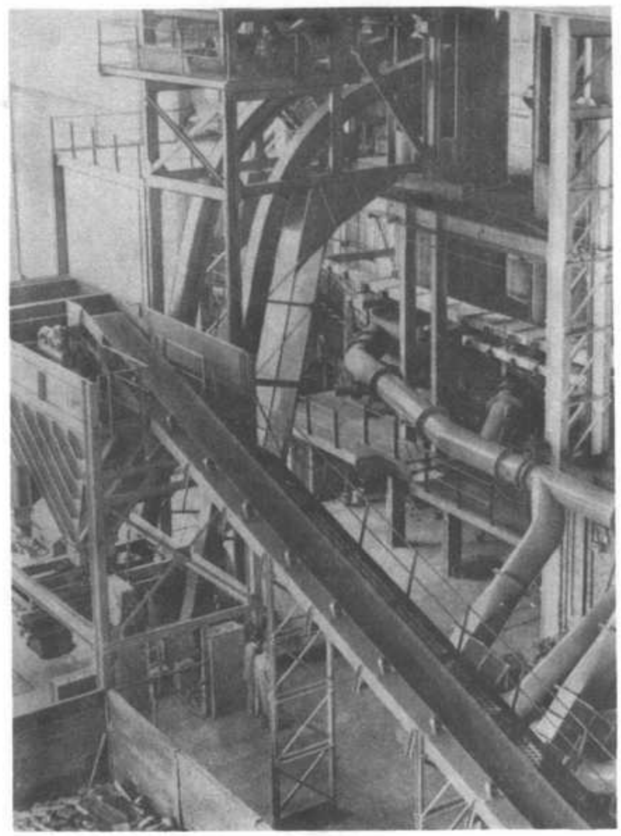


Fig.18 Inclined belt conveyor used to fill material hopper.

Automatic charging plant

A further interesting example of a charging system is illustrated in Figs. 21 & 22. In this case the whole operation is largely automatic, only one operator being required to sustain the output of 8 tonnes per hour.

The day-to-day stock bunkers for metallic materials are arranged in a semi-circle with the operator control cabin located at the mid-point of the diameter. A radial type crane, pivoted at the control cabin and equipped with a variable flux-controlled magnet, is used to make up the charges in a centrally disposed weigh hopper. The completed charge falls from the weigh hopper into a drop-bottom charging bucket

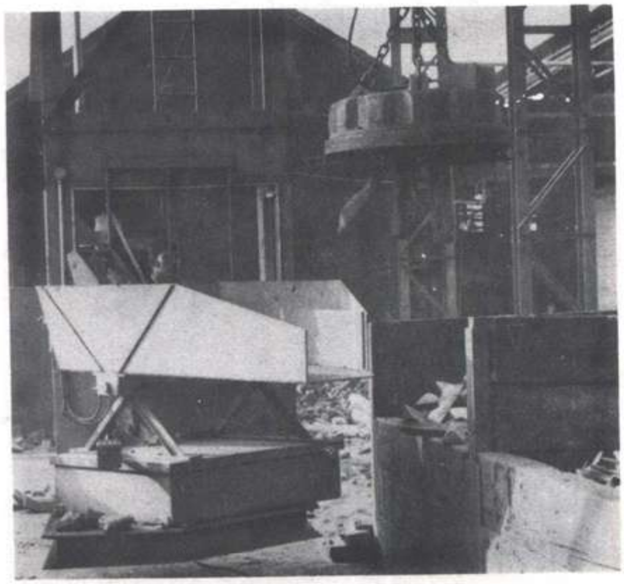


Fig.19 Magnet system of charge make-up.

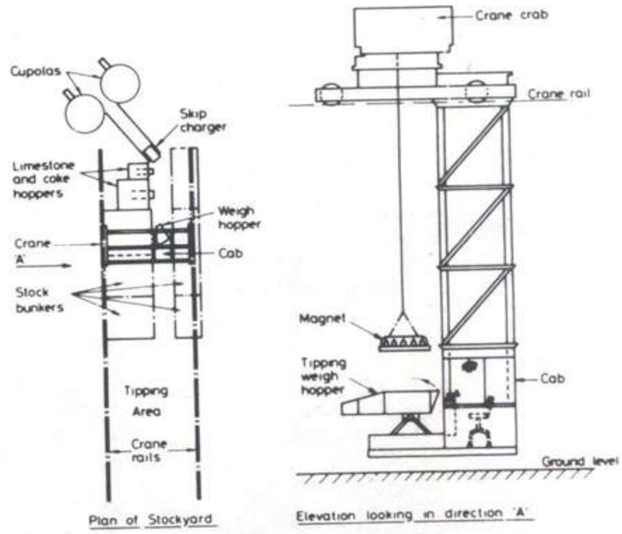


Fig.20 Layouts magnet system of charge make-up (See Fig.19).

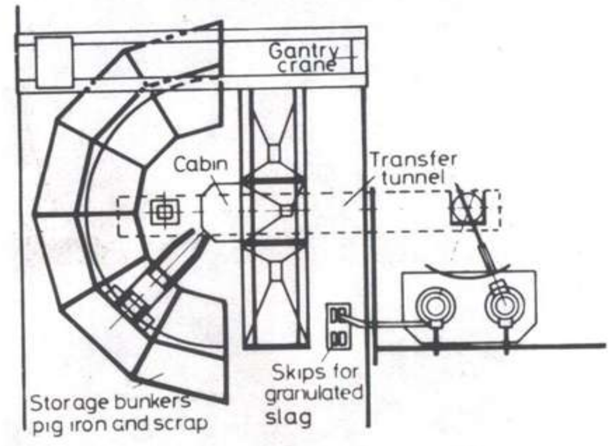


Fig.21 Automatic charging plant : plan.

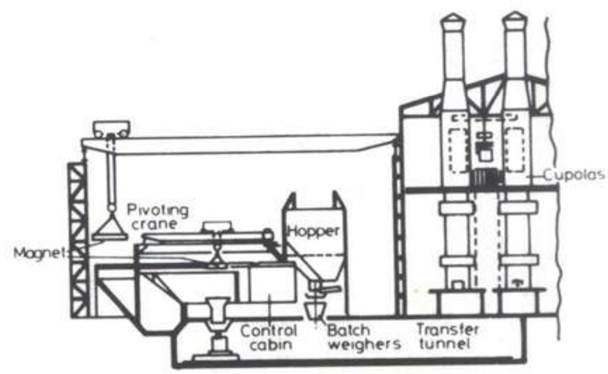


Fig.22 Automatic charging plant : elevation.

which is carried on a bogie and conveyed through a tunnel to the cupola charging hoist. This is a vertical hoist which raises the bucket to charging-sill height, where it is taken by a monorail charging beam into the cupola shaft.

Two 35-tonne capacity coke hoppers and one 45-tonne limestone hopper are provided behind the control cabin. The required quantities of coke and limestone are discharged automatically from these hoppers along vibratory chutes into a weigh hopper resting on a transfer bogie in the tunnel. In addition to the dial scales in the control cabin, each weighing mechanism is linked to a print-out system which records the weight of each type of material charged. The operator's control desk carries a mimic panel which shows the state of the charging mechanisms at any time. The coke and limestone bunkers and the day-to-day stock bunkers are replenished by means of a conventional magnet gantry crane which covers the stockyard area.

Shop-floor Controls and Equipment

The primary object of cupola control is to provide metal at the desired rate, having a suitable composition and temperature to produce satisfactory castings.

Weighing metal and coke

In simple terms there are only three variables in cupola operation: metal, coke and air. Considering metal, the analysis of the pig iron and ideally, the analysis of bought scrap, should be known. Foundry return scrap of known compositions should also be segregated. Accurate weighing of metallic charge materials is imperative if metal composition is to be controlled adequately.

The simple control exercised over the coke is weighing, and this can be carried out using the simple monorail charge reclamation system or the more sophisticated charging systems with coke and limestone weighing bunkers. The calibration of the weighing machine should be suitable for the size of the charge weighed; for example, it is no use trying to weigh to an accuracy of 1kg if the subdivisions on the scale are in 5kg units.

There is some argument that the coke charge can be more accurately measured by volume than by weight, since the moisture content of coke can vary from 0 – 12 per cent. The problems encountered with volume measurement are, however, variations in the size of coke, and that ideally the container should be filled with each coke charge. This means that if the coke charge is altered the size of the container has also to be altered.

Blast control

The air volume must be controlled, since it takes approximately one tonne of air to produce one tonne of metal. To control air supply many cupolas are fitted with windbelt pressure gauges, and unless these are used with discretion they can give rise to misleading information. The reason for this is considered in Fig. 1.

This graph shows three system-resistance curves, typical of cupola characteristics under various operating conditions. These curves are superimposed on the characteristic curve for the particular fan being employed.

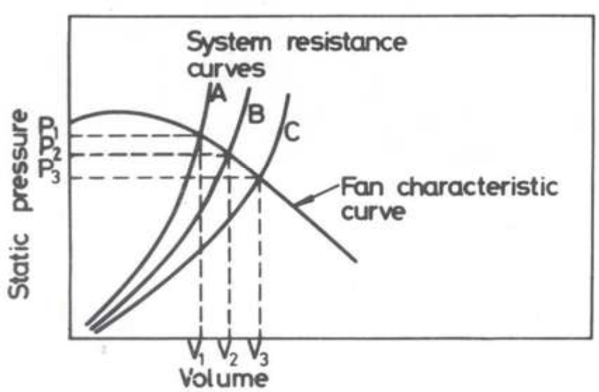
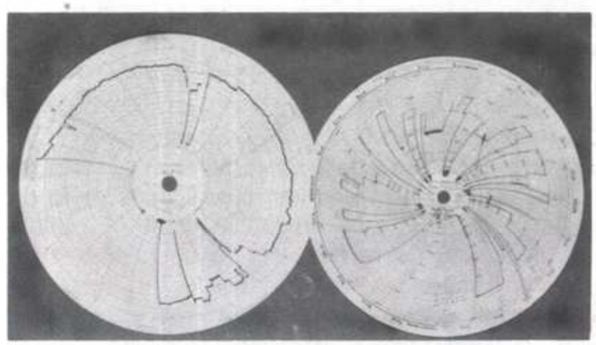


Fig. 1. Fan characteristics

Curve B, representing normal cupola operation, shows that the fan will deliver volume V_2 of air at a windbelt pressure of P_2. If, however, the cupola stack resistance increases owing to, say, blocked tuyeres, the windbelt pressure will increase and the volume delivered by the fan will decrease, as shown by Curve A. In practice, the furnace operator will notice the increase in windbelt pressure from P_2 to P_1 and will assume that the cupola is being overblown. He will therefore reduce the blast volume until pressure P_2 is restored, and this will aggravate the existing situation – since a reduction in volume will result in the cupola being underblown.



Continuous operation — Intermittent operation

Fig. 2. Volume indicator control chart.

For this reason it is better to control air supply using a volume indicator recorder, and a typical chart using such an instrument is shown in Fig. 2. It is pertinent to note that the intermittent blast operation shown on one recorder chart will lead to production of metal at a lower temperature than that when the furnace is operated continuously, as shown on the other recorder chart.

Metal temperature

Metal temperature can be measure by the conventional platinum/platinum-rhodium immersion thermocouple using either silica or graphite sheaths. Although a graphite sheath has a longer working life, the thermocouple has a slower response than when using a silica sheath. For rapid temperature determinations the expendable type of thermocouples are finding increasing application, and typical of these are the Leeds & Northrup Tectip, the Land Dipstick, and the Thermanol Diptronic (Fig. 3).

The temperature may be shown on a potentiometric indicator or a recorder. If metal temperatures are recorded they can be very easily correlated with other data in scrap analysis systems. It is sometimes necessary to have a continuous temperature record of the metal leaving a cupola, and this can be achieved by using an alumina-sheathed thermocouple as shown in Fig. 4. The disadvantages of this technique are that the sheaths are expensive, and are susceptible to thermal and mechanical shock.

It has been found that a radiation pyrometer can give accurate and reliable results provided that it is sighted onto a clean, moving metal stream and that fumes are kept away from the target area. This technique has been applied with the pyrometer positioned at a cupola tapping-box, at a cupola launder, and above a receiver spout (Fig. 5).

Fig. 3. Expendable-tip pyrometer

Control tests

(a) Chill test For a long time the chill test was the only form of shop-floor control available to the foundryman, and the difference in chilling characteristics was used as an indication of the grade of iron or the efficiency of an inoculation process. Typical fractures of wedge test pieces are shown in Fig. 6, and the mould used is shown in Fig. 7. The forced-chill test is, however, more sensitive than the wedge test, and a range of chill test pieces is shown in Fig. 8. The test piece having the greatest depth of chill has a carbon equivalent value of 3.74, and the piece showing the smallest depth of chill 4.32. The ring-type chiller and core used for producing a forced-chill test piece is shown in Fig. 9; the test piece is approximately 6 x 30 x 50 mm. The slot in the core is placed in a different position in the ring for each test, to avoid localized overheating of the chiller and a consequent decrease in chilling efficiency.

(b) Thermal analysis The major advance in shop-floor control of metal composition has been the development of thermal analysis. In this technique a sample of molten metal is cast into a mould, usually an expendable mould, and as it solidifies a cooling-curve is plotted automatically on a temperature recorder. The liquidus arrest temperature recorded can be correlated with the carbon equivalent liquidus value, which is a relation

$$\text{CEL} = \%\,\text{TC} + \frac{\%\,\text{Si}}{4} + \frac{\%\,\text{P}}{2}$$

This provides a useful guide to the composition of the iron, but does not give any indication of the individual levels of the elements involved.

However, if the sample is caused to solidify white instead of grey, by the use of a mould with a tellurium coating, the thermal analysis curve shows both the liquids and eutectic arrest temperatures (Fig.11). These two temperatures define a unique carbon level in the cast iron, and by substitution of the two values on the BCIRA Carbon Calculator an estimate of the

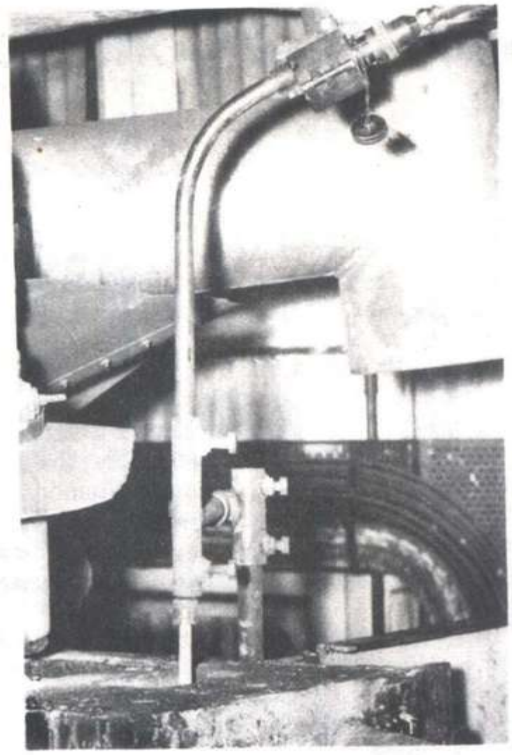


Fig. 4. Alumina-sheathed thermocouple

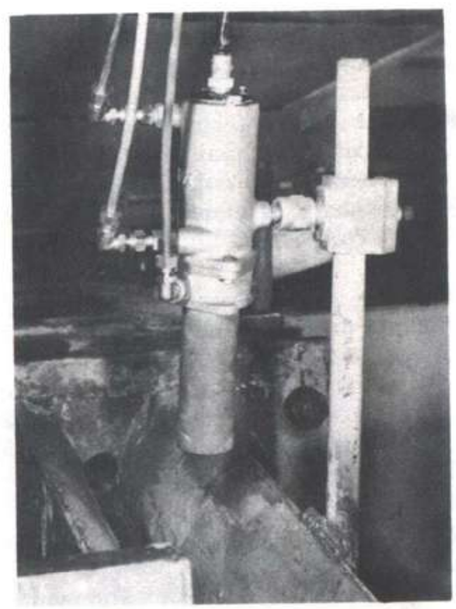


Fig. 5. Radiation pyrometer

carbon content may be obtained (Fig.12).

The technique can be used to predict carbon content to an accuracy of $\pm$ 0.05 per cent within about 2 minutes of pouring the sample, provided the composition satisfies the following requirement:

$$\%\,\text{TC} + \frac{\%\,\text{Si}}{9} + \frac{\%\,\text{P}}{3.5} = \quad < 4.05$$

and the iron is not heavily nucleated by inoculation or melting technique, and is not treated with magnesium or cerium, or heavily alloyed.

In many instances where the phosphorus content is a consistent and known value, the recorded CEL value and the calculated carbon content may be used to estimate silicon content by substitution in the equation

$$\% \text{Si} = 4(\text{CEL} - \% \text{TC}) - 2\,\text{P}\%$$

The accuracy of this technique is about ± 0.15 per cent silicon, which is adequate for control purposes.

In some cases the accuracy of the silicon estimation may be improved by producing a control graph of white eutectic temperature related to analysed silicon content (Fig.13).

The application of the thermal analysis is illustrated in Fig.14, where the grades of iron indicated by thermal analysis are compared with those anticipated from the charging procedure. This shows the usefulness of the technique in ensuring that the correct grade of iron is poured into moulds.

Silicon determinator

Another useful shop floor control is the silicon determinator, illustrated in Fig. 15. The method makes use of the electrical potential developed in a circuit of two dissimilar metals when a temperature difference is maintained between their junctions.

The cast iron sample, either a solid block or drillings, is placed in the recessed dish (cold junction of the circuit) and brought into contact with a spring-loaded heated copper probe (hot junction of the circuit) and the voltage developed is recorded. This reading is then substituted in a previously established calibration graph and the silicon content is read off (Fig.16). With the appropriate correction for interfering elements, usually manganese, an accuracy of $\pm$ 0.1 per cent silicon can be obtained within about one minute.

The condition of the sample is important; it should be either completely white or completely grey. The presence of mottle leads to decreased accuracy. Care should be taken where drillings are used, to ensure that they are not 'burnt' during preparation. The sample should be sufficient to fill the recessed dish and should be free from dust or fines. Solid samples should be ground to a consistent finish without overheating.

Conclusion

It should be emphasized, in conclusion, that controls should be instituted at shopfloor level, not based on obtaining historical results from laboratories remote from the foundry.

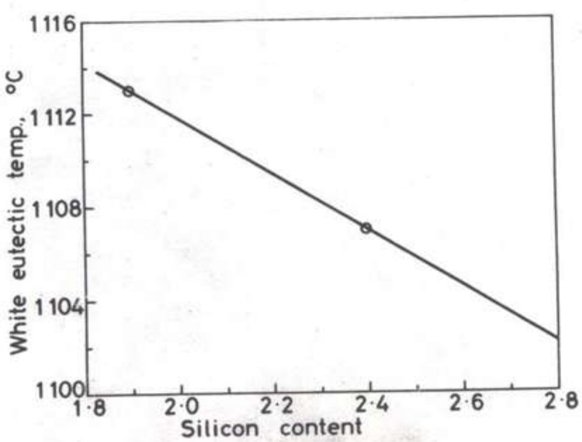


Fig.13. White eutectic temperature v. silicon content

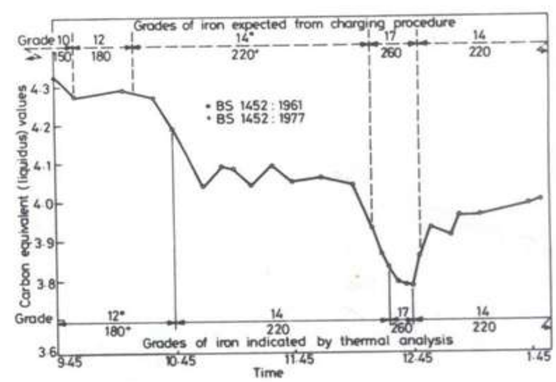


Fig.14. CEL v. grades of iron

Fig.15. Silicon determinator

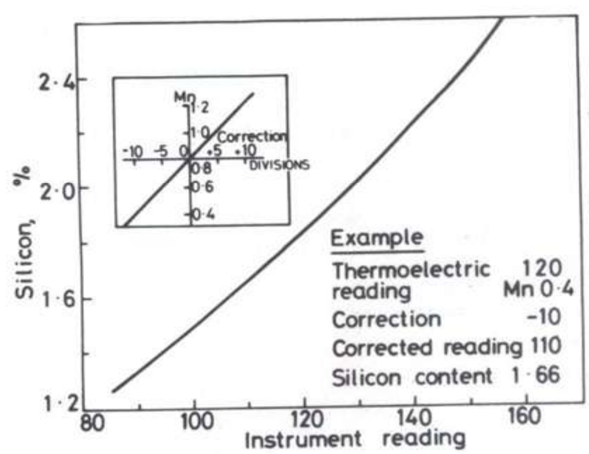


Fig.16. Silicon calibration curve

Desulphurization and Carburization of Iron

In the ironfounding industry, two of the most important metal-treatment processes to concern the foundryman are carburization and desulphurization, and there are a number of reasons why a foundry should consider carburizing or desulphurizing cupola-melted metal.

First, these two processes enable irons of a given grade or composition to be produced from cheaper raw materials. For example, if a proportion of the pig iron in the charge is replaced by cast iron scrap, desulphurization may be required, or if the pig iron is replaced by steel scrap carburization may be necessary – perhaps also with desulphurization.

Second, in the production of nodular irons, the higher the sulphur content before treatment the greater must be the quantity of nodularizing alloy added. Hence by reducing the sulphur content of the iron to a low level before treatment, substantial savings in nodularizing alloy can be achieved and defects due to the inclusion of dross in castings avoided.

Third, the ability to carburize and desulphurize iron gives the foundry flexibility, to produce several different grades of iron from the same basic cupola charge mixture.

Factors affecting the efficiencies of treatment processes

The efficiency of carburization and desulphurization depends on the following factors:

1. Agent used
2. Metal temperature
3. Metal composition
4. Degree of mixing between agent and metal.

1. Agent used The grade of carburizer has a significant effect, both on the rate of carbon solution and on the recovery of carbon. This was demonstrated when a low-carbon-equivalent iron at a temperature of 1 500 °C was tapped onto various types of carburizer (Table 1).

A high-purity graphite gave a high carbon recovery of 74 per cent, a lower-grade graphite a recovery of 48 per cent, anthracite a recovery of 40 per cent, crushed coke a recovery of 38 per cent, and coaldust a recovery of only 24 per cent.

The conclusion from these results is that the higher the purity of the carburizer, the greater the carbon recovery. Apart from the poor recoveries using the lower-purity carburizers such as coaldust, anthracite and crushed coke, these materials also generate considerable quantities of smoke and fume and give rise to excessive quantities of dross on the metal surface.

Where an efficient metal-treatment process is used, higher carbon recoveries will be obtained from the lower-grade carburizers than those shown in Table 1. However, a further point to consider in the selection of a carbonizer, particularly in nodular-iron production, is its sulphur content – since most of the

Table 1 – Type of carburizer versus carbon pick-up

Carburizer type. Size less than 6.4mm mesh	C pick-up %	C yield %
High-purity graphite	0.37	74
Grade 2 carburizer	0.24	48
Grade A electrode scrap	0.24	48
Anthracite	0.20	40
Coke dust – dried	0.19	38
Kish graphite	0.19	38
Coaldust	0.12	24

sulphur in a carburizer is absorbed by the metal. This is important in electric melting where large carburizing additions are made. For example, a carburizer containing 1.0 per cent sulphur will raise the sulphur content of the metal by 0.01 per cent for each 1.0 per cent increase in carbon content. In nodular-iron production this will increased nodularizing costs, and a low-sulphur carburizer is therefore desirable.

No matter what carburizer is used, the solution of carbon in iron is an endothermic process and metal temperature is reduced by at least 8 °C for every 0.1 per cent increase in carbon content. Therefore, if it is intended to tap metal at 3.0 per cent carbon from an acid cupola and carburize to 3.8 per cent by an external treatment, the metal temperature will be decreased by about 65 °C in addition to normal radiation losses.

The most common agent used for desulphurizing molten cast iron is calcium carbide. Sodium carbonate, more commonly referred to as soda ash, and burnt lime are also used to a limited extent. Generally, no definite rules can be made on the choice of desulphurizing agent, since this depends on several factors, the most important of which is the process to be adopted. The important factors to be considered in the selection of desulphurizing agents are shown in Table 2.

Apart from the many disadvantages of soda ash it is, perhaps, the only desulphurizing agent that can produce a significant degree of desulphurization by simply tapping onto the agent in a ladle, burnt lime and calcium carbide being ineffective without a high degree of metal turbulence. Typical results would be a reduction in sulphur content from 0.10 per cent to 0.04 per cent from a 1.0 per cent addition of soda ash. Soda ash would therefore be useful for those small producers of nodular iron who operate an acid cupola but have installed no elaborate treatment process for efficiently removing sulphur.

Where an efficient metal-treatment process is installed, and metal is to be carburized and desulphurized simultaneously, it is possible to use a cheaper carburizer having a higher sulphur content than that used in electric-furnace carburization, the desulphurizing agent preventing the sulphur of the carburizing agent from entering solution in the iron.

Table 2 – Characteristics of sodium carbonate and calcium carbide as desulphurizers

Factor	Sodium carbonate	Calcium carbide
Slag produced	Very fluid. Complete removal difficult.	Granular. Easier to remove.
Fume	Appreciable extraction essential.	Not troublesome.
Silicon loss	Up to 0.3%	Very small.
Temperature loss	Large	Smaller since reaction exothermic.
Refractories	Preferably basic.	Immaterial.
Approximate price per tonne	£105	£180
Storage requirements	Must be kept dry.	To comply with statutory regulations.

2. Metal temperature The effect of metal temperature on carbon recovery was demonstrated by tapping iron onto graphite placed in a ladle.

Fig. 1 shows that the highest efficiency of carburization was obtained with the highest metal temperature. For a graphite addition of 1 per cent, the carbon recovery at 1 500 °C was four times that obtained at a temperature of 1 300 °C.

Likewise, desulphurization with calcium carbide is also improved by treating at high metal temperatures. Fig.2 shows the quantity of sulphur removed when metal was treated with 1 per cent of commercial calcium carbide and agitated with 0.23m^3/min (8 ft^3/min) of air for 1 minute in a porous-plug ladle. The efficiency of desulphurization at a treatment temperature of 1 320 °C was approximately half that achieved at a temperature of 1 550 °C.
Sodium carbonate,

achieved at a temperature of 1 550 °C.

Sodium carbonate, however, is not so dependent upon metal temperature, and has only slightly less efficiency at lower metal temperatures.

3. Metal composition The efficiency of carburization depends on the carbon-equivalent of the iron. The results of tapping irons of different carbon-equivalent values onto various quantities of graphite at a metal temperature of 1 500 °C are shown in Fig.3. This demonstrates that the solution of carbon is retarded in irons of high carbon-equivalent value and at a graphite addition of 1 per cent, carburization efficiency for a 4.2 per cent carbon-equivalent iron is only approximately 15 per cent, whereas that of a 3.1 per cent carbon-equivalent is almost 80 per cent.

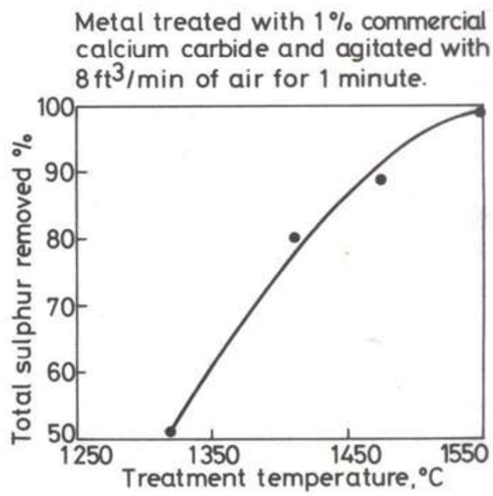


Fig. 2. Effect of metal temperature on desulphurization.

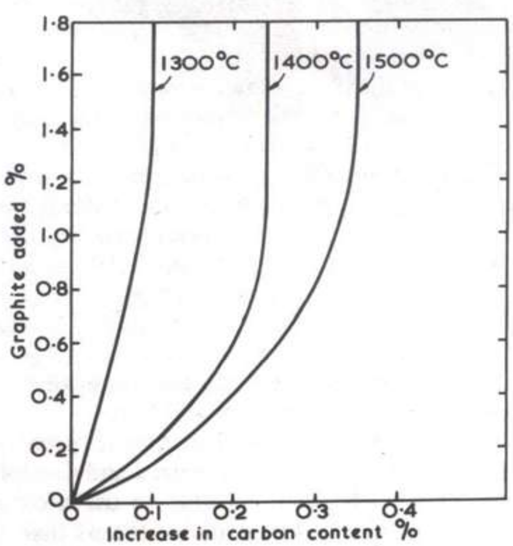


Fig. 1 Effect of metal temperature on carbon pick-up.

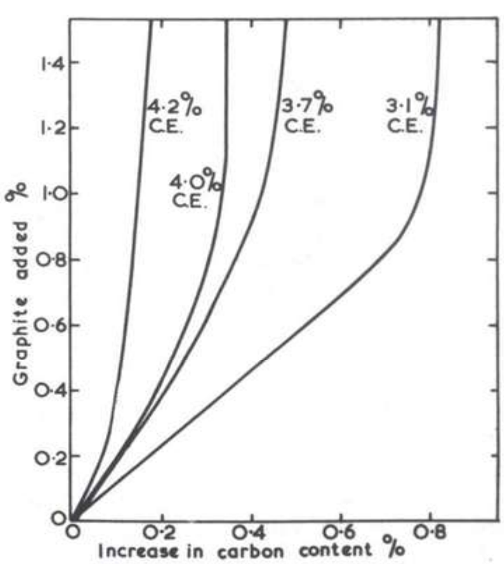


Fig. 3. Effect of metal composition on carbon pick-up.

Similarly, with desulphurization the efficiency is dependent upon the level of sulphur before treatment and the final sulphur content required. The higher the sulphur content of the iron, the more easily is a given reduction obtained.

4. Degree of mixing Little difficulty is generally experienced in making alloy additions to molten cast iron for the modification of chemical composition, since materials are generally in metallic form which readily goes into solution. This is not the case, however, with carburization and desulphurization as both carburizer and desulphurizing agents are light, non-metallic materials. If added to a ladle of molten metal, these materials will float on the metal surface and very little reaction will occur.

The importance of a mixing action is demonstrated in Fig.4. This gives the results obtained when iron was tapped onto graphite and sodium carbonate followed by mixing of the metal and the reactants with compressed air blown through a submerged graphite lance. A considerable degree of carburization and desulphurization was achieved by tapping onto the reactants but, by merely mixing the contents of the ladle for three minutes with induced turbulence, further significant increases in carburization and desulphurization were achieved.

Metal treatment processes

A number of processes have been developed to enable the ironfounder to carburize and desulphurize cast iron to desired levels with consistency. Without exception, all these processes involve some method of artificially creating turbulence within the metal to achieve contact between the reactant and the metal and thus increase both reaction rates and efficiencies.

Porous-plug ladle

The porous-plug ladle offers a low-cost technique for agitating the metal and the desulphurizing and carburizing agents, as shown in Fig. 5. The introduction of the gas through the porous plug in the bottom of the ladle creates a vigorous stirring action, and use is made of this to improve the efficiency

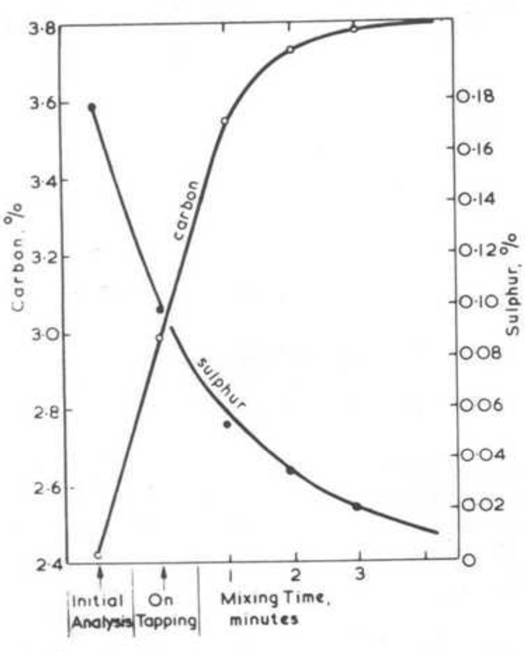


Fig. 4. Effect of mixing on carburization and desulphurization.

of metallurgical reactions. Initially the process was only used to a limited extent owing to the poor design of the plugs, which were rigidly fixed in the ladle bottom and required considerable time and labour for their frequent replacement. However, some years ago a new design was introduced that enabled the porous plug to be extracted simply and rapidly and replaced by a new porous plug. The design of this plug is shown in Fig.6, and is now widely used by the industry when carburizing and desulphurizing molten iron. This design allows a worn plug to be extracted from a hot ladle and replaced by a new plug in about four minutes.

The operating procedure with the porous plug is simple. The ladle is filled with molten metal to about two-thirds of its capacity. It is preferable to provide the ladle with a refractory cover, to reduce metal spillage and temperature loss. The desulphurizing and carburizing additions may be made prior to, or at any time during, the filling of the ladle. Compressed air or nitrogen at a flow rate of approximately 0.14-0.23m^3/min (5-8 ft^3/min) is passed through the plug, and a treatment time of between 2 and 4 minutes is normally required to complete the reaction. Fig.7 shows a porous-plug ladle in operation, soda ash in this case being used to desulphurize the metal – with consequent production of a large quantity of white fume. If calcium carbide is used there is very little fume.

When treating cupola-melted metal in a porous-plug ladle it is normal to tap continuously onto the desulphurizing and carburizing agents placed in the ladle, and to commence blowing once the ladle is about a third full, so that treatment is almost complete by the time the ladle has filled. This method reduces standing time, and hence temperature loss.

The porous plug is normally used for about ten treatments before replacement. The use of nitrogen gas in place of compressed air increases the life of the plug.

Using a porous-plug ladle it is possible to carburize to desired levels at a recovery of 80-90 per cent and sulphur content can be reduced to 0.01 per cent or lower from 0.1 per cent with an addition of 1 per cent of calcium carbide. It is unusual for more than 10 tonnes of metal to be treated in a single treatment with the porous plug, but in view of the short treatment time it is generally not necessary to treat large quantities. A typical duplexing installation using a porous-plug ladle is shown in Fig.8.

In recent years the use of the porous plug has been extended from the ladle treatment process to a continuous treatment process, shown in Fig.9, operating in conjunction with an acid cupola. A number of foundries operate this sytem for reducing sulphur levels in grey iron production, and it has also been demonstrated that this system could be used for the production of iron of low sulphur content for nodular iron production.

Shaking-ladle

Another important process for the carburization and desulphurization of iron is the shaking-ladle, developed about twenty years ago in Sweden. Basically the process consists of a specially designed ladle mounted in a framework and subjected to a rotary motion. A typical illustration is shown in Fig.10. The motion imparted to the ladle and its contents is the same as that which takes place when a cup of tea is shaken to dissolve the last bit of sugar in the bottom of the cup.

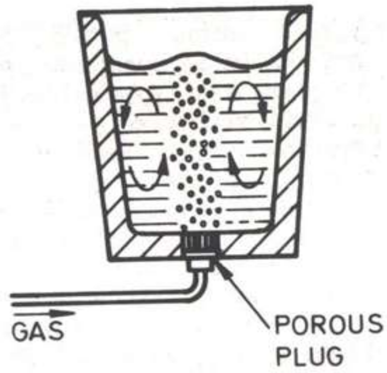


Fig. 5. Porous-plug agitation.

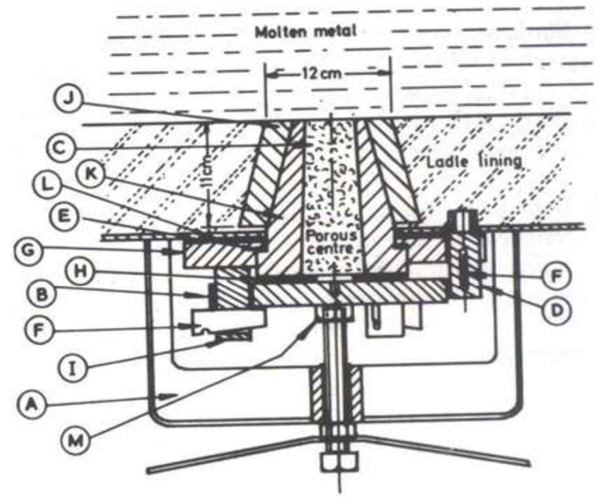


Fig. 6. Cross section of porous-plug assembly with extractor.

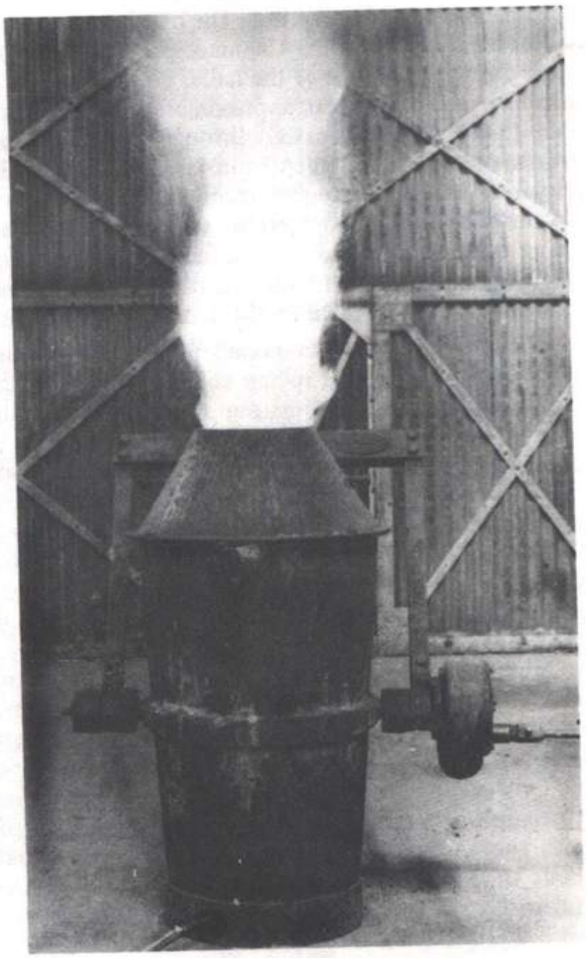


Fig. 7. Porous-plug desulphurization with soda ash.

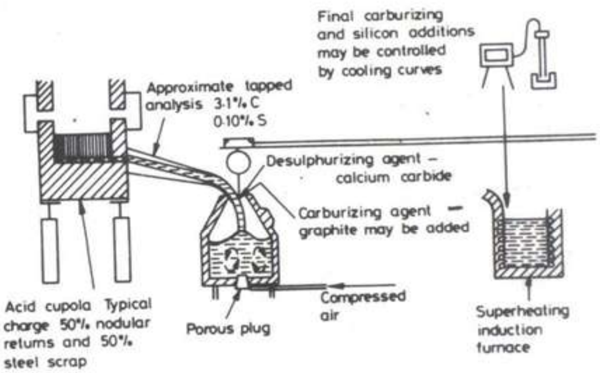


Fig. 8. Duplexing process for nodular iron production.

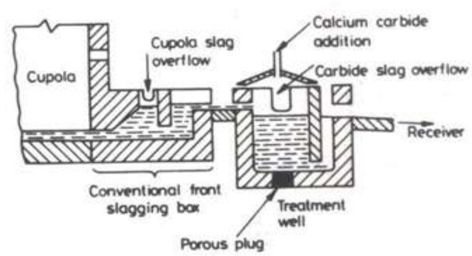


Fig. 9. Arrangement for continuous porous-plug treatment.

With the shaking-ladle the speed of rotation and degree of eccentricity are critical, to obtain the correct wave motion in the ladle, shown in Fig.11.

A high degree of desulphurization can be obtained with the shaking-ladle, and Fig.12 shows that about 1 per cent of carbide should be used to reduce sulphur to the low level required in nodular iron production. A shaking-time of 4 - 6 minutes is required for the best results. Besides desulphurization, other additions may also be made in the shaking-ladle to adjust metal composition; and operational results using a 5-tonne shaking-ladle are shown in Fig.13.

Although both the porous-plug ladle and the shaking-ladle can achieve high degrees of carburization and desulphurization suitable for nodular iron production, the shaking-ladle involves the use of equipment of high cost and, therefore, finds application in large foundries producing nodular iron, ingot moulds and refined irons. The porous-plug is a low-cost process and is favoured by small to medium-size foundries.

Other treatment processes

The shaking-ladle and porous-plug ladle are the two main processes used in the UK for the carburization and desulphurization of molten iron produced from a cupola. There are, however, other methods which may be used either to carburize or desulphurize cast iron but, generally, they are not widely used in the UK.

As already mentioned, metal may be tapped

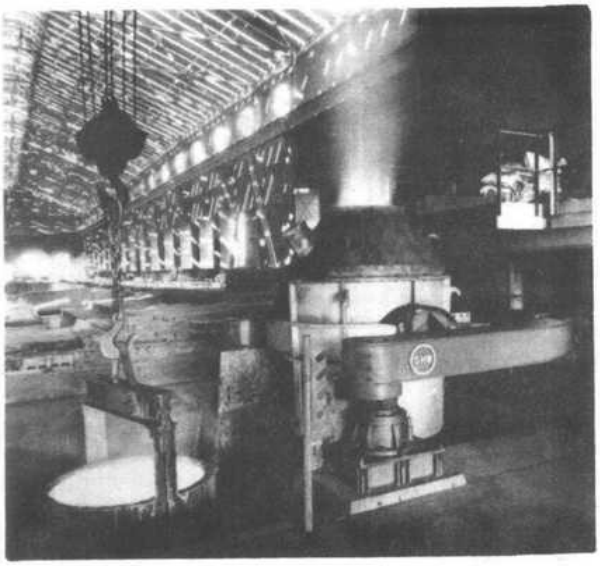


Fig.10. Shaking-ladle installation.

Fig.11. Wave motion in shaking-ladle.

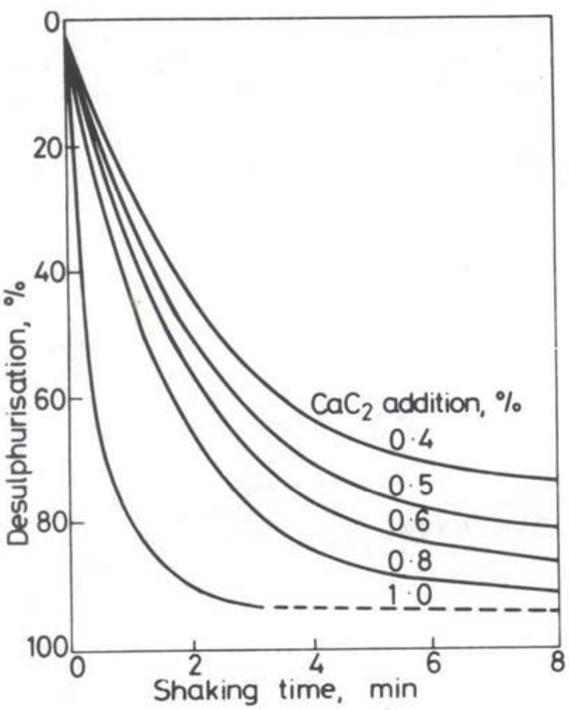


Fig.12. Degree of desulphurization using calcium carbide in shaking-ladle.

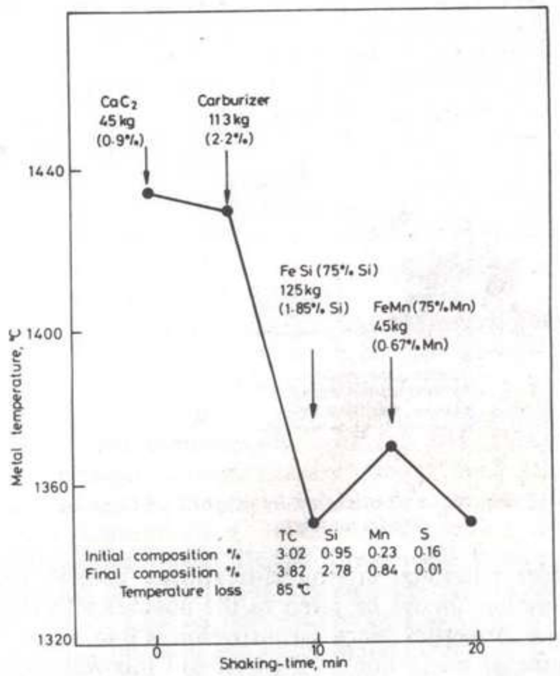


Fig.13. Operational results for 5-tonne shaking ladle.

direct onto the reagent placed in the bottom of the ladle, and with soda ash a significant degree of desulphurization can be achieved. The efficiency of tapping onto reagents can be improved by double-ladling, in which the contents of one ladle containing the reagents are poured into a second ladle to improve agitation and efficiency of reaction. The use of double-ladling is, however, restricted by serious loss in metal temperature, by the time and effort involved and the variable results obtained.

The injection technique has also been used as a batch or continuous process for carburizing and desulphurizing molten iron. In this method a powder dispenser is used to meter the desulphurizing or carburizing agents, which are pneumatically conveyed along flexible hose to the injection lance immersed in the metal. A typical powder dispenser for this purpose is shown in Fig.14. Although the process has been used in a specially designed treatment vessel placed at the end of the cupola launder, it is best suited to the batch carburization of iron in holding-furnaces such as the electric channel furnace, rotary furnace and the direct-arc furnace. The use of this process in a rotary furnace is shown in Fig.15. The efficiencies of carburization are generally about 60 per cent in the rotary furnace, 90 per cent in the channel furnace, and 70 per cent in direct-arc furnaces.

As an alternative to injection, the carburizing and desulphurizing additions may be thrown onto the metal surface and the metal mixed with the additions by blowing a gas into the metal through a submerged lance.

Cupola-melted metal may be carburized to the extent of 0.1 or 0.2 per cent by adding graphite to the cupola launder, as shown in Fig.16. In one plant a recovery of 70 per cent was achieved. However,

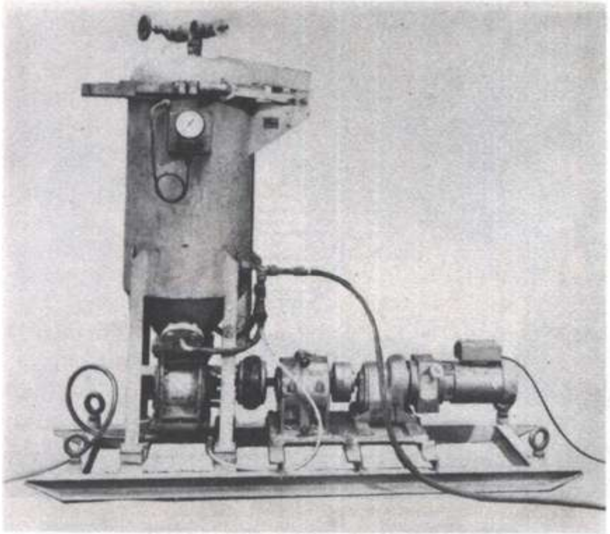


Fig.14. Powder dispenser.

Fig.15. Carburization in a rotary furnace by graphite injection.

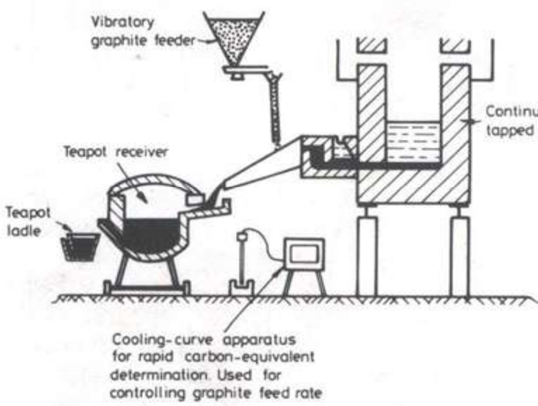


Fig.16. Carburization by graphite addition to cupola launder.

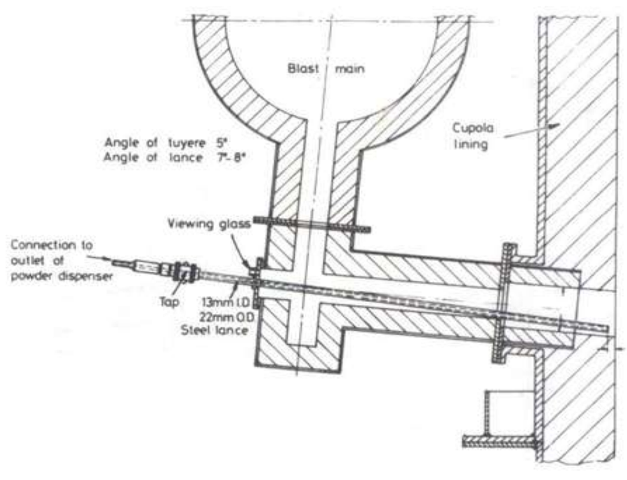


Fig.17. Tuyere-injection lance assembly.

before adopting any metal-treatment process consideration should be given to the possible effects on metal properties, since carburization will increase the degree of nucleation of the iron and this will render the castings more prone to shrinkage defects. If the inoculating effect of the carburizer is to be minimized, the technique of injecting graphite through the tuyeres of the cupola can be adopted; a typical system is shown in Fig.17. The recovery of graphite using this technique is of the order of 20 - 30 per cent and this technique is, therefore, not as economical as other techniques. There is the advantage, however, that the endothermic solution of carbon, which normally results in a loss of metal temperature on carburization, is compensated for in the cupola and there is no decrease in metal temperature using this process.

Foundry Coke and Fluxes

The industry uses about 600 000 tonnes of foundry coke each year. Foundrymen are accustomed to receiving this, complaining about its quality from time to time, (mostly without good supporting evidence) and in general know little about how it is made and what are its important properties.

The manufacture of foundry coke, how its properties can be determined, what its important properties are, and its specification are discussed.

How is foundry coke made?

Figs. 1 & 2 show a schematic representation of a coke-oven plant.

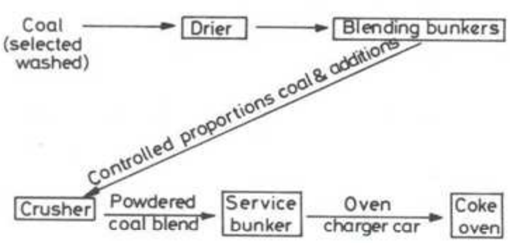


Fig. 1. Coke production - coal preparation.

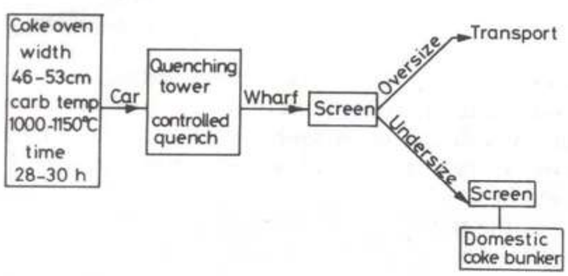


Fig. 2. Coke production - coke handling.

Selected coals, washed to remove impurities, are obtained and dried and stored in blending bunkers. Controlled proportions of the coals and additions, such as coke breeze, to make the required blend are fed to a crusher and the powdered coal fed into a service bunker. The crushed coal is transferred to an oven charger car and charged into the coke oven.

The coke oven of 46 - 53 cm width is rectangular and externally heated. After charging, the heat flows from the oven walls to the centre of the oven and the coal goes progressively through carbonization changes until the temperature of the centre of the charge is similar to that at the walls. After standing for 2 - 3 hours at this temperature of 1 000 - 1 150 °C the coke is discharged. A normal carbonizing time is of the order of 28 - 30 hours. An exception to this practice is the Coedely foundry coke plant, where the coal is charged into a 53 cm width oven as a briquette and a carbonization time of 39 - 40 hours is used.

During this process the coke oven gas is passed to the by-product plant for the production of tar, benzole and chemicals.

The coke is discharged into a coke car and quenched in a controlled manner before discharge onto the wharf. From the wharf the coke is conveyed to the screens and the undersize (usually less than 64mm or 76mm) is screened into several size ranges for the domestic market. The oversize coke passes along a belt, where there is visual inspection to remove black ends, and then to a boom loader for loading into lorries or rail wagons for transport to the foundry.

Coals for foundry coke production and location of coke ovens

Selected coals have to be used for foundry coke production. Coals are classified by coal rank code numbers and the prime coking coals or medium-volatile coals necessary for foundry coke production, having a volatile matter of 19.6 - 32.0 per cent, are known as 301 rank coals. For foundry coke the coal must also have a low sulphur content.

The principal deposits of these coals were found in South Wales and Durham and, for this reason, foundry coke plants were established in these areas. However, particularly in Durham, the 301 rank coals are diminishing in quantity and are becoming harder to mine. Consquently, in the near future, although foundry coke will continue to be produced in the traditional areas, it will be made by blending coals other than 301 rank coals which are in plentiful supply from economically operated pits. This has been made possible by extensive research and development work by the National Coal Board and the British Carbonization Research Association with collaboration from BCIRA. It is reasonable to predict, therefore, that there are adequate reserves of coals suitable for foundry coke production, and the principal future problem will be in finding labour to mine this coal. It can also be expected that foundry coke will continue to be produced in South Wales at the Cwm and Coedely coking plants and in Durham at the Norwood and Derwenthaugh coking plants, with the Lambton plant being available if required. These plants are operated by National Smokeless Fuels Ltd, a subsidiary of the National Coal Board. Additionally, it is expected that foundry coke will be produced as a by-product at the Bedwas plant in South Wales, owned by British Benzol and Coal Distillation Ltd. This plant uses Welsh coking coals obtained from the National Coal Board.

Tests to determine coke quality

The coke having been obtained, next to be considered is whether there are tests available to determine its quality and variations from any set standard, or whether the only test available to the foundryman is to use the coke in the cupola and assess its suitability in the light of the cupola performance obtained.

Sampling

Before considering tests available, it should be emphasized that it is essential to obtain a representative sample of the coke; no dispute about coke quality can be substantiated without a representative sample.

British Standard 1017: Part 2 – The Sampling of Coal & Coke, describes several methods of sampling coke to obtain a representative sample.

Most of these are unsuitable in their application to single consignments of foundry coke, and involve an onerous and expensive task. Where a foundry requires a coke sample it is suggested that an individual consignment should be segregated and, while it is being used, a forkful of coke should be taken at frequent intervals and these should be retained to form a bulk sample. To avoid breakage or degredation of the coke during subsequent handling, the sample should be stored in a dustbin.

Size analysis

It will be shown later that the most important property of foundry coke is its size. It is necessary, therefore, to know how to determine this using a representative sample. The method of determining size is described in British Standards 1293 and 2074. This involves the following steps:

1. Dry the sample to a moisture content not exceeding 8 per cent, and weigh the sample (drying is rarely necessary).
2. Size the coke using standard square-aperture steel plate/sieve plates. The operation is carried out by hand-placing, and a piece of coke is considered to be retained on a particular sieve if it will not pass through the sieve in any position.
3. When the complete sample has been sized the amount of coke retained on each sieve is weighed.
4. The results are presented in the form in Table 1.

From these it is also possible to derive the mean or average size of the coke, using a well-established formula employed in the coking industry. In the example shown, the mean size can be calculated to be 114mm (4.47in).

Testing – analytical

The laboratory testing of foundry coke involves moisture, ash, volatile matter and sulphur. There are standard methods for the determination of these in British Standard 1016.

With respect to moisture content, British Standard 1016: Part 4 describes the method of determining this in a laboratory sample and thus permits calculation of the fixed carbon content of the coke. Part 2 of the standard describes the method of determining the total moisture content of the coke, i.e. the moisture for which the foundry pays. The standard method requires the coke to be crushes rapidly in a jaw crusher to 6mm (0.5in) size, and to be dried in an oven having air circulation.

This involves special equipment, and at BCIRA it has been found that a satisfactory assessment of total moisture content can be obtained by placing a single layer of a weighed amount of coke on the bottom of a basket consisting of a steel-plate bottom and wire-mesh sides. This is dried overnight at a temperature of 93 °C, and the basket reweighed to determine the loss of moisture.

Testing – physical

The standard tests used to determine the impact resistance and the abrasion resistance of coke are the shatter and micum tests described in Part 13 of British Standard 1016.

Shatter test (Fig. 3) In this test coke over 51mm (2 in) is subjected to a standardized dropping procedure: the resistance of the coke to breakage is measured by the percentage of the coke which remains on different apertures after the test.

Twenty-five kg of coke prepared in the same size grading of the bulk sample is used. It is placed in the box, dropped and returned to the box until four drops have been completed. After the fourth drop the coke is sized and the total amount retained on a 2 in square mesh sieve expressed as the 2 in shatter index and that on a 1½ in sieve as the 1½ in shatter index.

Micum test (Fig. 4) This is an empirical test intended to simulate the effect both of dropping pieces of coke and of rubbing them against each other or against a hard surface as, for example, during transport or handling.

In the test 25 kg of a correctly proportioned sample of the coke is rotated at a constant speed for 100 revolutions of the drum and the percentage of coke remaining on a 40 mm round aperture sieve usually determined.

Table 1 Size analysis of foundry coke

Size	Per cent	Retained on screen, cumulative per cent	
+ 7 in	3.7	3.7	A
+ 6 in	16.7	20.4	B
+ 5 in	12.5	32.9	C
+ 4 in	23.5	56.4	D
+ 3½ in	15.2	71.6	E
+ 3 in	12.8	84.4	F
+ 2½ in	9.7	94.1	G
+ 2 in	4.0	98.1	H
+ 1½ in	0.7	98.8	I
+ 1 in	0.2	99.0	J
+ ½ in	0.3	99.3	K

Mean size (in) = 4.47

$$\text{Mean size (in)} = \frac{A + B + C + \tfrac{3}{4}D + \tfrac{1}{2}(E + F + G + H + I + J + K) + 25}{100}$$

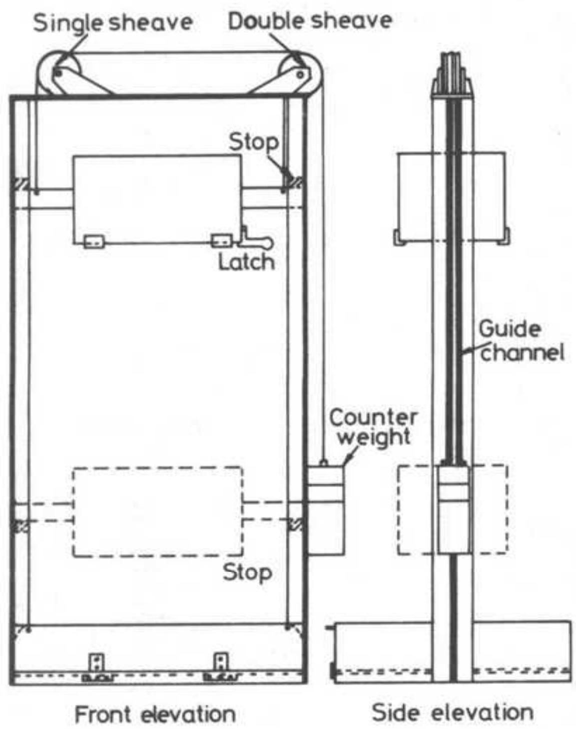


Fig. 3. Shatter test apparatus.

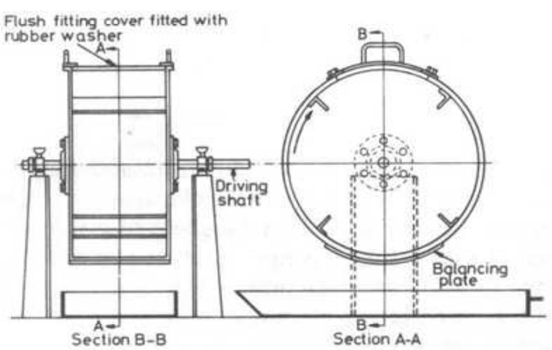


Fig. 4. Micum test - the half-micum drum.

That remaining on the 40 mm sieve (the M40 index) gives an indication of the strength of the coke and the index increases with increasing coke strength.

That passing through the 10 mm sieve (the M10 index) gives an indication of the resistance of the coke to abrasion. The lower this index, the more resistant is the coke to abrasion.

Specification of foundry coke

With the knowledge that standard tests are available to determine the properties of foundry coke, it would seem possible to specify these properties so that foundries can be sure that the coke received can be guaranteed to give good consistent cupola performance.

A specification for foundry coke must involve some items such as ash, volatile matter, fixed carbon and sulphur, relatively easy to define and understand.

Ash content Ash content is determined by the ash content of the coal used and how far it is possible to go economically in removing impurities from the coal by washing. The producer has, therefore, some control over the ash content of the coke.

High ash content is undesirable, as it reduces the fixed-carbon content of the coke and, therefore, the calorific value and it generates a greater volume of slag in the cupola – which entails an increased heat requirement. It has also been known for many years that the rate of solution of coke carbon in molten iron can be related to the ash content of the coke.

More recently, another point concerning the nature of the ash in foundry coke and its relation to carbon pick-up has been found, following work by BCIRA in collaboration with National Smokeless Fuels Ltd. This was initiated by complaints at the end of 1972, and during 1973, that there had been a deterioration in the amount of carbon pick-up obtained during cupola-melting using Welsh foundry cokes. Examination of records from two foundries confirmed this, and showed that the deterioration could be related to a progressive increase in the ash content of the cokes. The possibility existed also that the poorer carbon pick-up had not been due entirely to the higher ash contents, since some other unmeasured and un-recorded property of the cokes could have altered as a result of variation in the types and proportions of coals used in the oven blend.

To investigate this, BCIRA developed a laboratory test to detect the rate of solution of coke carbon in iron. The test results have shown clearly that ash content is an important factor influencing carbon pick-up.

It has also been found that, if the coke is heated to 900 °C in an oxidizing atmosphere in a muffle furnace until it is completely burnt, the resulting ash consists, after screening through a 100-mesh screen, of about 1 to 2 per cent of coarse gritty particles and the remainder very fine material. It seems likely that the fine ash is derived from mineral matter in the coal impossible to remove by any washing process, and the coarse ash from bands or partings in the coal seams some of which can be removed by washing the coal.

Further investigation has shown that carbon pick-up is determined predominantly by the fine or inherent ash content of the coke, and is inversely proportional to the fine-ash content. Carbon pick-up is also dependent on the fusibility of the fine ash contained in the coke. The mechanism by which the fine ash inhibits carbon pick-up seems to be that it forms a barrier on the coke surface, thus restricting the amount of carbon available to the metal.

Since both the fine-ash content of the coke and its composition are determined by the coals used in the production of the coke, it seems that if foundries are to receive cokes giving consistency of carbon pick-up the coke producer will have to exercise care in the selection of coking coals.

Volatile matter Volatile matter is also undesirable, since it reduces the fixed-carbon content of the coke. It can also indicate incomplete carbonization, which may give coke having poor properties. The producer also has some control over the volatile-matter content of the coke.

Fixed carbon Fixed carbon is calculated by taking the sum of the moisture content of the analysis sample and the ash and volatile matter from 100. The carbon content of the coke is important, since this determines the calorific value of the coke. In other words, the higher the carbon content the more value is being obtained for money. Obviously, therefore, the higher the carbon content the better.

Sulphur Sulphur is well known as an unwanted element in any type of cast iron and thus, the lower the sulphur content of the coke, the better. Unfortunately, the sulphur content of the coke depends on the sulphur content of the feedstock coal available, and there is no method known to remove sulphur from coal. Furthermore, deposits of low-sulphur coal are being worked out or becoming uneconomic to work. The industry must accept, therefore, that the sulphur content of foundry coke is likely to increase, and can only accept the assurance of the coke producer that the lowest sulphur-content coals available will be made available for foundry-coke production.

Moisture Moisture in the coke when dispatched from the coke oven is obviously undesirable, since it is an expensive way to buy water and it reduces the amount of carbon available in the coke. It is necessary for the coke to contain some moisture, to avoid fires on conveyor belts and in lorries and waggons, and this must be recognized by the industry. To ensure compensation for excessive moisture, it is advisable to check the weight of coke received against that dispatched and, when appropriate, to claim for adverse differences in weight.

Size It will be apparent that it should be possible to specify these coke properties to ensure the coke producer does not deviate from them. However, all of these properties are of less importance than coke size – which directly affects coke consumption, melting rate and metal temperature. The principal cause for complaint by the industry in recent years has been the supply of small coke, or coke which breaks down to small size by the time it reaches the cupola melting zone. Why is coke size important? This is illustrated in Fig.5, which shows that:

1. With coke of less than 75mm (3 in) size the metal temperature is decreased when using a constant coke charge. This applies to all sizes of cupolas.
2. When using 64mm (2.5 in) coke the metal temperature is 25 °C lower than when using 90mm (3.5 in) coke.
3. To achieve the same metal temperature using 64mm coke as with 90mm coke about 2.5 per cent additional coke will be required. At constant blast rate this will reduce the melting rate by about 20 per cent.

A further effect of reduced coke size, shown in Fig.6, is to increase the blast pressure required to deliver a given volume of air to the cupola. In many cases the cupola blowing equipment will not be able to provide the extra pressure to maintain the blast rate when using small coke, and thus the air delivered will decrease – with further adverse effect on the melting rate.

It is evident from this information that above a size of about 90mm (3.5 in), increasing coke size has no further beneficial effect. (Recent work at BCIRA using large-size American coke and specially produced large British coke has confirmed these conclusions, particularly for large-diameter cupolas). This is probably due to the fact that large pieces of coke tend to be fissured, and such pieces break easily during handling, charging and inside the cupola under impact of the charges. Large, heavily fissured coke

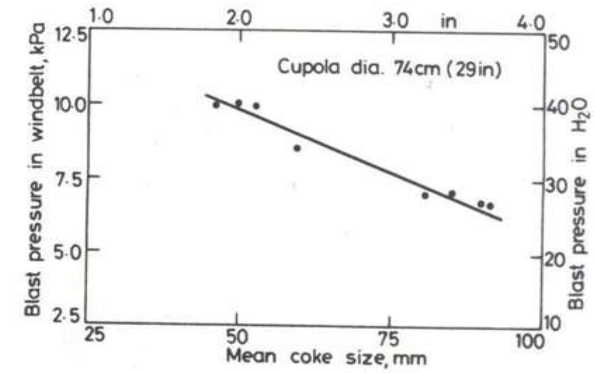


Fig. 5. Effect of coke size on metal temperature.

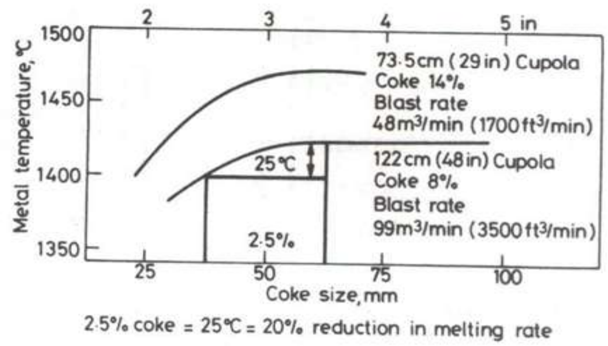


Fig. 6. Effect of coke size on blast pressure.

can, therefore, lead to deterioration in furnace performance.

To achieve optimum cupola performance it would seem reasonable to suggest that coke size at the coke oven on dispatch should be adequate to give, at the cupola, furnace coke having a mean size greater than 90mm (3.5 in) with not more than 4 per cent less than 50mm (2 in). It is thought that, based on cokes at present being supplied and used with little complaint by foundries, an appropriate minimum mean size at the coke oven will be 107mm (4.2 in) for some plants and 102mm (4.0 in) for others.

It is also recognized that in any specification there must be a test to measure the degradation of the coke between the point of dispatch and the cupola furnace. It might be thought simple to obtain this information, using the shatter and micum tests. There is, however, some doubt whether these tests provide the information needed, although there is some indication from German work that larger micum indices than the conventional M_{10} or M_{40}, such as M_{80} or M_{100}, might provide more reliable information about coke quality. It is also possible that measurement of the mean coke size after the shatter test could provide this information.

Foundry coke specifications

In recent years there has been considerable discussion between representatives of the industry (represented by CFA) and National Smokeless Fuels Ltd, and specifications for foundry coke supplied by NSF Ltd have been agreed. These are given in Table 2.

These specifications are intended to represent coke properties at the point of dispatch at the coke oven. They recognize what is achievable with feedstock coals available, and will be reviewed from time to time in relation to the availability of coals and improvements

Table 2 Foundry coke specifications

SPECIFICATIONS					
Coke plant	Cwm	Coedely	Norwood	Lambton	Derwenthaugh
Moisture	4% max	5.5% max	3.0% max	3.0% max	3.0% max
Ash	9% max	9% max	9% max	9% max	9% max
Volatile matter	1.0% max	0.7% max	0.7% max	1.0% max	1.0% max
Sulphur	0.85% max	0.85% max	1.0% max	1.0% max	1.0% max
2 in shatter index	90 min	90 min	90 min	90 min	90 min
Size					
Mean size	4.2 in min	4.0 in min	4.2 in min	4.2 in min	4.0 in min
Undersize		Not more than 4% less than 2 in.			

All these properties are requirements at the point of dispatch at the coke-oven plant.

NOTE: Some foundries may need to specify a maximum coke size and this can be agreed with National Smokeless Fuels Ltd.

in coking practice. At present the 2 in. shatter index is being used as a measure of the resistance of the coke to degradation during transport.

The industry is now being made aware of how NSF Ltd is complying with these specifications, by regular reports published in the CFA Bulletin. These reports are prepared by BCIRA for CFA from daily figures of the specified coke properties at the coke ovens. BCIRA also maintains quality control graphs so that comment on trends in the properties can be included in the reports.

Cupola Fluxes

The normal purpose of adding a flux with the cupola charge materials is to form a fluid slag, along with impurities charged into the furnace such as sand and rust adhering to the metallic components and lining material removed during melting.

With basic slag operation the flux is added in amounts sufficient to alter the composition of the slag to that required. The material employed is limestone, and good quality material has the following approximate composition:

	%	%
Lime (CaO)	54 (min)	96 (min)
Carbon dioxide (CO_2)	42 (min)	
Silica (SiO_2)	2 (max)	4 (max)
Alumina (Al_2O_3)	1 (max)	
Ferric oxide (Fe_2O_3) and Magnesia (MgO)	1 (max)	

If material having a higher silica content is used, the amount of limestone required has to be increased. With good-quality limestone the amount required in the cupola charge is 25-30 per cent of the weight of the coke charge.

Fluorspar Fluorspar is sometimes used as a means of improving the fluidity of a cupola slag, particularly a basic slag. It serves no purpose other than to improve slag fluidity.

Tapping Methods and Receivers

Continuous tapping, generally with the use of a receiver, has become more widely practised with the increasing trend towards continuous production methods in the foundry.

Very often the criteria for judging the desirability of installing continuous tapping systems and receivers are overlooked, or not considered in every detail. As a result of this, continuous tapping systems are occasionally used in applications to which they are not too well suited, or are not used when they could be – to advantage.

While the decision to use an intermittent or continuous tapping system must ultimately depend on the particular conditions existing at the foundry concerned, the following advantages and disadvantages should always be very closely considered.

Advantages of continuous tapping

(1) The most obvious advantage of continuous tapping is that it eliminates the necessity for frequent tapping and botting. This is of special importance in mechanized foundries which require a constant availability of molten metal for filling ladles, generally of small capacity relative to the cupola output. As the throughput, hours of operation and degree of mechanization diminish, continuous tapping becomes less advantageous until, in the jobbing foundry, there is little application for it.

(2) When a cupola is intermittently tapped the quantity of metal in the well can vary considerably during the melt unless the tapping cycle is carefully controlled. With continuous tapping a constant, though relatively small, quantity of metal is maintained in the well. The wide variation in carbon pick-up sometimes experienced with intermittent tapping, particularly when melting low-carbon irons, may be avoided since the metal has only a brief but uniform period of contact with the coke before reaching the taphole.

(3) When continuous tapping is used in conjunction with a receiver of sufficient holding capacity to meet variable shop demands, melting may be controlled at a steady rate, thereby fulfilling one of the essential conditions of correct cupola operation – to obtain uniformity of metal temperature and composition.

(4) Many of the furnaceman's difficulties are avoided when the cupola is converted to continuous tapping, e.g. it eliminates the occasional failure to bott the taphole in time, the hard taphole, the leaking taphole, slag in the tuyeres and slag in the ladle.

Disadvantages

(1) With continuous tapping, little metal is stored in the cupola well. A receiver is therefore generally required to collect sufficient metal to even out fluctuations in composition. This involves installation, operational and maintenance costs.

(2) Although the metal temperature at the cupola taphole may be somewhat higher, the rate of metal flow down the launder is low compared with intermittent tapping. A substantial loss of temperature occurs as the metal runs down the launder and accumulates in the receiver or ladle. This loss is greater at low melting rates. Thus, for melting rates of less than about 3 tonnes per hour the thin stream of metal tends to suffer an excessive temperature loss.

Continuous tapping methods

The following are the main methods in use for obtaining a constant stream of metal from the cupola without the necessity for closing the taphole after each ladle is filled.

Continuous front-tapping and slagging The most favoured method of obtaining a continuous stream of metal from a cupola utilizes a front-slagging spout. The design of front-slagging spouts may vary to some extent, but the principle of operation is always the same. Fig. 1 shows a simple but effective design similar to that used on the BCIRA experimental cupola.

The taphole is usually sealed with sand before blowing commences (Fig.1a) and is not normally opened until a sufficient volume of metal has collected in the well to flush the spout and run over into the ladle or receiver. As melting proceeds, slag accumulates in the well (Fig.1b) and as it does so the metal-slag interface is depressed until, eventually, slag escapes through the taphole (Fig.1c). Metal and slag then both flow continuously from the furnace at the rate at which they are melted. The slag ascends towards, and overflows, the slag notch.

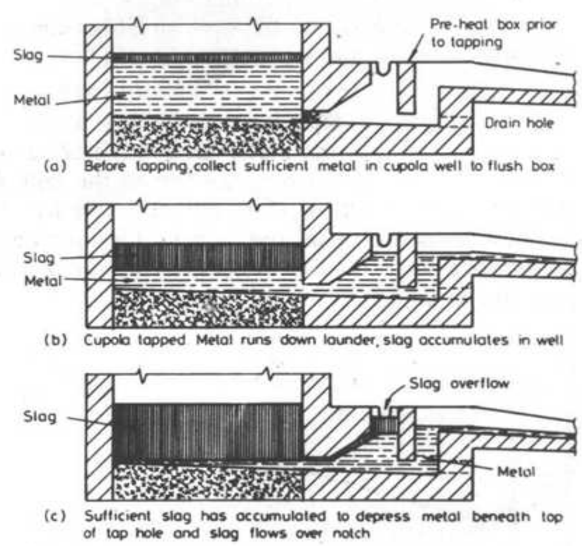


Fig. 1. Continuous front tapping & slagging.

When metal and slag are running freely from the furnace, the ferrostatic pressure of the head of metal X (Fig.2) is equal to the fluid pressure of height of slag S in the well, together with the gas pressure P on the surface of the slag.

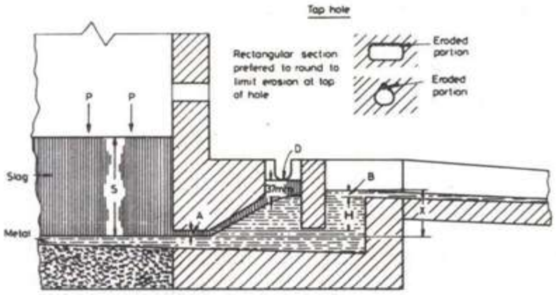


Fig. 2. Dimensions for front-slagging box.

The head of metal X (in) is related to the depth of slag S (in) and the pressure P (in w.g.) as follows:

$$0.25\ X = 0.087\ S + 0.036\ P$$
$$\text{or}\ X = 0.348\ S + 0.144\ P$$

where 0.25, 0.087 and 0.036 are the densities of molten iron, molten slag and water respectively, expressed in lb/in^3.

Since the ratio of densities is unchanged, the same relation between X, S and P holds good in metric units; for heads in cm:

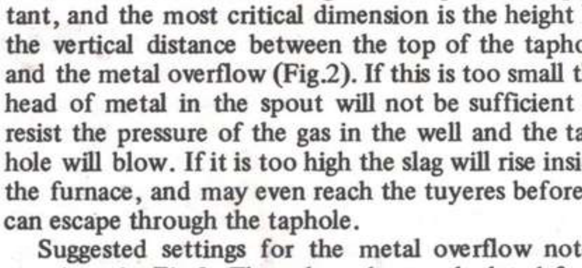
$$X = 0.348\ S + 0.144\ P$$

The gas pressure P in the well is less than the blast pressure in the windbelt, and is usually about 80 per cent of the latter, although it may be considerably lower if high blast velocities (i.e. small tuyere areas) are used.

The correct dimensioning of the spout is important, and the most critical dimension is the height H, the vertical distance between the top of the taphole and the metal overflow (Fig.2). If this is too small the head of metal in the spout will not be sufficient to resist the pressure of the gas in the well and the taphole will blow. If it is too high the slag will rise inside the furnace, and may even reach the tuyeres before it can escape through the taphole.

Suggested settings for the metal overflow notch are given in Fig.3. These have been calculated from the above formula to provide a slag height in the cupola well of approximately 20 cm (8 in). A greater slag height may be maintained by increasing the height of the metal overflow, and vice versa.

In using the formula for calculating the metal overflow height to provide a given slag height, it should be noted that the metal head X (Fig.2) is not the vertical distance between the top of the taphole and the metal overflow, but exceeds this by the thickness B of the metal overflowing the notch and the distance A of the slag-metal interface below the top of the taphole.



Calculated to give slag height of 20cm (8in) from formula: $x = 0{\cdot}348S + 0{\cdot}144P$

$x = H + A + B$

A+B assumed to be 3·8cm (1 in)

P assumed to be 0·8 x windbelt pressure

Fig. 3. Setting of metal overflow notch.

The size of the taphole into the cupola is relatively unimportant, provided that it is large enough to give an unrestricted flow of metal and slag. On the BCIRA cupola the hole is 64mm (2.5 in) deep and 89mm (3.5 in) wide, with the corners radiused. As shown in Fig.2, this is preferred to a round taphole since it reduces the vertical erosion at the top of the taphole and so preserves a more constant dimension throughout the melt.

The box is made up by ramming refractory material round a suitable former. For acid cupolas this may be a high-fusion-point ganister, but when high tonnages are melted or the metal tapping temperature is high the use of graphite or plumbago-bearing ganister is to be preferred. Several proprietary materials of this type are now available. Very good results have also been obtained with the use of a proprietary high-alumina ramming material.

Continuous front-tapping and rear – or side-slagging

In this system, which is illustrated in Fig.4, the metal and slag are withdrawn through separate siphons. Its use is generally restricted to larger cupolas, and in certain respects it has advantages compared with the system of continuous front-tapping and slagging. With the latter system, only a very small reservoir of metal is held in the cupola well, and this gives rise to variations in the composition of the metal at the cupola spout. Such variations are generally serious when melting charges consist of materials of dissimilar chemical and physical characteristics. It is usual, however, to provide a receiver to even out the variation in composition of the metal leaving the furnace. In addition, the metal has only a brief period of contact with the coke in the well, and this tends to restrict the pick-up of carbon. With the system of continuous front-tapping with separate rear or side slagging, the amount of metal held in the cupola well can be fixed by appropriate design of the metal and slag siphons. However, unless very deep siphons are provided, the quantity of metal stored in the well is still limited; consequently, the use of this method to fill casting-ladles of relatively small capacity compared with the charge weight, without the use of a receiver, should be confined to such applications where the metal charges consist of a mixture of materials which do not vary markedly in composition, e.g. pig iron and cast iron scrap. Another advantage of this sytem is that it permits a better adjustment of refractory materials to the conditions of their service. For example, the slag hole and siphon box may be constructed with carbon ramming paste, and the metal hole and box with graphitized ganister or plastic high-alumina ramming material. This is a special point in its favour when operating with basic slags.

The slag-hole is not opened until about ½ to 1 hour after melting has started, to ensure that a sufficient quantity of slag will have accumulated in the well to flush and heat up the system and avoid freezing-up. Careful observation should be made during this period to ensure that the slag level does not reach the tuyeres. Provision must also be made for draining the well at the end of the shift.

The method of calculating the dimensions of the metal and slag siphons to provide desired metal and slag depths is given in Fig.4. A more detailed consideration of the design of siphons for continuous tapping systems is given by Pelczarski,[1] who provides a nomogram for continuous front-tapping and rear-slagging systems, from which the essential dimensions

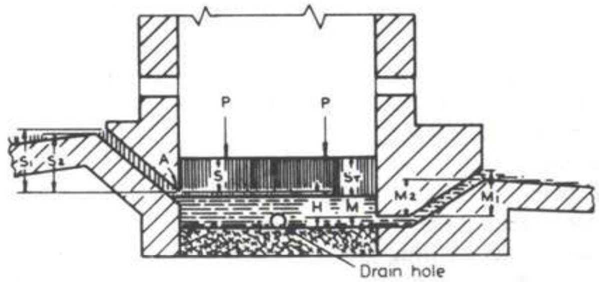


Fig. 4. Continuous front tapping and rear slagging.

Slag Siphon

for hydraulic balance: $0.087 S_1 = 0.087 S + 0.036 P$

or $S_1 = S + 0.414P$ .. (1)

Metal Siphon

for hydraulic balance:

$0.25 M_1 = 0.25 M + 0.087 S_T + 0.036P$

or $M_1 = M + 0.348 S_T + 0.144P$ (2)

Example: Required: Total slag height S_T = 9in.
Metal height M = 6in.

Cupola windbelt pressure = 30in.H_2O

Assume pressure in well = 0.8 x windbelt pressure = 24in.H_2O

Dia. of metal outlet = 3in.

Dia. of slag outlet = 3in.

Metal slag interface in well to be 1in. below edge A

Calculation for slag siphon: (formula (1))

$S_1 = S + 0.414P$

$S = S_T - 1 = 9 - 1 = 8$

$\therefore S_1 = 8 + 0.414 \times 24 = 8 + 9.9 = 17.9$in.

Calculation of metal siphon: (formula (2))

$M_1 = M + 0.348 S_T + 0.144P$

$= 6 + 0.348 \times 9 + 0.144 \times 24$

$= 6 + 3.13 + 3.46 = 12.6$in.

Make actual slag and metal overflow heights, S_2 and M_2 a little less than calculated values to allow for thickness of slag and metal streams at overflow.

Distance H between top edges of slag and metal outlets
= 6in + 1in = 7in.

Note: All these calculations can be performed in metric units so long as all the pressure heads are expressed in the same unit throughout; e.g. centimetres, when pressure P has to be cmH_2O (and 1in.H_2O, sometimes termed 1in.w.g. = 2.54 cmH_2O).

may be read to suit varying requirements.

The siphon brick This system of tapping is illustrated in Fig.5. At the back of the siphon brick there is an opening 'A' into the cupola and this allows metal to enter channel 'B' which passes up through the centre of the brick. On the front of the brick are three tapholes 'C', 'D' and 'E', each connected to the central channel. Sometimes only two holes are provided. At the beginning of the melt all three tapholes are left open, and when metal starts to melt it is allowed to run through the bottom hole 'C' until it is thoroughly heated. This hole is then botted up and the metal is allowed to rise up the central channel and flow out through the second hole 'D'. Before the metal runs through this hole the tapping spout should be made up with sand to the higher level. When the second hole is sufficiently hot it should be botted up and the same procedure adopted for the third hole, the spout being made up to give an easy flow of metal from the top hole while the metal is rising.

All that is required to stop the flow from the taphole is to reduce or shut off the blast. This reduces the pressure P in the cupola well, causing the metal level in the well to rise and that in the siphon brick to fall. The height H should be at least 1cm every 686 Pa (1 in for every 7 in water gauge) blast pressure, to ensure that sufficient metal will always be retained in the well to seal the opening into the siphon brick.

The slag may be removed either intermittently or continuously from a slag-hole located at a convenient distance beneath the tuyeres. If the slag is allowed to collect, the amount of metal held in the well will

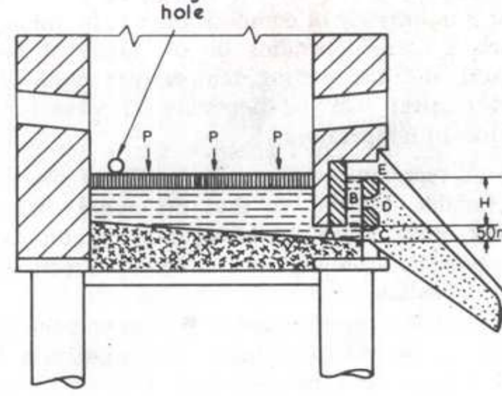


Fig. 5. Siphon-brick method of cupola tapping.

be correspondingly less and there is some justification for keeping the slag-hole open continuously to obtain the best mixing of the charge components.

This tapping system is very useful where small quantities of metal, for example in hand shanks, are required at regular intervals. Its use for this purpose is generally confined to foundries producing castings in phosphoric iron. Metal is tapped by opening the blast valve and partially or completely closing it again after each ladle is filled. When operated in this way with intermittent blowing, the cupola will require more coke than in normal operation to maintain the metal temperature. Furthermore, as melting proceeds only very slowly when the blast is off, this type of operation will not give the output of metal obtainable with continuous blast on a furnace of the same internal diameter. When a cupola is to be operated with this system, therefore, its rated output should be considerably in excess of that actually required.

In some cases the siphon brick is employed as a means of obtaining an uninterrupted supply of metal from the cupola, the blast being maintained and metal flowing from the taphole at the rate at which it is melted. When used in this way either a tilting spout or receiver is required. On very long shutdowns it may be necessary to drain off the slag and then drain the well by opening the bottom hole of the siphon brick. The well must be drained in this way at the end of the melt before the bottom is dropped.

Receivers

Any vessel interposed between the cupola and the carrying-ladles may be considered to be a receiver. In deciding if a receiver will be of benefit, a careful study should be made of the advantages and disadvantages to be obtained.

Advantages

(1) If the receiver is of sufficient capacity it serves to average out the variations in metal composition at the cupola spout. To fulfil this function alone, the larger the capacity the better, but for a given throughput the loss of metal temperature increases with the size of the receiver. A compromise must therefore be made. When melting a mixture of pig iron and cast iron scrap the capacity of the receiver should be the equivalent of at least two metal charges.

If the proportion of steel scrap in the charges is high the receiver should hold the equivalent of at least three to four metal charges. If the charges consist predominantly of steel scrap with a high proportion of ferro-alloys the receiver may hold up to 1 hour's output if uniformity of composition is to be obtained. In such a case, depending on the grade of metal produced and the casting temperature required, a heated receiver may be necessary in view of the likely loss of temperature.

(2) A receiver provides a reservoir of metal to meet variable demand, so that the cupola may be operated under more consistent blowing conditions than is possible when the metal is tapped directly into the casting ladles.

It should be appreciated that if a certain volume of metal is required for uniformity of composition, this volume cannot simultaneously be available to meet variations in demand. In practice it is usual to compromise, and maintain a receiver about two-thirds full to allow for fluctuations in demand or breakdown in supply.

(3) A receiver lends itself to continuous tapping. Continuous tapping without a receiver has only a very limited application, but is sometimes used in conjunction with a tilting spout from the ends of which a series of ladles may be filled alternately. On the other hand, when a receiver is used, continuous tapping is usual but not essential.

(4) Should the metal require any treatment, for example desulphurization, this can be carried out more easily in a receiver, as the necessity for removing the slag from each ladle is avoided.

Disadvantages

(1) Additional equipment and maintenance costs.

(2) Metal tapped from a normal receiver is always lower in temperature than when tapped direct from the cupola. The temperature loss when metal passes through a receiver is seldom less than 40 °C and can be considerably greater, particularly in the case of relatively small cupolas or when the receiver is oversized in relation to the cupola melting capacity. Unless special precautions are taken to preheat the receiver before metal is tapped into it, the temperature loss will be greatly exaggerated at the beginning of the melt.

Receiver design There is a wide variety of receiver designs, but they can be very broadly classified into two categories – fixed and tilting. Very few fixed receivers are now in use. Receivers of the tilting type are usually barrel ladles of circular or U-section (Fig. 6). They may be static or mobile. Receivers should be covered, leaving the metal inlet opening as small as is consistent with safety. They should be equipped with teapot spouts, whereby the metal is drawn from the bottom, ensuring a supply of clean metal.

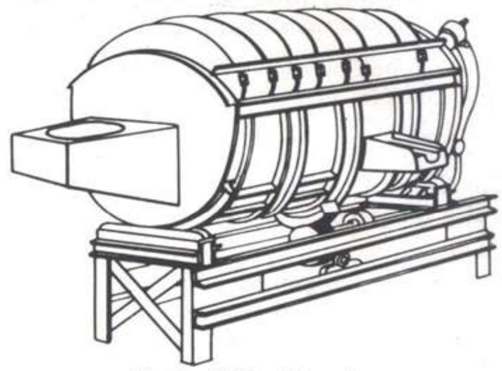


Fig. 6. Tilting U-receiver.

Heated receivers Heated receivers are usually U-shaped in section and provided with permanent oil or gas burners. They have the obvious advantage over the unheated types that the metal temperature loss is smaller. However, it should be clearly recognized that these receivers are not intended to raise the temperature of the metal they contain. They are much more expensive to install, operate and maintain, but where high, steady metal temperature, uniformity of composition, and a large reservoir of metal to maintain continuity of metal supply are of cardinal importance, the expense may be justified.

Electrically heated receivers Electric furnaces have found increasing application as holding units to receive metal from a cupola furnace. In this application the electric furnace provides buffer storage of iron, so that the cupola can be operated continuously, independent of variations in metal demand from the foundry. Following any breakdown in the foundry, metal can be supplied immediately on restart, at the correct temperature for pouring, and the expensive pigging of metal can be avoided. The large batch capacity of the electric furnace also assists in homogenizing the composition of the cupola-melted iron. Compared with fuel-fired heated receivers, electrically heated receivers have the facility to maintain or increase metal temperature.

Reference

1. Pelczarski (S.M.):
British Foundryman, 1962, v. 55, September, pp. 375–81.

Clean-air Requirements

Legislation

Interest in clean air is nationally very high, with the result that we have had for many years a continually changing situation as new Acts and Regulations have been passed. The scene is likely to be as changeable in the future as it has been in the past. The position presented is that obtaining in late 1977. Costs change even more rapidly than legislation, and for this reason are deliberately ignored except in relative terms.

In the UK, emissions from cold-blast cupolas and from hot-blast cupolas are supervised by different bodies and subject to different Acts and requirements. So far as cold-blast cupolas are concerned supervision is by Local Authority officers under the Clean Air Acts of 1956 and 1968. Neither Act is specific, and each gives only general guidance on the measures to be taken to reduce emission. More detailed guidance, which still leaves a number of factors unresolved, is given in recommendations issued in 1968 by the (then) Ministry of Housing and Local Government. The following is taken from the publication[1] containing these recommendations.

'Summary of Recommendations

(i) Smokeless methods should be used for lighting-up cupolas.

(ii) Combustion should be maintained in the cupola shaft. This can be achieved, in the majority of cases, either by suitable operation of the cupola or by means of an independent flame.

(iii) All cupolas must be fitted with means for minimizing grit and dust emission. Dry arresters should normally be permitted for use on cupolas melting up to 3 tons per hour and operating up to 250 hours per annum, and wet arresters should be required on cupolas melting above 5 tons per hour and operating 500 hours per annum or more. Between these two limits the type of arrester required should be decided by discussion.

(iv) New cupolas, and existing cupolas where justifiable complaint is made regarding smoke, gases or smells, should be fitted with chimneys as prescribed by 'Chimney Heights'[2] except that the height should never be less than 65 feet. In the case of existing or new cupolas where there is or is likely to be a metallurgical fume problem, fume arrestment should first be considered. If arrestment is shown to be impracticable, then the fume should be dispersed from a chimney of a height normally not less than 120 feet. In new cupolas where it is not certain whether a metallurgical fume problem will arise, foundations and structures should be strong enough to enable the chimney height to be increased to 120 feet or more.

Determination of chimney height

In following 'Chimney Heights'[2] the hourly emission rate of suplhur dioxide has to be estimated. The maximum rate of fuel consumption can usually be estimated.

A typical figure for the sulphur content of foundry cupola coke is 0.7 per cent. It can be assumed that 50 per cent of the sulphur is absorbed by the molten metal and a further 30-50 per cent of the remaining sulphur will be absorbed by the water spray where a wet arrester is fitted, i.e.

Assuming sulphur content of coke is 0.7 per cent

Equivalent sulphur content after allowing for sulphur removed by the metal is 0.35 per cent. (This figure should be taken when using 'Chimney Heights' to calculate the height of cupolas having dry arresters.)

Equivalent sulphur after allowing for sulphur removed in the metal and wet arresters is 0.20 per cent. (This figure should be used when calculating the discharge height for cupolas fitted with wet arresters.)'

While the above outlines the 1977 position, the likely shape of future requirements for cold-blast cupolas is already known.

In 1972 the Ministry of Housing & Local Government convened the Second Grit & Dust Working Party to recommend emission limits for cold-blast cupolas and other furnaces. The Council of Ironfoundry Associations represented the industry on the Working Party, whose report (Feb.74) was issued by the Department of the Environment in July 1974.[3] The report forms the basis for Regulations that are likely to be issued in 1980; the Secretary of State in issuing them is not obliged to adopt the recommendations of the Working Party, but earlier experience suggests that such recommendations will be generally accepted.

Allowable emission limits will be set for the first time, by a Regulation made under the Clean Air Act, 1968, for the amount of solid matter that may be discharged from cold-blast cupolas of various sizes. The Working Party's recommendations have taken account of limits for other classes of furnace and those in use in other countries, as well as a knowledge of the design and cost of equipment required to meet those limits. Until now, only the type of collector has been recommended.

Measurement of emission

The limits set, for cupolas up to 10 t/h capacity, are likely to apply exclusively to grit and dust, and since cupolas also emit metallurgical fume a technique for measuring these to the exclusion of fume is necessary. This may be done by using a size selecting sampler (the BCIRA cyclone probe)[4] which removes dust and grit from the sampled air stream. Particles smaller than dust (defined by the Working Party as particles of less than 1 μm) pass through the cyclone but may be retained on an optional backing filter. The weight of emission caught in the cyclone is the amount to be used in estimating the emission rate.

For larger cupolas (over 10 t/h capacity) the Working Party recommended that the weight of metallurgical fume caught in the backing filter should be added to that caught by the cyclone in estimating the emission rate, to take account of the possible metallurgical-fume nuisance at this capacity.

Control of emission

For small cupolas (under 3-4* t/h) the suggested limits were set so as to permit the use of simple natural-draught wet arresters; for even the smallest cupolas, wet arresters will now be necessary. (Tables 1 & 2).

Tables of recommended allowable emission limits.

Table 1 Existing cupolas

Ton per hour capacity	Allowable emission of grit & dust, lb/h	Allowable emission of grit & dust and fume	Possible collector required to meet limit
1	6.6	–	Simple wet arrester
2	13.2		
3	19.8		
4	26.4		
4 + step-change	17.1	–	Multicyclones or Medium-intensity scrubbers
5	18.0		
6	18.9		
7	19.6		
8	20.3		
9	20.9		
10	21.5		
11	–	22.1	High-intensity scrubber or fabric filter or electro-static precipitator
12		22.6	
13		23.1	
14		23.6	
Over 14 ton/h add 0.5 lb/h per additional ton/hour.			

Table 2 New cupolas

Ton per hour capacity	Allowable emission of grit & dust, lb/h	Allowable emission of grit & dust and fume	Possible collector required to meet limit
3	19.8	–	Simple wet arrester
3 + step-change	15.7		
4	17.1	–	Multicyclones or Medium-intensity scrubbers
5	18.0		
6	18.9		
7	19.6		
8	20.3		
9	20.9		
10	21.5		
11	–	22.1	High-intensity scrubber or fabric filter or electro-static precipitator
12		22.6	
13		23.1	
14		23.6	
Over 14 ton/h add 0.5 lb/h per additional ton/hour.			

Unlike cold-blast cupolas, which come under the jurisdiction of Local Authority, hot-blast cupola melting is a scheduled process under the Alkali, etc., Works Act 1906, and, as such, is supervised by the Alkali Inspectorate. The stringent requirements to which hot-blast cupolas are subject are given in the Chief Inspector's Directive 10/73.[5]

For the medium-range cupolas (from 3-4 t/h* to 10 t/h) some form of fan-powered collector system will be needed. This can be of medium efficiency, since it is not required to collect visible metallurgical fume; multicyclones or medium-intensity scrubbers will be suitable. A considerable increase in collection efficiency becomes possible when a fan is installed, and this is reflected in a step-change in the allowable emission rate; (as a concession to industry it was suggested that the step-change for allowable emission rate should come at 4 t/h for existing cupolas, and 3 t/h only for new cupolas).

For cupolas over 10 t/h, the emission should be at – or near – invisible rate, and limits were suggested accordingly. Only collectors of the highest efficiency, such as high-intensity scrubbers, fabric filters or electrostatic precipitators, can meet these limits.

There is likely to be some exemption for medium and large cupolas used for only limited periods, but this is still a matter for discussion.

Application of limits to new & existing cupolas

As soon as the Regulations take effect, it is likely that all cupolas ordered will have to comply with them. The Working Party recommended that for existing cupolas there should be an eight-year period of grace before medium - and high-efficiency collection systems have to be fitted; they should, in the meantime, at least be fitted with simple natural-draught wet arresters within, say, three years.

Definition of melting rate

For the purpose of the Regulations, melting rate will probably be measured in terms of the internal diameter at tuyere level. While this does not give an exact melting rate, it does give a figure that is easily measured by those in charge of administering the forthcoming Regulations. The Working Party recommend that the following formula should be used:

$$\textbf{Melting rate (t/h)} = \textbf{0.6} \times \textbf{cross-sectional area at tuyeres } (\textbf{ft}^2).$$

Discharge Height

The Working Party was not called upon to recommend stack height, but suggested that the minimum stack height remain at 65 ft, as recommended in Cold-blast Cupolas.[1]

Smoke emission

Smoke was outside the scope of the Working Party, and its report therefore makes no mention of the smoke generated by dirty scrap. Cold-blast Cupolas[1], however, recommends that wherever possible smoke should be burnt before emission, if necessary with the help of afterburners.

Compliance with requirements

(a) Cold-blast cupolas – smoke

Cupola stack gases may burn spontaneously when mixed with air entering the charge-hole. If they do not they can often be made to burn by the use of an afterburner. In general, the higher the proportion of coke charged and the shorter the distance between tuyeres and sill level, the greater is the likelihood of being able to obtain stack-gas combustion.

An afterburner is simply a suitably positioned gas or oil burner, as shown in Fig.1. Minimum fuel consumption for small and medium-sized cupolas is likely to be 18-32 l/h (4-7 gal/h) of oil, or its gas equivalent.

* Expected definition for small cupolas: new cupolas up to 3 t/h; existing cupolas up to 4 t/h.

Low coke charges (under 11 per cent) produce a weak gas at, or below, the limits of combustion. In this rather unusual case, combustion of the stack gases is at least difficult and may be impossible in a conventional cupola. Where the coke charge lies between 11 and 14 per cent, spontaneous combustion at the charge hole may occur but, if not, can usually be initiated with an afterburner. With coke charges above 14 per cent spontaneous ignition is usual, but where it does not occur an afterburner will maintain combustion with ease. (See Table 3)

Table 3

	Coke charge % (metal: coke ratio)		
	Below 11 (9:1)	11-14 (9:1-7:1)	Above 14 (7:1)
Spontaneous combustion above charge hole	no	sometimes	usually
Ability of afterburners to maintain combustion	difficult	usually	always

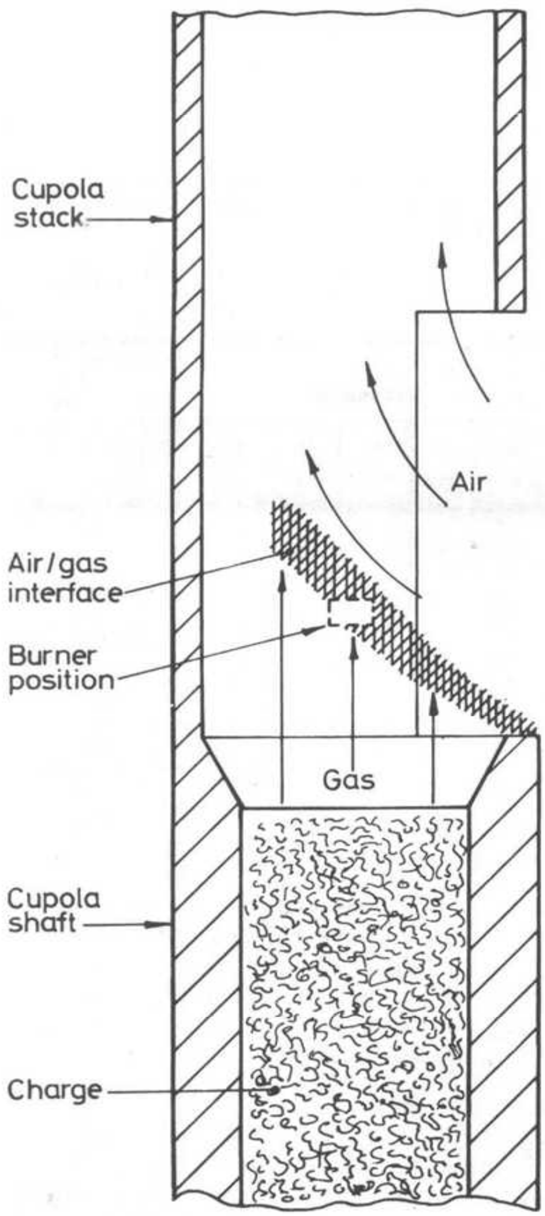


Fig. 1. Position of the after-burner at the charge-hole.

The easiest and cheapest way of getting rid of cupola smoke while lighting up is by using a gas or oil burner to ignite the coke. It is helpful to make a tunnel in the bed at the fettling door by packing the coke round a 15cm (6 in) diameter pipe which is subsequently removed. The gas or oil flame is then directed into the tunnel. Lighting-up by a gas or oil burner is quicker and often cheaper than using wood or waste materials.

(b) Cold-blast cupolas – grit, dust and fume

Present recommendations call for simple dry or wet arresters for the collection of grit and dust from most cupolas. The design of arrester is similar whether dry or wet, though if run dry, a refractory lining is required. Detailed sizes are given in Figs. 2 & 3 and Table 4; but as a general statement, the arrester should be large (to limit resistance to the passage of gas), thick (to give a reasonable life) and fitted some distance above the charge-hole (to provide sufficient stack draught).

Wet arresters will collect half or more of the solids in the cupola gases and 1/3 to 1/2 of the sulphur dioxide. Dry arresters are applicably less efficient, and cannot remove sulphur dioxide.

Exceptionally, more efficient collection may be asked for and this will invariably require a fan-powered collector. Medium-efficiency collectors, e.g. cyclones, can remove the bulk of the solids but do not reduce the opacity of the gases. To make the gases nearly invisible, only a collector capable of cleaning the gases down to 115 mg/m^3 (0.05 grains/ft^3) is of any use. Few such collectors have yet been used on cold-blast cupolas in the UK.

Before cupola gases can be cleaned in anything other than simple wet or dry collectors, they must first be collected into a duct. If we exclude the closed-top cupola (e.g. MBC design) there is a choice between taking the gases from above the charge-hole or withdrawing them through off-takes below stock top level.

Above-charge-hole offtake:

Simple – allows combustion in the cupola stack,

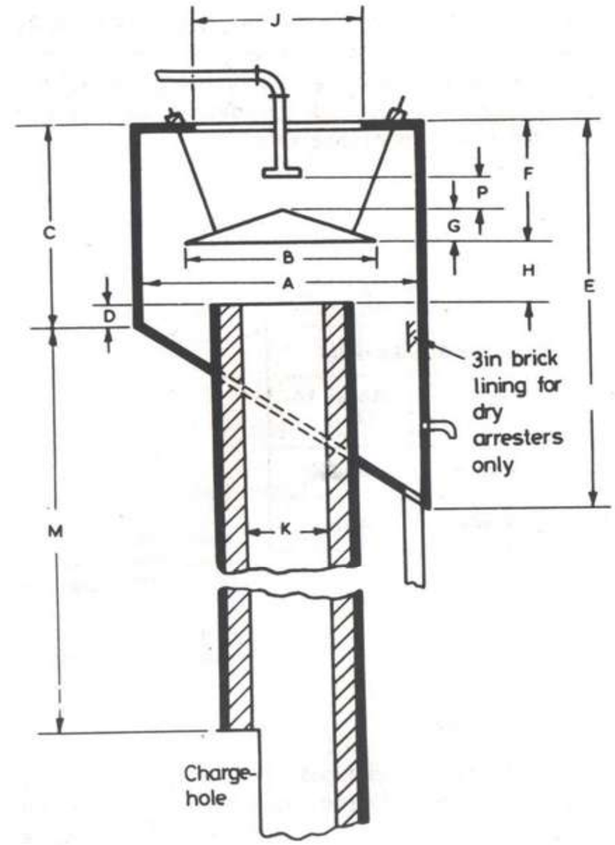


Fig. 2. Arrester dimensions for use with non-tapered cupola shells.

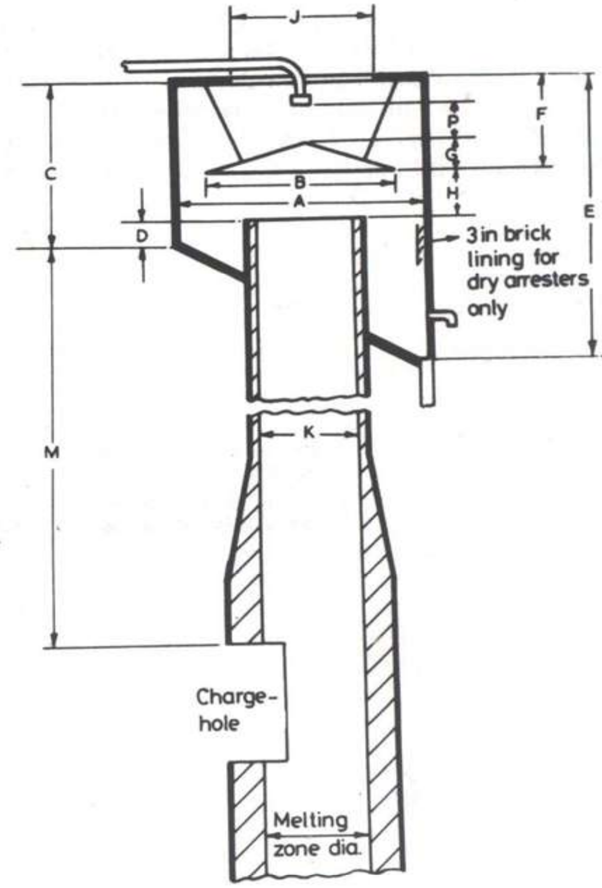


Fig. 3. Reduced shell diameter for cupolas having small charge-holes : diameter K must not be less than the melting-zone diameter.

Table 4
Basic design dimensions for arresters

Melting zone dia., cm	Lined diameter above charging door (K) cm	Arrester shell dia. (A), cm	Cone dia., (B) cm	Dimensions J cm	H cm	G cm	P cm	M cm	D cm	F cm	C cm	E cm	Water flow, l/min
53	69	160	107	122	38	18	18	610	23	53	114	206	136
61	76	175	114	130	46	20	20	610	23	61	130	229	182
69	84	198	130	145	46	23	23	610	23	69	137	251	227
76	91	213	137	160	53	23	23	610	23	69	145	267	273
84	99	229	145	168	61	25	25	610	23	76	160	290	341
91	107	244	152	183	61	25	25	610	30	76	168	305	409
99	114	259	168	198	69	28	28	610	30	84	183	335	500
107	122	274	175	213	76	30	30	610	30	91	198	358	591
107	130	290	183	229	76	30	30	640	30	91	198	366	636
114	137	305	191	244	84	33	33	700	30	99	213	389	727
122	145	320	198	251	84	33	33	730	30	99	213	396	818
130	152	335	206	268	91	36	36	760	30	107	229	419	955
137	160	351	213	274	91	36	36	790	30	107	229	434	1000
145	168	366	221	290	99	38	38	850	30	114	244	457	1091
152	183	396	236	320	107	38	38	915	30	114	251	480	1182

but increases the amount of gas to be cleaned. (Fig.4)

Below-charge offtake:
Imposes a control problem but reduces collector size, and may restrict choice of cleaner. (Fig.5)

High efficiency collectors include fabric filters, electrostatic precipitators and high-energy scrubbers.

Fabric filters

Fabric filters have been used extensively for cupolas in parts of the USA. The gases are cooled, usually by controlled evaporation of water to 250 °C and then blown into a glass-fibre fabric filter. Fig.6 shows a typical layout. Gases must be free from smoke and oily vapours, otherwise 'blinding' of the filter material occurs. The gases must therefore be burnt before cleaning. Newer designs utilizing heat exchangers for cooling, and where the collector is under negative pressure, are also available.

Electrostatic precipitators

Electrostatic precipitators collect particles by electrostatic attraction. One system is illustrated in Fig.7. They are more expensive than other equivalent methods of gas cleaning, but their power requirements are lower. Smoke and oily vapours should first be removed by combustion. Precipitators have found

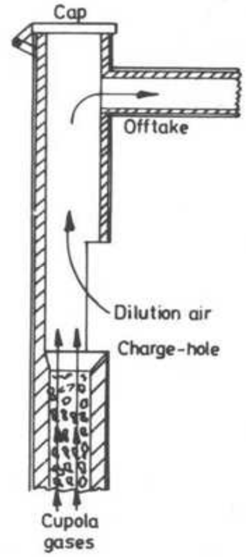


Fig. 4. Offtake above the charge door.

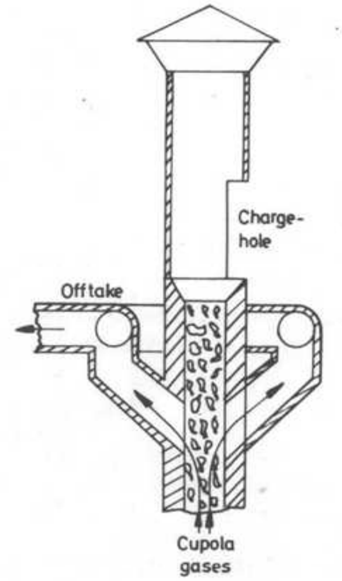


Fig. 5. Offtake below the charge level.

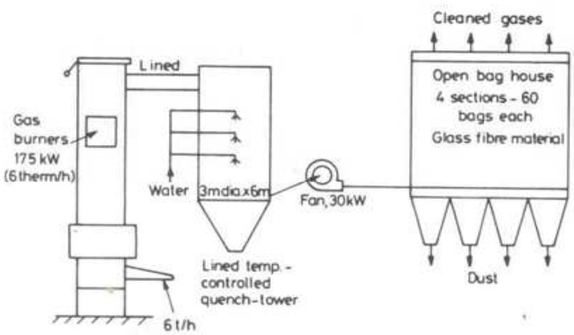


Fig. 6. Typical bag-filter cupola gas-cleaning installation.

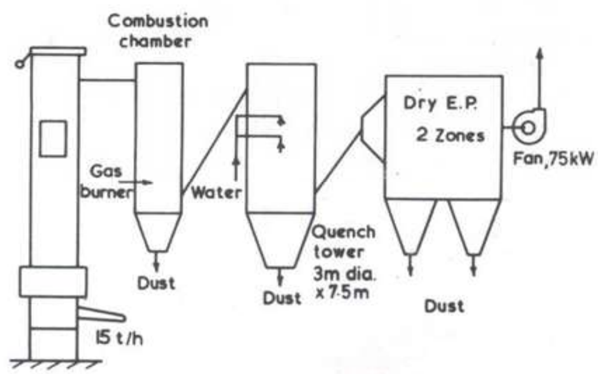


Fig. 7. Typical dry electrostatic-precipitation cupola gas-cleaning installation.

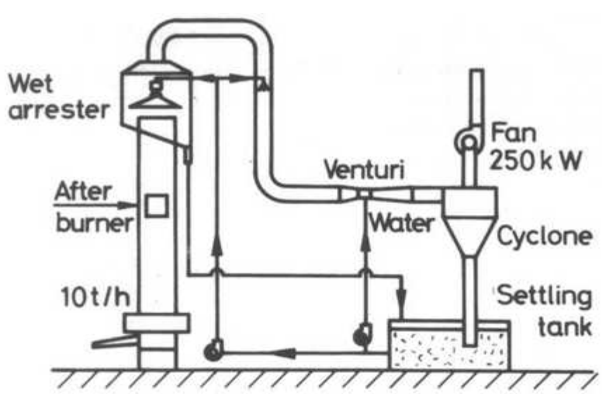


Fig. 8. Venturi scrubber for cold-blast cupola.

relatively little favour for cupolas, possibly because of the specialized maintenance required and the fact that they are rather sensitive to gas temperature and humidity.

High-energy scrubbers

The most popular form of metallurgical fume collector for cupolas uses some design of high-energy scrubber. Of these the venturi scrubber (Fig.8) has been used most widely, but the disintegrator has advantages in some applications. Gases are pre-cooled with excess water and drawn through a venturi tube into which more water is injected. The wetted dust is removed in a cyclone or other simple collector. While such plants have the merit of simplicity, they require the highest power input, and produce a water-pollution problem which may be expensive to solve.

Hot-blast cupolas

All new hot-blast cupolas must be fitted with 'full-fume arrestment', i.e. gas-cleaning down to a dust content not exceeding 115mg/m^3 (0.05 grains/ft^3) at s.t.p.* Existing cupolas have to comply with this standard by the end of 1978. Such cleaning requires fabric filters, electrostatic precipitators or high-efficiency washers. If a separate blast heater is used, the layout of the cleaning plant will be the same as for cold-blast cupolas. In the case of recuperative plant, all the gases will be withdrawn below charge level and, if the gases are to be cleaned before the recuperator, the choice of cleaner is effectively limited to some form of wet collector. Disintegrators are becoming increasingly popular for the purpose because their efficiency is independent of any variations in gas flow caused by alterations in operating conditions. Additionally, they are simple, robust, and require little maintenance. A typical layout is shown in Fig. 9.

* Defined by Alkali Inspectorate as 60 ^{o}F and 30 inches of mercury (15 ^{o}C and 1 bar)

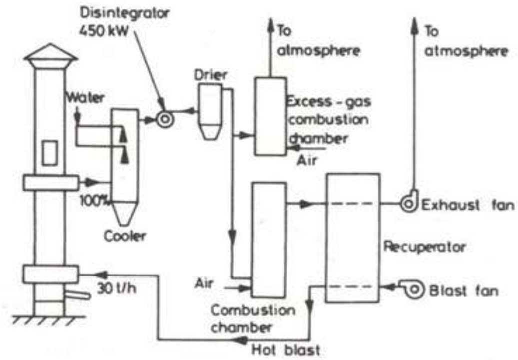


Fig. 9. Disintegrator gas-cleaning for 30t/h hot-blast cupola.

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Specification of Cupola Plant

The economics of melting in ironfoundries is influenced by a considerable number of complicated factors. It is difficult, therefore, to present a general case for the choice of any particular type of melting-plant; this has to be made from careful technical and economic appraisal of the conditions and requirements relating to an individual foundry. A summary of the main factors to be considered when appraising melting schemes is given below:

Basic requirements

Metal grade.
Minimum and maximum metal demand.
Type of demand – intermittent or continuous.
Ladle sizes and usage.
Tapping and pouring temperatures.
Period metal required – single shift or double shift.

Other considerations

Availability and cost of materials.
Metal treatments.
Quality considerations.
Space availability.
Fuel availability and cost
– coke
– electricity supply and tariff
– oil
– gas
Clean-air requirements.
Maintenance requirements.

Cupola design

If, after consideration of these factors, it is decided that the most suitable plant is a cold-blast cupola melting-plant, it is possible to specify the cupola or to assess a manufacturer's specification in the following manner using the recommended design data given in Table 1.

1. Establishing melting rate The cupola should be capable of melting faster than the average hourly requirements of the foundry for molten metal, to allow for off-blast periods, de-slagging and variability in demand. The melting rate should be specified to be at least 10 per cent higher than the average hourly production rate.

2. Decide on metal:coke ratio. In general, the amount of coke increases with the tapping temperature, the amount of steel scrap in the charge, and the need for carbon pick-up. If in doubt, a high coke charge should be assumed for the purpose of specification. Locate the melting-rate in Table 1, Column 1, for the metal: coke ratio selected.

3. Clean air – control of smoke, fume and grit emissions.
With knowledge of the melting rate required and the coke charge to be used, a decision can be made about the equipment necessary to deal with Clean Air requirements and consultation with the Local Authority can be carried out.

4. Blowing rate. The blowing rate required for the melting-rate and metal:coke ratio is obtained from Column 2.

5. Blowing equipment. The recommended blower capacity to meet the blowing rate requirement is given in Column 5. The volume is 20 per cent higher than that for the recommended specific blast rate of 114 m^3/m^2 min (375 ft^3/ft^2 min) and the discharge pressure is approximately 50 per cent higher than the anticipated windbelt pressure when operating at this specific blast rate.

6. Blast control equipment. Consideration should be given to blast-control equipment required.

7. Melting-zone area. With knowledge of the blowing rate, the melting zone area (Column 3) can be calculated, based on a specific blast rate of 114 m^3/m^2 min (375 ft^3/ft^2 min) of melting-zone area. From this the internal diameter of the furnace can be calculated (Column 4).

8. External shell diameter. Having established the internal diameter, the external shell diameter can be derived from the lining thickness considered necessary. Generally a thickness of 22.5cm (9 in) is satisfactory for a melting period up to four hours, and for a longer period – up to about eight hours – the lining thicknesses should be at least 30 cm (12 in). If a melting period in excess of eight to ten hours is involved, provision should be made to water-cool the shell at the melting zone to reduce consumption of refractory.

9. Well. The design of the well depends on the tapping system selected. If the furnace is tapped intermittently the well capacity should be equal to the weight of two to four metal charges, depending on the charge make-up. For example, with charges containing high percentages of steel scrap the well capacity should be equal to four charges. With continuous tapping, where metal mixing takes place in a receiver, the depth of the well can be less than with intermittent tapping, and will depend on the design of the tapping system.

It is important to realize that well depth has a marked effect on the temperature of metal melted in a continuously tapped cupola, and each centimetre (inch) reduction in depth will increase the metal tapping temperature by 1.5 °C (4 °C). This applies also to intermittently tapped cupolas. It is important, therefore, that the well depth i.e. the distance from the sand bottom to the bottom row of tuyeres, should be no greater than necessary, and it certainly should not exceed 1m (3 ft).

Other points to observe when designing the well are to make sure that the tap-hole and slag-hole are accessible. Do not, for example, have a tap-hole accessible only to a left-handed cupola operator, and engage a right-hand operator. It is also important not to forget the fettling-door, and for this to be of adequate size and accessibility.

Approximate well capacity is given in Column 6. When calculating well depth, the height from the tap-hole to the slag hole should be used.

10. Height of base plate above ground level. This depends on factors such as ladle size, whether a receiver is used, and the height of ladles suspended from a monorail. In general, a height less than 1m makes it difficult to remove the cupola drop, particularly if this is collected in a skip.

11. Drop-bottom doors. Drop-bottom doors should be of rigid design, and ribbed to avoid distortion. When designing the cupola it should be ensured that, when dropped, the doors will not impede removal of the drop. With large cupolas, and consequently heavy doors, or with cupolas where the base plate is greater than 1-1½m (3 to 4 ft) above ground level, thought should be given to mechanical operation of the doors with a simple winch arrangement or an air cylinder.

12. Windbelt. The purpose of the windbelt, or windbelts if divided-blast is being used, is to provide a chamber from which the air can be uniformly distributed to each tuyere. Normally, a windbelt having an external diameter 60-100cm (2 to 3 ft) greater than the shell diameter and a depth of 75-120cm (2 ft 6 in to 4 ft) is adequate.

When the cupola is being designed, thought should be given to possible future operation using water-cooling, which requires the windbelt to be detached from the cupola shell.

13. Blast main. The blast main should be large enough to conduct the air from the blower to the windbelt with a minimum of pressure loss. Blast-main diameters vary from 30cm (12 in) with small cupolas up to 60cm (24 in) with larger cupolas. The blower should be located so that the blast main contains the minimum number of bends.

To regulate air delivery, a valve should be positioned in the blast main near to the windbelt. This should be easily accessible to the cupola attendant.

14. Tuyeres. In most cupolas the tuyeres are positioned below the windbelt, with air supplied from the windbelt through an elbow connection. This arrangement enables simple valves or shutters to be fitted to individual tuyeres to cut off the air supply to a tuyere. To avoid tuyeres filling with slag or metal, one tuyere should be slightly lower than the others and have a fusible plug at the bottom of the elbow. With intermittently tapped cupolas the tuyeres are normally positioned 15cm (6 in) above the slag hole.

The total area of the tuyeres is normally 1/4 to 1/7 of the melting-zone area. Tuyere shape, whether round or rectangular, is determined by personal preference of the foundry.

The tuyeres should be readily accessible through the cover plates, and a platform should be installed to permit this – rather than asking the cupola attendant to stand on a box, or to clean tuyeres using a bar held above his head.

15. Height to charging-door sill. The position of the tuyeres having been established, the height from the tuyeres to the charging-door sill can be established. This should be related to the melting rate of the cupola, and for a rate up to 5 t/h should be 5m (16 ft), for 5 to 8 t/h, 5.8m (19 ft), and for over 8 t/h, 6.7m (22 ft).

16. Stack height. With knowledge of the heights of the cupola below charge door level, the charge hold, and the emission discharge to meet statutory requirements or fume cleaning equipment, the height of the stack above the charging door can be derived.

Other design features

Charging equipment

(a) The charge-bucket dimensions should be sufficient to accommodate the proposed charge materials, including coke if charged with the metal, and while this can be calculated it is better to determine the capacity of the bucket using the actual charge. Many foundries have forgotten to check the size of the bucket in the quotation for new plant. The charge weight is normally 10 per cent of the melting rate, but may be smaller when using high steel-content charges with limited well capacity.

(b) Do not underdesign the charging equipment by economizing on the thickness of steel used.

(c) Include safety devices such as limit switches and slack-wire control, and guard the pit which houses the charging bucket at ground level. With charging machines having a breeches chute, make sure the cupola not being used cannot be charged.

(d) Ensure, in the case of cupolas using skip-hoist chargers, that the pit cannot be filled with slag or the cupola drop.

(e) With mechanically charged cupolas always provide an auxiliary door at or a little below charging-door sill level, and an adequate platform, so that the height of the cupola bed can be measured.

Material thickness

When assessing manufacturer's quotations, careful comparison should be made of metal thicknesses employed. For guidance, typical metal thicknesses found suitable are:

Cupola shell	**10mm (3/8 in) mild steel**
Windbelt	**6mm (¼ in) mild steel**
Blast main	**14 gauge mild steel**
Base frame	**20x10cm (8 x 4in) RSJ**
Base plate	**40mm (1½ in) mild steel**
Bottom doors	**20mm (¾ in) mild steel with ribbed reinforcement**
Columns	**20x10cm (8 x 4in) RSJ with gaiter**
Tuyeres and elbows	**Cast iron**

Working conditions

Many well-designed cupola plants are operated with difficulty, and dangerously, because insufficient thought was given at the design stage to ways in which the working conditions of the cupola operators could have been aided. Some features frequently forgotten are:

Carbon monoxide

There have been serious accidents, some fatal, due to exposure of personnel to carbon monoxide arising during cupola operation. A cupola shell is not gas-tight and it contains gases, rich in CO under pressure. CO leaks from the bottom doors, breast-plate, tapping box, riveted joints, etc., and is detectable around most cupolas. The gases at the charge-hole contain 10-20 per cent CO, and any leakage of cupola gases from the charge-hole is potentially dangerous.

Simple wet arresters wash the gases and the water returns to a settling-tank at ground level. The water will drag down some CO-laden gases released in the settling-tank or slag-granulation sluice.

Precautions which should be taken to reduce the risk of CO leakage include:

(a) Design arresters correctly.

(b) Burn stack gases wherever possible to convert most of the CO present to less harmful CO_2.

(c) Use mechanical charging to remove operators from the cupola charging platform.

(d) Provide good ventilation round the cupola, especially at charging level.

(e) Use above-roof-level open troughs for wet arrester water return, to allow gases trapped in the water to be dispersed harmlessly outside the foundry.

Slag removal

Collect slag in a properly designed ladle or skip; do not allow it to accumulate on the floor. It is difficult to justify slag granulation for other than a small number of large-output cupolas.

Platforms

Provide properly constructed platforms at the base of the cupola at charging-door level and at the arrester. Do not expect the cupola attendant to tap the cupola or to clean tuyeres while standing on a box, or to measure the coke bed standing on a ladder. Make sure that, in an emergency, the cupola attendant can leave a platform at the base of the cupola without difficulty. Provide simple access to settling-tanks of wet-arrester equipment.

Air-control valves

Position air-control valves so that in an emergency the cupola attendant can shut off the air supply to the cupola without delay.

Cupola drop

The importance of adequate height under the cupola to ensure easy removal of the drop has been mentioned already. Consideration should also be given to collection of the drop in a properly designed skip, which should include drain holes. Give thought to the removal of the cupola drop from the cupola area to ensure that there are no restrictions to this. Consider also the provision of protection around the base of the cupola to avoid the drop flying around the foundry.

Bottom doors

Consider the method to be used to close the bottom doors and also how a prop holding the doors is to be pulled away without creating a hazard for the operators.

Table 1a. Summarized cupola data (SI units)

1			2	3	4	5		6	7	8	9
Melting rate at various metal:coke ratios under steady operating conditions t/h			Recommended blowing rate*, m^3/min	Cross-sectional area of melting zone, m^2	Diameter of melting zone cm	Recommended blower capacity		Approximate well capacity kg/cm height	Total tuyere area, cm^2	Number of tuyeres	Approximate bed weight, kg/cm height
Metal:coke ratio						Volume, m^3/min	Discharge pressure kPa				
10:1	8:1	6:1									
1.6	1.3	1.1	18.8	0.164	46	22.7	10.0	6.0	225–420	4	0.7
2.2	1.9	1.5	25.5	0.223	53	30.6	10.2	8.0	325–550	4	1.0
2.9	2.5	1.9	33.4	0.292	61	40.2	10.5	10.5	420–740	4	1.3
3.7	3.1	2.5	42.3	0.370	69	51.0	10.7	13.3	515–935	4	1.7
4.5	3.9	3.1	52.1	0.456	76	62.3	11.0	16.3	645–1130	4	2.0
5.5	4.7	3.7	63.2	0.552	84	75.9	11.2	19.8	775–1300	6	2.5
6.6	5.6	4.4	75.0	0.657	91	90.0	11.5	23.3	935–1645	6	2.9
7.7	6.5	5.2	88.1	0.771	99	106	11.7	27.5	1100–1935	6	3.5
9.0	7.6	6.1	102.5	0.896	107	123	12.0	31.7	1290–2225	6	4.0
10.3	8.7	6.9	117.2	1.026	114	141	12.2	36.7	1450–2500	8	4.6
11.7	10.0	7.9	133.7	1.168	122	160	12.7	41.7	1680–2900	8	5.2
13.2	11.2	8.9	150.0	1.318	130	180	13.0	46.7	1870–3290	8	5.9
14.8	12.6	10.0	168.5	1.478	137	202	13.2	53.3	2100–3680	8	6.6
16.5	14.0	11.2	188.3	1.646	145	126	13.7	58.3	2350–4100	8	7.4
18.3	15.6	12.4	208.4	1.824	152	251	14.2	65.0	2610–4550	8	8.2
22.1	18.8	14.9	292.0	2.206	168	303	14.9	78.3	2870–5030	10	10.0
26.5	22.5	17.9	301.6	2.626	183	360	15.7	93.3	3740–6580	10	11.8
31.1	26.4	21.0	354.0	3.083	198	425	17.2	110	4390–7740	10	13.5
36.1	30.6	24.3	410.6	3.574	213	493	18.7	128	4510–9030	10	16.0

* Actual blowing rate specified at $115 m^3/m^2 min$ to take into account satisfactory cupola operation at values within $^+$ 20 per cent of this figure.

† Recommended blower volume capacity based on recommended blowing rate in Column 2 plus 20 per cent; discharge pressure approximately 50 per cent higher than anticipated windbelt pressure.

Table 1b. Summarized cupola data (Imperial units)

1			2	3	4	5		6		7	8	9	
Melting rate at various metal:coke ratios under steady operating conditions, t/h			Recommended blowing rate*, ft^3/min	Cross-sectional area of melting zone, ft^2	Diameter of melting zone, in	Recommended blower capacity†		Approximate well capacity, cwt		Total tuyere area, in^2	Number of tuyeres	Approximate bed weight, cwt	
Metal:coke ratio						Volume, ft^3/min	Discharge pressure, in w.g.	Per ft height	Per 3 in height			Per 5 ft height	Per ft height
10:1	8:1	6:1											
1.6	1.3	1.1	665	1.77	18	800	40	3.6	0.9	35–65	4	2.2	0.4
2.2	1.9	1.5	900	2.40	21	1 080	41	4.8	1.2	50–85	4	3.0	0.6
2.9	2.5	1.9	1 180	3.14	24	1 420	42	6.3	1.6	65–115	4	3.9	0.8
3.7	3.1	2.5	1 495	3.98	27	1 800	43	8.0	2.0	80–145	4	5.0	1.0
4.5	3.9	3.1	1 840	4.91	30	2 200	44	9.8	2.4	100–175	4	6.1	1.2
5.5	4.7	3.7	2 230	5.94	33	2 680	45	11.9	3.0	120–215	6	7.4	1.5
6.6	5.6	4.4	2 650	7.07	36	3 180	46	14.0	3.5	145–255	6	8.8	1.8
7.7	6.5	5.2	3 110	8.30	39	3 740	47	16.5	4.1	170–300	6	10.4	2.1
9.0	7.6	6.1	3 620	9.64	42	4 350	48	19.0	4.8	200–345	6	12.0	2.4
10.3	8.7	6.9	4 140	11.04	45	4 970	49	22.0	5.5	225–400	8	13.8	2.8
11.7	10.0	7.9	4 720	12.57	48	5 660	51	25.0	6.3	260–450	8	15.7	3.1
13.2	11.2	8.9	5 300	14.19	51	6 360	52	28.0	7.1	290–510	8	17.7	3.5
14.8	12.6	10.0	5 950	15.90	54	7 140	53	32.0	7.9	325–570	8	19.9	4.0
16.5	14.0	11.2	6 650	17.72	57	7 980	55	35.0	8.8	365–635	8	22.1	4.4
18.3	15.6	12.4	7 360	19.63	60	8 850	57	39.0	9.8	405–705	8	24.5	4.9
22.1	18.8	14.9	8 900	23.75	66	10 700	60	47.0	11.7	445–780	10	30.0	6.0
26.5	22.5	17.9	10 650	28.27	72	12 700	63	56.0	14.0	580–1020	10	35.5	7.1
31.1	26.4	21.0	12 500	33.18	78	15 000	69	66.0	16.5	680–1200	10	40.5	8.3
36.1	30.6	24.3	14 500	38.47	84	17 400	75	77.0	19.2	700–1400	10	48.0	9.6

* Actual blowing rate specified at 375 ft^3/ft^2 min to take into account satisfactory cupola operation at values within $^+$ 20 per cent of this figure.

† Recommended blower volume capacity based on recommended blowing rate in Column 2 plus 20 per cent; discharge pressure approximately 50 per cent higher than anticipated windbelt pressure.