Chapter 10

Electric Furnace Steelmaking

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Over the past 20 years the use of the electric arc furnace (EAF) for the production of steel has grown considerably. There have been many reasons for this but primarily they all relate back to product cost and advances in technology. The capital cost per ton of annual installed capacity generally runs in the range of \$140–200/ton for an EAF based operation. For a similar blast furnace–BOF based operation the cost is approximately \$1000 per annual ton of installed capacity. As a result EAF based operations have gradually moved into production areas that were traditionally made through the integrated route. The first of these areas was long products—reinforcing bar and merchant bar. This was followed by advances into heavy structural and plate products and most recently into the flat products area with the advancement of thin slab casting. At the current time, approximately 40% of the steel in North America is made via the EAF route. As the EAF producers attempt to further displace the integrated mills, several issues come into play such as residual levels in the steel (essentially elements contained in the steel that are not removed during melting or refining) and dissolved gases in the steel (nitrogen, hydrogen, oxygen).

Both of these have a great effect on the quality of the steel and must be controlled carefully if EAF steelmakers are to successfully enter into the production of higher quality steels.

There have been many advances in EAF technology that have allowed the EAF to compete more successfully with the integrated mills. Most of these have dealt with increases in productivity leading to lower cost steel production. These are described in the detailed process sections.

10.1 Furnace Design

The design of electric arc furnaces has changed considerably in the past decade. Emphasis has been placed on making furnaces larger, increasing power input rates to the furnace and increasing the speed of furnace movements in order to minimize power-off time in furnace operations.

10.1.1 EAF Mechanical Design

Many of the advances made in EAF productivity over the last 20 years are related to increased electrical power input and alternative forms of energy input (oxygen lancing, oxy-fuel burners) into the furnace. These increased energy input rates have only been made possible through improvements in the mechanical design of the EAF. In addition, improvements to components which allow for faster furnace movement have reduced the amount of time which the furnace stands idle. Thus the objective has been to maximize the furnace power-on time, resulting in maximum productivity.

10.1.1.1 Furnace Structural Support

In the 1960s and 1970s it was common to install electric arc furnaces at grade level. These furnaces would have pits dug out at the front for tapping and at the back for pouring slag off into slag pots. This configuration lead to many interferences and delays and is no longer recommended for large scale commercial operations. Modern furnace shops usually employ a mezzanine furnace installation. Thus the furnace sits on an upper level above the shop floor. The furnace is supported on a platform which can take on several different configurations. In the half platform configuration, the electrode column support and roof lifting gantry is hinged to the tiltable platform during operation and tapping. When charging the furnace, the complete assembly is lifted and swiveled. This design allows for the shortest electrode arm configuration. In the full platform design, the electrode column support and roof lifting assembly is completely supported on the platform. These configurations are shown in Fig. 10.1.

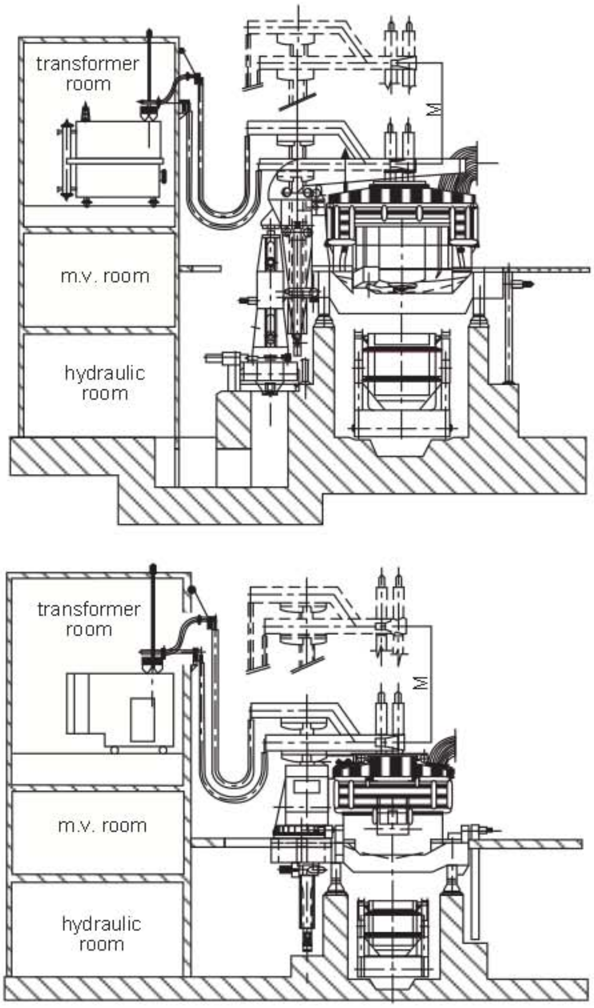


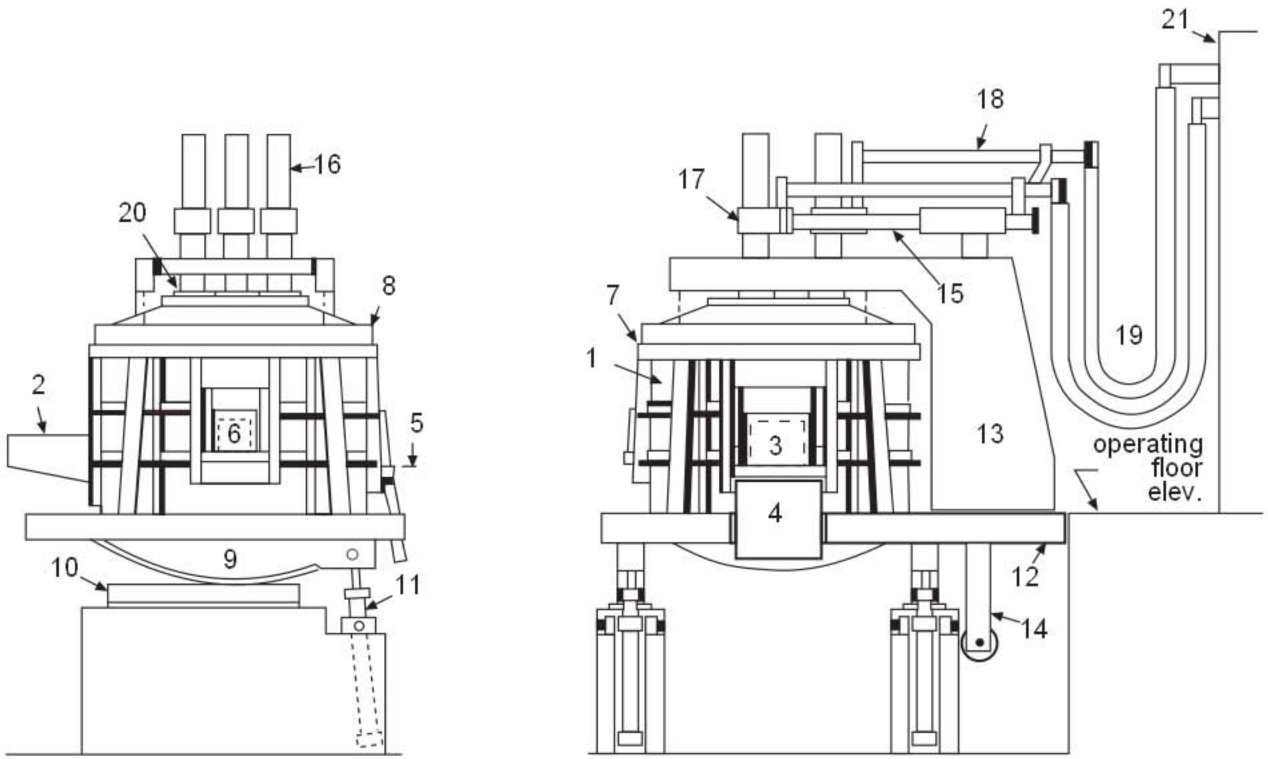
Fig. 10.1 Furnace platform configurations. *(Courtesy of Danieli.)*

10.1.1.2 General Furnace Features

The EAF is composed of several components as shown in Fig. 10.2. These components fall into the functional groups of furnace structures for containment of the scrap and molten steel, components which allow for movement of the furnace and its main structural pieces, components that support supply of electrical power to the EAF, and auxiliary process equipment which may reside on the furnace or around its periphery.

The EAF is cylindrical in shape. The furnace bottom consists of a spherically shaped bottom dish. The shell sitting on top of this is cylindrical and the furnace roof is a flattened sphere. Most modern furnaces are of the split shell variety. This means that the upper portion of the furnace shell can be quickly decoupled and removed from the bottom. This greatly minimizes down time due to changeout of the top shell. Once the top shell is removed, the furnace bottom can also be changed out fairly quickly. Some shops now follow a practice where the shell is changed out on a regular basis every few weeks during an eight hour downshift.

The furnace sidewall above the slag line usually consists of water-cooled panels. These panels are hung on a water-cooled cage which supports them. The furnace roof also consists of water-cooled panels. The center section of the roof which surrounds the electrode ports is called the roof delta and is a cast section of refractory which may be water cooled. The furnace bottom consists of a steel shell with several layers of refractory. This is discussed in more detail in Section 10.1.2. Fig. 10.3 shows a modern EBT EAF.



side door elevation

rear door elevation

Fig. 10.2 EAF general features. *(Courtesy of Center for Materials Production.)*

1. shell	8. roof ring	15. electrode mast arm
2. pouring spout	9. rocker	16. electrode
3. rear door	10. rocker rail	17. electrode holder
4. slag apron	11. tilt cylinder	18. bus tube
5. sill line	12. main (tilting) platform	19. secondary power cables
6. side door	13. roof removal jib structure	20. electrode gland
7. bezel ring	14. electrode mast stem	21. electrical equipment vault

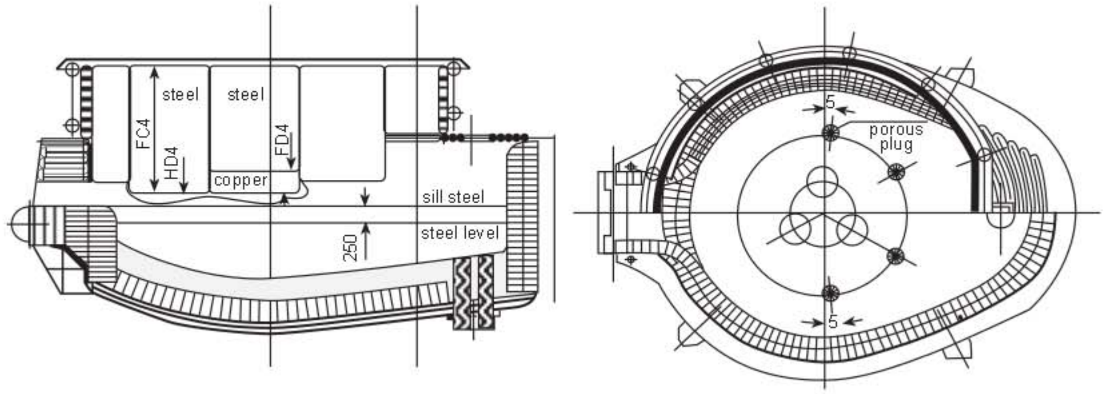


Fig. 10.3 Plan and section views of a modern EBT EAF. *(Courtesy of Danieli.)*

10.1.1.3 Water-Cooled Side Panels

One of the most important innovations in EAF design was water cooling, Fig. 10.4. Although this was used to a limited extent in older furnace designs for cooling of the roof ring and door jambs, modern EAFs are largely made up of water-cooled panels which are supported on a water-cooled cage. This allows for individual replacement of panels with a minimum of downtime. By water cooling the cage structure, it can be ensured that thermal expansion of the cage does not occur. Thus warping of the cage due to thermal stresses is avoided as are the resulting large gaps between the panels. Water-cooled panels allow very large heat inputs to the furnace without damaging the furnace structure. In the older EAF designs, these high power input rates would have resulted in increased refractory erosion rates and damage to the furnace shell.

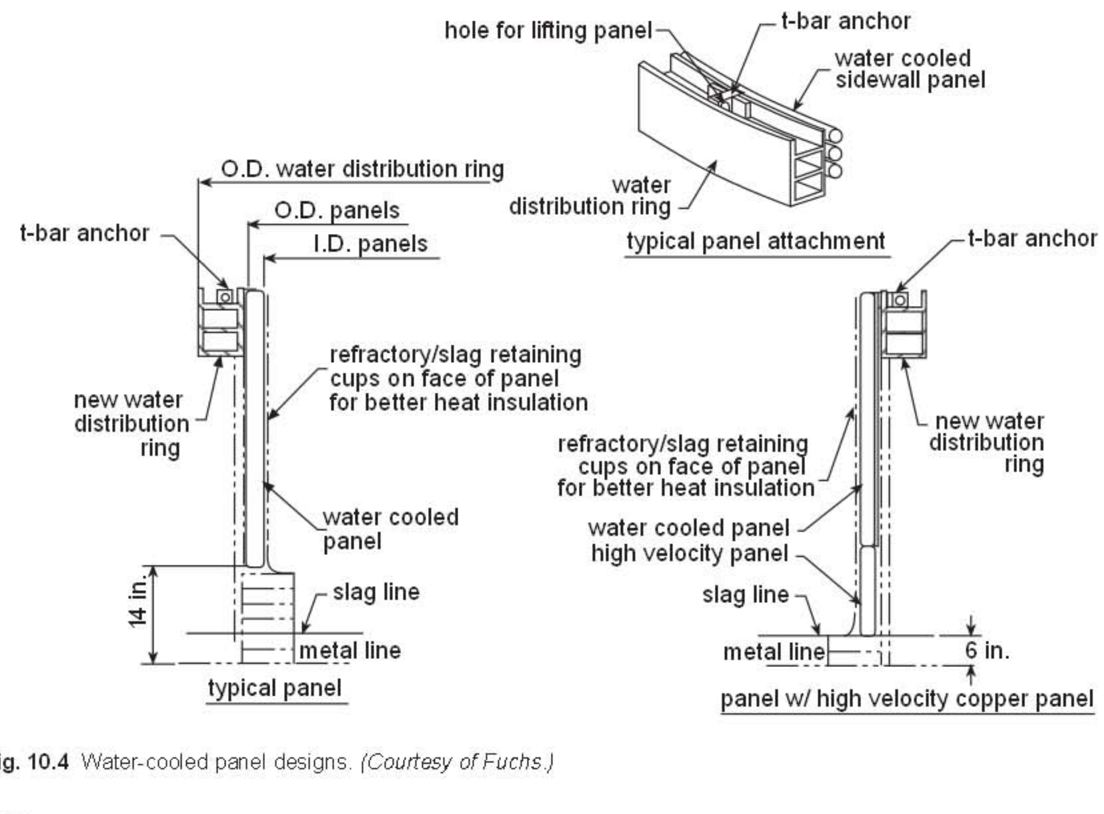


Fig. 10.4 Water-cooled panel designs. *(Courtesy of Fuchs.)*

Parameters which have a strong influence on panel life include water quantity and quality, water flow rate and velocity, inlet water pressure and pressure drop across the panel, pipe/panel construction material, and pipe diameter.

Water-cooled panels must withstand both high thermal loads as well as high mechanical loads. The greatest mechanical load occurs during furnace charging. Scrap can strike the panels causing denting of pipes or even splitting and rupture. Thus the pipe selection must allow for a wall thickness which can withstand these forces. At the same time, a minimum wall thickness is desired in order to maximize heat transfer to the cooling water. This tradeoff must be evaluated in order to arrive at an optimum pipe thickness. Generally, the minimum pipe thickness used is 8 mm. Maximum pipe thickness will depend on the thermal load to be removed and the amount of thermal cycling. In practical application the wall thickness is usually 8–10 mm.

In most applications, boiler tube grade A steel is used for water-cooled panels. These steel grades are generally reasonably priced, are easy to work with and provide a suitable thermal conductivity for heat transfer (approximately 50 $Wm^{-1}K^{-1}$) up to thermal loads of 7 million kJ $m^{-2}hr^{-1}$ (10,000 BTU $ft^{-2}min^{-1}$). However, this material is also susceptible to fatigue caused by thermal cycling.

In areas which are exposed to extremely high heat loads, copper panels might be used. Copper has a thermal conductivity (383 $Wm^{-1}K^{-1}$) approximately seven times that of boiler tube enabling it to handle very high thermal loads up to 21 million kJ $m^{-2}hr^{-1}$ (approximately 31,000 BTU $ft^{-2}min^{-1}$). Copper panels will result in much greater heat transfer to the cooling water and thus to optimize energy consumption in the EAF, copper panels should only be used in areas where excessive heats loads are encountered (e.g. close to the bath level, oxy-fuel burner ports etc.). This will of course be dependent on the amount of slag buildup over these panels. Generally, slag buildup will be thicker on copper panels due to the greater heat transfer capacity. Alternatively, the distance between the start of water-cooled panels and the sill level can be increased if steel panels are to be used in the lower portion of the water-cooled shell.

For most boiler tube grade water-cooled panels, a pipe diameter of 70–90 mm is usually selected. For copper panels, the choice of tube diameter will usually be a cost based decision but frequently falls into the same range. In some high temperature zones (e.g. lower slag zone), a smaller diameter copper pipe is used with higher water velocity to prevent steam formation.

Cooling panel life is primarily dependent on the amount of thermal cycling that the panel is exposed to. During flat bath conditions, the panels can be exposed to very high radiant heat fluxes. Following charging, the panels are in contact with cold scrap. Thus the amount of thermal cycling can be considerable. The optimum solution to this problem involves providing maximum cooling while minimizing the heat flux directly to the panel. In practice it has been found that the best method for achieving these two goals is to promote the buildup of a slag coating on the panel surface exposed to the interior of the furnace. With a thermal conductivity of 0.12–0.13 $Wm^{-1}K^{-1}$, slag is an excellent insulator. Cups or bolts are welded to the surface of the panels in order to promote adhesion of the slag to the panels.

It is important that the temperature difference across the pipe wall in the panels does not get too high. If a large temperature differential occurs, mechanical stresses (both due to tension and compression) will build up in the pipe wall. Most critically, it is important that the yield stress of the pipe material not be exceeded. If the yield stress is exceeded, the pipe will deform and will not return to its original shape when it cools. Repeated cycling of this type will lead to transverse cracks on the pipe surface and failure of the panel.

Panel layout is related to several factors including thermal load, desired temperature rise for the cooling water, desired pressure drop across the panel, and desired pipe diameter.

Generally, designers aim for a water velocity which will result in turbulent flow within the pipe. Thus the minimum required water velocity will vary depending on the inside diameter of the pipe which is used. A minimum flow velocity of approximately 1.2 m s^{-1} is quoted by several vendors.[1,2,3] The maximum velocity will depend on the desired pressure drop but will usually be less than 3 m s^{-1}. The range quoted by several vendors is 1.2–2.5 m s^{-1}. This will help to minimize the possibility of steam formation in the pipe. Some systems are designed so that a water velocity of at least 2.5 m s^{-1} is provided to ensure that the steam bubbles are flushed from the panel. Failure to remove smaller steam bubbles can result in a larger bubble forming. Steam generation could lead to greatly reduced heat transfer (by a factor of ten) and as a result possible panel overheating and failure in the area where the steam is trapped. In extremely high temperature regions such as the lower slag line, water velocities in excess of 5 m s^{-1} are used to ensure that steam bubbles do not accumulate in the panel.

A vertical configuration is preferred in most modern designs because it minimizes scrap holdup and also can help to minimize pressure drop across the panel. Generally water cooling systems on EAFs are designed for an average water temperature rise (i.e. outlet − inlet) of 8–17°C with a maximum temperature rise of 28°C. Most furnace control systems have interlocks which will interrupt power to the furnace if the cooling water temperature rise exceeds the maximum limit.

In high powered furnaces Fuchs Systems Inc. (FSI) recommends that a water flow rate of 150 l min^{-1} m^{-2} (3.65 gpm ft^{-2}) for a side wall panel or 170 l $min^{-1}m^{-2}$ (4.14 gpm ft^{-2}) for a roof panel should be available. For DC furnaces an additional 10 l $min^{-1}m^{-2}$ should be made available.

Pressure drop across the panel is a parameter frequently ignored by most furnace operators. In order to get uniform flow to each panel, it is imperative that the pressure drop across panels be as even as possible. FSI recommends that the water pressure exiting the panels be at least 20 psi. Frequently, panels of different design are mixed within the same supply system. This occurs because replacement panels are frequently sourced from local fabricators as opposed to the original furnace vendor. Slight changes in materials, dimensions and configuration can lead to big changes in the panel pressure drop. Mismatched panels within the same supply circuit will lead to some panels receiving insufficient flow and ultimately panel failure.

Most water-cooled panels are designed to be airtight. However, replacement panels are sometimes made without complete welds between pipes. This may be a cost saving when considering the fabrication of the panel but will lead to increased operating costs and lost production. The EAF is operated under negative pressure and as a result, air will be pulled into the furnace through any openings. Thus openings should be minimized, especially those in and around the water-cooled panels.

10.1.1.4 Spray-Cooled Equipment

Spray cooling has also shown itself to be a viable means for cooling EAF equipment. Spray-cooled components are typically used for roof and sidewall cooling. The main components of a spray-cooled system are the inner shell, spray nozzles and water supply system. Water is sprayed against the hot surface (inner shell) of the cooling element. The outer shell acts purely for containment of the water and steam. The inner shell is fabricated from rolled plate. A uniform spray pattern is essential for proper heat removal from the hot face. Maintenance of a slag coating on the exposed side of the hot face is also integral to regulating heat removal and protection of the cooling panel. In general, at least half an inch of slag coating should be maintained.

The water supply system for spray-cooled panels is simpler than that used for conventional water-cooled panels because the operation is carried out at atmospheric pressure. Strainers are provided to ensure that suspended solid material is not carried through the circuit where nozzle clogging could become a concern.

10.1.1.5 Furnace Bottom

The furnace bottom consists of a spherical plate section. This section is refractory lined and this lining will generally consist of a safety lining with a rammed working lining on top. The plate thickness is usually between 25 and 50 mm thick depending on the furnace diameter. Refractory lining thickness is generally a function of operating practices and change out schedule. For AC fur-

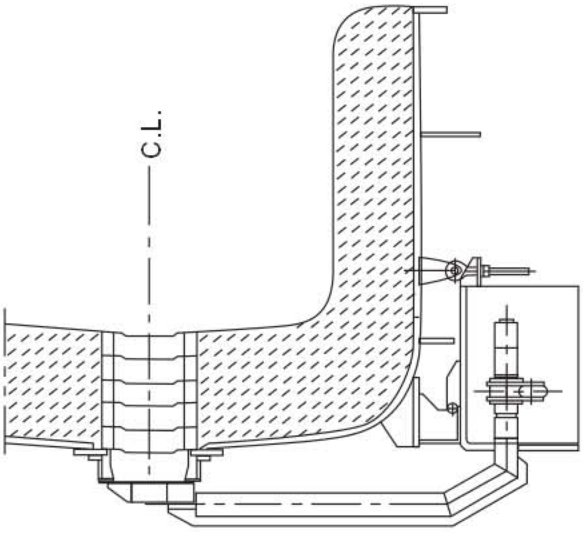


Fig. 10.5 Eccentric bottom tapping configuration. *(Courtesy of Danieli.)*

nace operations, the typical refractory thickness may range between 500 and 700 mm. DC furnaces typically have thicker working linings and the resulting refractory thickness can range from 750–950 mm. Furnace refractories are described in more detail in Section 10.1.2.

In some cases gas stirring elements are installed in the bottom of the furnace. If this is the case, special pocket blocks will be installed during installation of the brick safety lining. Alternatively, stirring elements are lowered into place and refractory is rammed around them.

The furnace bottom section will also contain the tapping mechanism. Most modern furnaces employ bottom tapping. The reasons for this are several, including slag free tapping, decreased steel temperature losses during tapping, hot heel practices, and mechanical considerations.

Bottom tapping requires that the ladle be moved into position under the furnace on a ladle car or using a ladle turret arm. This is beneficial as it frees up the crane for other duties.

For bottom tapping furnaces, the taphole is located in the upper section of the oval shell as shown in Fig. 10.5. The taphole is filled with sand following tapping, prior to tilting the furnace upright for charging. The sand is held in place with a slide gate or a flapper type valve which retracts in the horizontal plane.

Many furnaces with tapping spouts are in operation in North America and in the case of pit furnace operations, offer the only option. If a submerged siphon spout is used, slag free tapping can be achieved though maintenance can be an issue. A leveled spout allows for tapping of steel and slag together and is common in the stainless steel industry. Some furnace spouts have now been retrofitted with slide gates on the spout so that quick shutoff of the tap stream can be achieved. This also allows for slag free tapping. Fig. 10.6 shows several different spout configurations.

Typically, during furnace operation, the bath depth will not exceed 1 m in depth. Bath depth will be a function of furnace diameter and typically ranges from 0.7–1.0 m. During bottom tapping, it is important to maintain a minimum height of steel above the taphole to prevent vortexing which will cause slag entrainment. The typical recommendation is that the height of liquid metal be 2.5 times the taphole diameter.

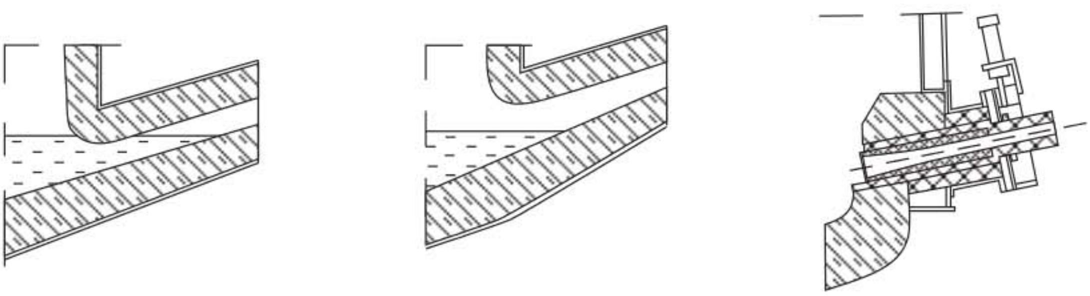


Fig. 10.6 Tapping spout configurations. *(Courtesy of Danieli.)*

10.1.1.6 Furnace Movements

In the course of electric arc furnace operations it is necessary for several of the furnace components to move. Typical requirements for movement include roof raise/rotation to allow for scrap charging, electrode raise/lower and swing to allow for scrap charging, electrode raise/lower for arc regulation, furnace tilt forwards for tapping, slag door up/down for deslagging operations, furnace tilt backwards for slag removal, and electrode clamp/unclamp to adjust the working length of the electrode.

Most furnace movements are made using a central hydraulic system which is usually sized for the maximum flow requirement of the individual task requiring the greatest flow rate of hydraulic fluid. Thus the size of the system can be kept to a minimum while still meeting the requirements for furnace operations.

10.1.1.7 Furnace Tilting

The EAF is tilted for both tapping and slag removal. In the case of furnace tapping, the maximum forward tilting angle will be dependent on the type of furnace bottom. For conventional spout tapping, it may be necessary to tilt to an angle of 45° to fully tap the furnace. For bottom tapping furnaces the maximum tilt angle is usually 15–20°. An important requirement of slag free tapping is that the furnace can be tilted back quickly as soon as slag begins to carry over into the ladle. The typical maximum forward tilting speed is 1° per second. Most modern furnaces have a tilt back speed of 3–4° per second. Any attempt to tilt back faster than this will place undue stress on the furnace structure. Many furnace designs now arrange the center of the cradle radius so that the furnace is balanced slightly back tilted towards the deslagging side. Many furnaces are also designed so that if hydraulic power is lost, the furnace center of gravity will cause the furnace to tilt back up into its resting position.

The furnace movement can be controlled by using either one or two double acting cylinders. The furnace bottom sits on a cradle arm which has a curved segment with geared teeth. This segment sits on a rail which also has geared teeth. As the tilt cylinder is extended, the furnace rocks forward to tap the furnace.

For removing slag from the furnace, the furnace must be tilted backwards. The tilt cylinder is contracted fully causing the furnace to tilt backwards. The slag pours off the top of the steel bath and out through the slag door. The slag may be collected in pots or may be poured onto the ground where it will be removed using a bulldozer. The typical tilt angle for deslagging is 15–20°.

One newer design of furnace bottom has the furnace cradle sitting on a bogie wheel. A cylinder is mounted on an angle as shown in Fig. 10.7. In this configuration the taphole moves towards the deslagging side of the furnace during tapping. The required tilting force is greatly reduced for this configuration.

10.1.1.8 Furnace Roof

The furnace roof can be dome shaped or as is more common with water-cooled roofs used in modern practice, the roof resembles a shallow cone section. The roof consists of a water-cooled roof ring which forms the outer perimeter of the roof cage. This cage acts as part of the lifting structure for the roof. Water-cooled panels insert into this cage which has a cylindrical opening at the center. The refractory delta section is inserted to fill this opening. This delta section is designed to allow the minimum opening around the electrodes without risk of arcing between the electrodes and the water-cooled panels. The whole furnace roof is cantilevered off the roof lift column. Typically, roof and electrode support can be swiveled together or independently. The electrode stroke allows the electrodes to be swiveled with the roof resting on the furnace shell which allows for removal and replacement of the delta section without removing the roof. Typically, for a full platform design, a swiveling support with pivot bearing, bogie wheel and gantry arm is employed. For large furnaces, a roof lifting gantry is required. The swiveling can be achieved using either a

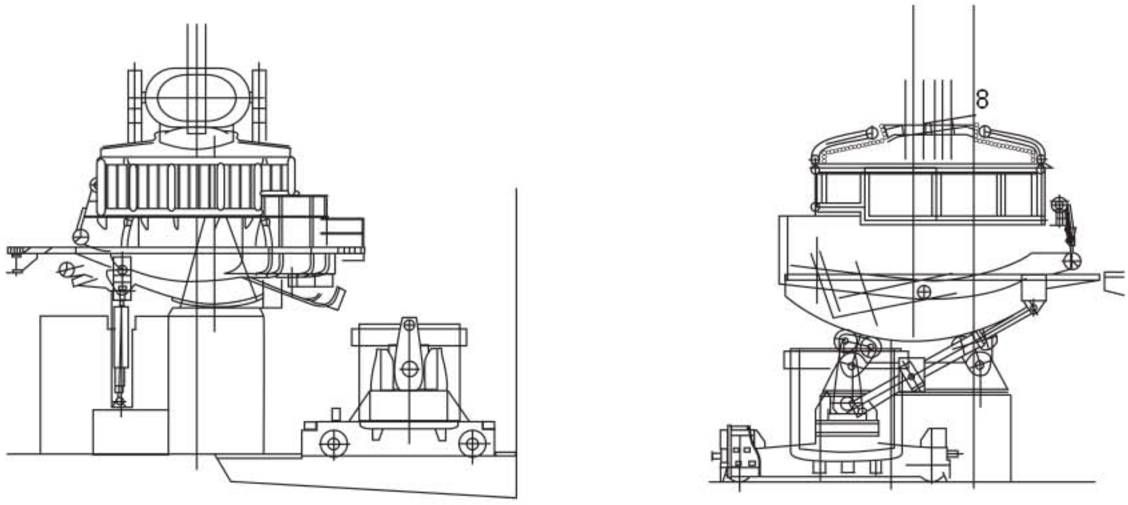


Fig. 10.7 Furnace tilting configurations. *(Courtesy of Danieli.)*

bogie wheel support or a thrust bearing. Twin shell furnace arrangements where the roof must pivot between furnace shells sometimes employ a fork shaped gantry. Typically most modern furnaces can raise and swing the roof in about 20–30 seconds. Roof swing speed is usually 4–5° per second. This minimizes the amount of dead time during charging operations and increases furnace melting availability. Typical roof configurations are shown in Fig. 10.8.

10.1.1.9 Electrode Arms and Lifting Column

Electrode control performance is limited by the lowest natural frequency in the positioning system. It is therefore very important to ensure sufficient stiffness in the columns with respect to torsion and bending. Several possible configurations for the guiding of the columns is shown in Fig. 10.9. The choice of configuration is usually specific to a particular operation. However, the main objective is to avoid friction in the roller system while arranging the roller design to be compact yet rigid. It is important to note that each arm must be capable of individual movement to allow for electrode regulation. For lateral movement of the electrodes a common swing mechanism is used. The conventional design uses a hydraulic cylinder to move the swing column. Some newer designs

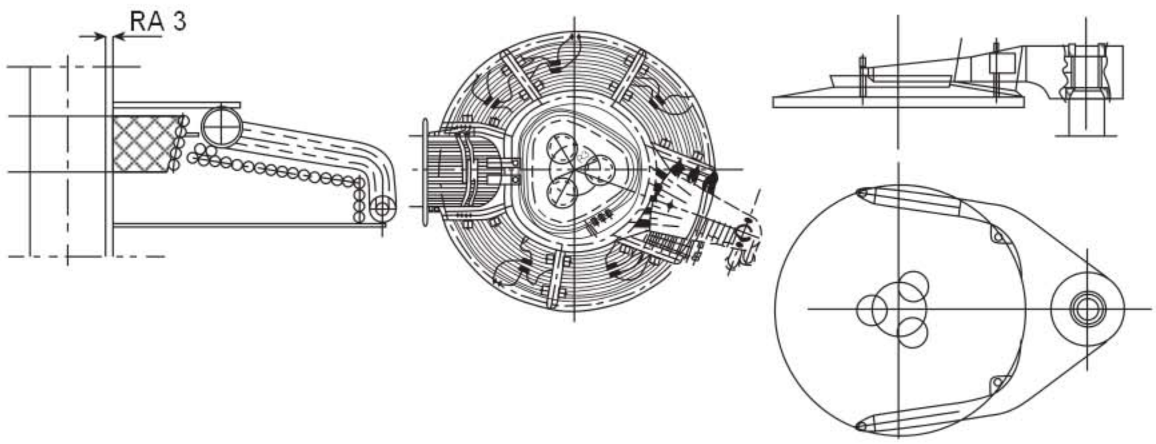


Fig. 10.8 EAF roof and roof lifting configurations. *(Courtesy of Danieli.)*

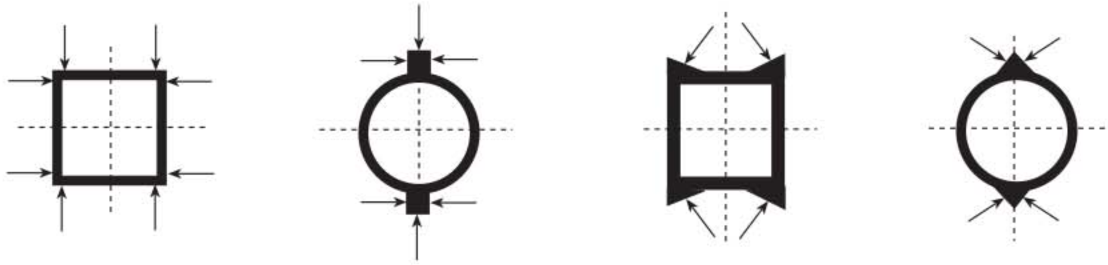


Fig. 10.9 Electrode arm column guiding configurations. *(Courtesy of Danieli.)*

now use a hydraulic motor inside the roller bearing. This cleans up the space around the cylinder but makes the mechanism less accessible for maintenance.

Several options are available for electrode arms in modern EAF operations. In conventional EAF design, the current is carried to the electrodes via bus tubes as shown in Fig. 10.10. These bus tubes usually contribute approximately 35% of the total reactance of the secondary electrical system.[4] Care must be taken to avoid induced current loops which can cause local overheating of the arm components. Thus the electrode clamping mechanism requires several insulation details.

Current conducting arms combine the mechanical and electrical functions into one unit as shown in Fig. 10.11. These arms carry the secondary current through the arm structure instead of through bus tubes. This results in a reduction of the resistance and reactance in the secondary circuit which allows an increase in power input rate without modification of the furnace transformer. Productivity is also increased. Current conducting arms are constructed of copper clad steel or from aluminium. Most new furnace installations now use current conducting arms.

Typical maximum electrode speed is approximately 30–35 cm per second when operating in automatic arc regulation mode. When operating in manual raise/lower mode the maximum speed is usually 15–20 cm per second.

10.1.1.10 EAF Secondary Electrical Circuit Components

The electric arc furnace secondary electrical circuit consists of several components including the delta closure, power cables, bus tubes, electrode holder(s), and the electrode(s).

Power exits the furnace transformer via low voltage bushings which are connected to the delta closure. The delta closure usually consists of either a series of copper plates, copper tubes or in some cases both. These are arranged so that the individual secondary windings of the transformer are joined to form a closed circuit. Usually the delta closure is contained in the transformer vault and protrudes through the vault wall to connect with the power cables. The delta closure is anchored using insulated material to minimize distortion which may be caused by vibration and/or electromagnetic forces. A delta closure is shown in Fig. 10.12.

The power cables are the only flexible part of the secondary electrical system. These allow for raising, lowering and swinging of the electrodes. Furnaces usually operate with between two and four

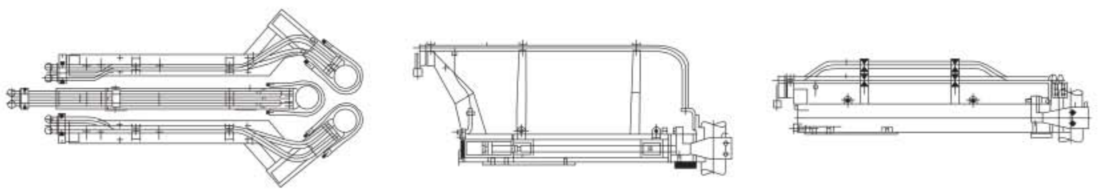


Fig. 10.10 Conventional bus tube electrode arms. *(Courtesy of Danieli.)*

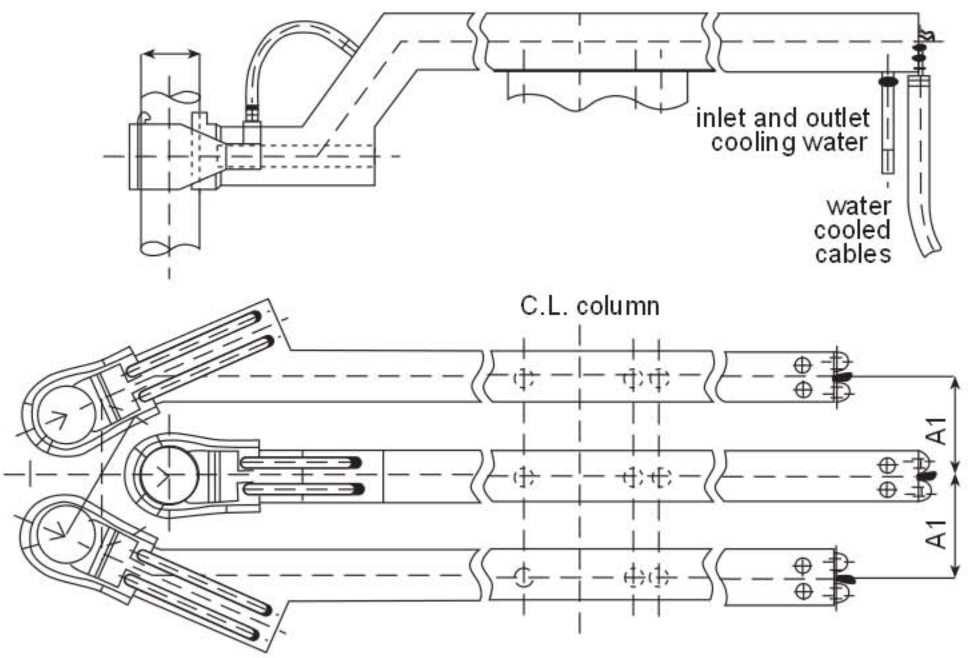


Fig. 10.11 Current conducting electrode arms. *(Courtesy of Danieli.)*

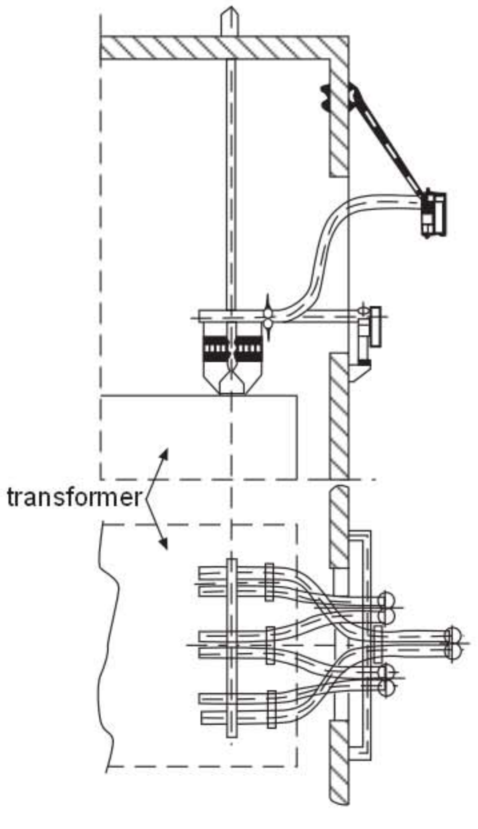


Fig. 10.12 Delta closure. *(Courtesy of Danieli.)*

cables per phase. The power cable generally consists of multiple copper wire strands which are soldered to copper terminals at either end of the cable. The copper cable is covered with a rubber compound which is secured at the terminals with nonmagnetic stainless steel bands. Water cooling is critical to the life of the cable. Usually, a dedicated closed loop cooling water circuit supplies cooling to all of the critical electrical components including the transformer, the delta, the power cables, bus bars and electrode clamp. Cooling water requirements range from 11–95 $l\ min^{-1}$ depending on the cable size.[5]

The power cables are attached to bus tubes which are mounted on the electrode arm. Bus tubes consist of rigid round copper pipe and provide the electrical connection between the power cables and the electrode holder. These bus tubes are also water cooled.

The electrode clamping mechanism provides the link between the electrode arm and the electrode. It is very important that the electrode be held firmly in place during furnace operations. For current conducting arms, it is not necessary to insulate the clamping mechanism from the arm because the arm is part of the electrical circuit. In the case of conventional electrode arms with bus tubes, the arm does not form part of the electrical circuit and must be electrically insulated from the clamp. Thus the electrode clamp is bolted onto the arm with a layer of insulation separating the two. The clamp casing which has contact shoes ensures that the electrode is making electrical contact. The copper bus bars which are supported on the arm are connected to the contact shoe. Clamping systems can be mounted external to the mast arm or can be mounted inside the arm. Most are now mounted inside the arm as this provides protection for the mechanism. A spring mechanism ensures that the shoes contact the electrode and a mobile yoke holds the electrode in place. The spring assembly is retracted using a hydraulic cylinder. This allows for release of the electrode for slipping operations or to remove an electrode column.

Fig. 10.13 Electrode holders. *(Courtesy of Fuchs.)*

The traditional electrode holder is annular or horseshoe-shaped and consists of a single piece copper construction. Some more recent designs have incorporated copper contact pads attached to a steel or nonmagnetic steel body. Electrode heads and contact pads are the most common failure point in the secondary power system. Electrode contact surface failures are usually related to dirt buildup between the contact face and the electrode. Insufficient clamping pressure can also contribute to failures. Contact pads/electrode holders should be changed out and cleaned during scheduled maintenance downtimes. Fig. 10.13 shows an electrode holder.

Water cooling is integral to service life for electrode holders. Several designs are common for electrode holders, one of the most common having a cast-in pipe which allows circulation of the water. Contact pads are usually castings or forgings with drilled holes to provide the cooling water channels.

Clamping force is a function of the electrode diameter. A general rule of thumb is that the clamping force in Newtons equals 850 times the square of the electrode diameter, in inches. Typical values quoted by vendors are 235 kN for 16 in. diameter electrodes and 460 kN for electrode diameters of 24 in.[6]

10.1.1.11 Electrode Spray Cooling

It has been recognized for some time that sidewall oxidation of the graphite electrodes is related to heating of the electrode surface. Because the current flow is down through the electrode into the bath, only the portion of the electrode below the electrode holder will heat up. Some early methods for reducing the amount of heating of the electrode involved a water-cooled graphite segment located at the clamping level. This method did reduce the amount of sidewall oxidation but required changing out the whole column once the bottom graphite segment was consumed. Other methods for reducing sidewall consumption involved special coatings which would oxidize at a higher temperature.

The most practical solution has now been adopted by most high powered furnace operations. This involves installation of a spray ring which is installed at the bottom of the electrode holder. The concept is that water will run down the surface of the electrode and will evaporate once it enters the furnace. Ideally, the water should be evaporated within the roof port where it will also provide some protection to the delta refractory. The required flow rate will vary depending on furnace operating conditions but typical flow rate for a 24 inch electrode is 7–15 $l\ min^{-1}$ per electrode. Typical savings in electrode consumption amount to 5–15%. This is however, highly dependent on the type of furnace operation (i.e. DC, high impedance AC, etc.).

10.1.1.12 Shell Openings

Several openings are usually provided for furnace operations. The most obvious are the three electrode ports which allow the electrodes to penetrate into the furnace through the roof. In addition, a fourth hole is provided in the furnace roof to allow for extraction of the furnace fume. A fifth hole may be provided for several reasons such as continuous DRI/HBI feed, coal injection or lime injection. These holes all occur high up in the furnace and as a result do not affect air infiltration into the furnace as much as lower openings.

The lower openings in the furnace include the taphole which is filled with sand and the slag door. The slag door was originally provided to allow decanting of the slag from the furnace. In most modern operations, it is also used to provide access to the furnace for oxy-fuel burners and oxygen lances. Several ports may also be provided around the circumference of the furnace shell for burners. Occasionally, an opening may be provided high up on the furnace sidewall to allow a water-cooled decarburization lance access to the furnace. Other openings may be provided low in the furnace sidewall or actually in the furnace hearth to allow for injection of inert gases, oxygen, lime or carbon.

10.1.1.13 Auxiliary Equipment

Auxiliary equipment is usually associated with injection of solids and gases into the furnace. Oxy-fuel or air-fuel burners are used to provide additional heat to the cold spots within the furnace as shown in Fig. 10.14. Typically, the cold spots occur between the electrodes and also in the nose of

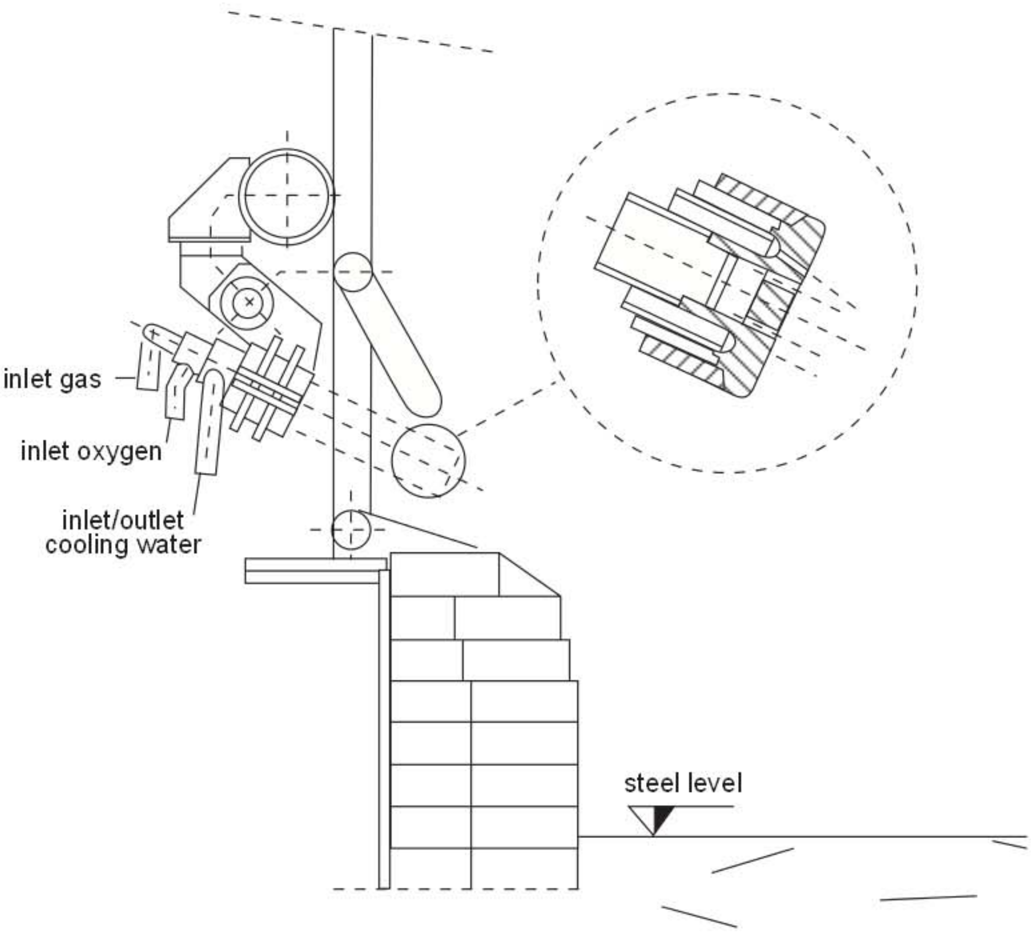


Fig. 10.14 Sidewall oxy-fuel burner. *(Courtesy of Danieli.)*

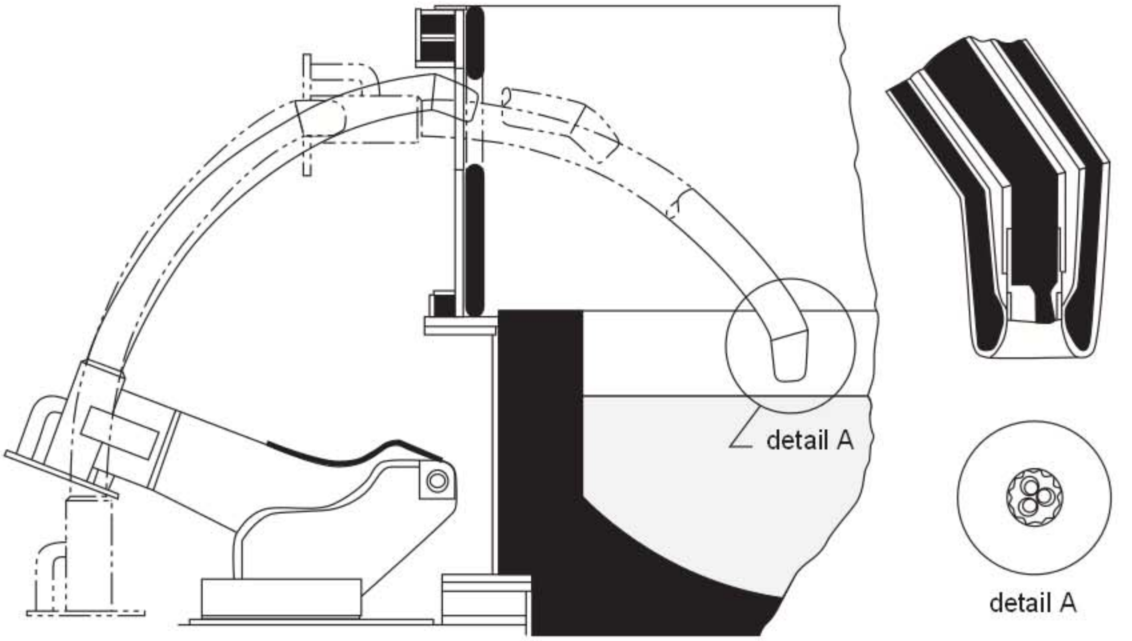


Fig. 10.15 Unilance. *(Courtesy of Empco.)*

the EBT furnace because it is further away from the electrodes. Burner efficiency is highest when the scrap is cold and decreases as the scrap heats up. This is described in more detail in the section on EAF operations. Burners are usually constructed of high alloy steel or stainless steel and are water cooled. The burner tips are frequently constructed out of copper. Burners must be capable of withstanding high heat fluxes especially early in the meltdown cycle when it is likely that the flame will flash back if its passage is obstructed with scrap. Most sidewall burners are stationary. Slag door burners tend to be retractable. The so-called banana lance supplied by Empco actually follows the scrap down into the furnace as it melts as shown in Fig. 10.15. This unit can operate in either oxy-fuel burner or oxygen lance mode. Burners are operated at different firing rates depending on the furnace operating phase. A minimum low firing rate is maintained at all times to ensure that the burner is not plugged by splashed slag.

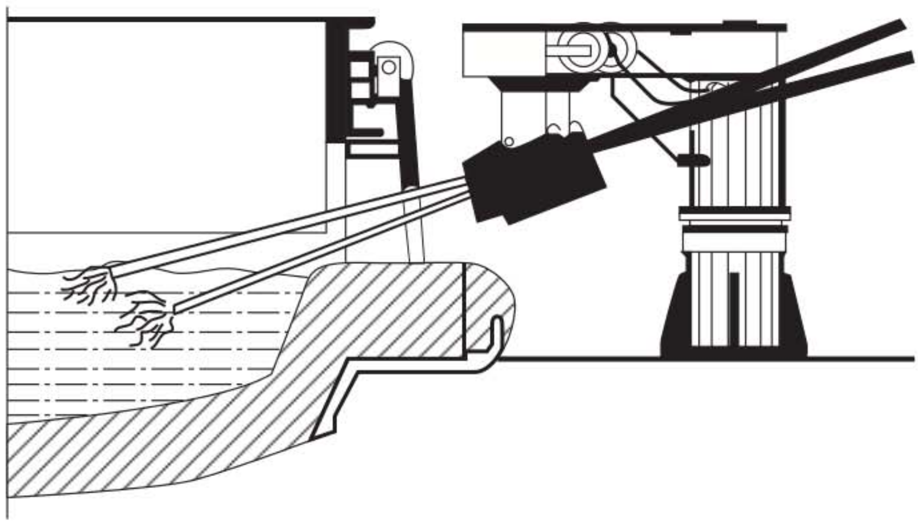


Fig. 10.16 Operation of consumable oxygen lance. *(Courtesy of Badische Stahl Technology.)*

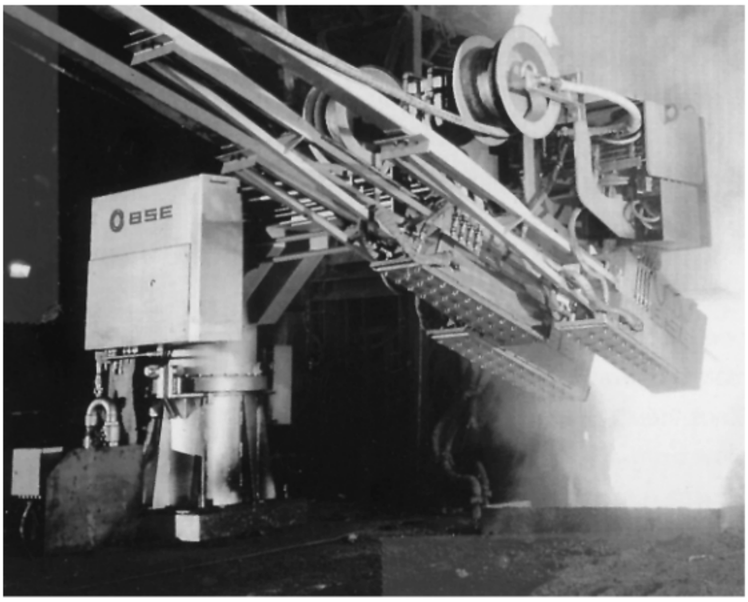


Fig. 10.17 Consumable oxygen lance manipulator. *(Courtesy of Badische Stahl Technology.)*

Oxygen lances come in many configurations but generally fall into two categories; consumable pipe lances and non-consumable water-cooled lances. Many oxygen lances are now multi-port lances and are also capable of injecting lime and carbon. Consumable oxygen lances are capable of actually penetrating the slag/bath interface as shown in Fig. 10.16. The consumable pipe burns back during operation and periodically, a new section of pipe must be inserted. A consumable lance manipulator is shown in Fig. 10.17. Water-cooled lances do not actually penetrate the bath. Instead they inject oxygen at an angle through the slag at high pressure so that the oxygen penetrates the bath. A water-cooled oxygen lance manipulator is shown in Fig. 10.18.

Another important auxiliary function is sampling the bath and obtaining a bath temperature. This information is critical for the furnace operator so that adequate bulk additions can be made at tap.

Fig. 10.18 Water-cooled oxygen lance. *(Courtesy of Fuchs.)*

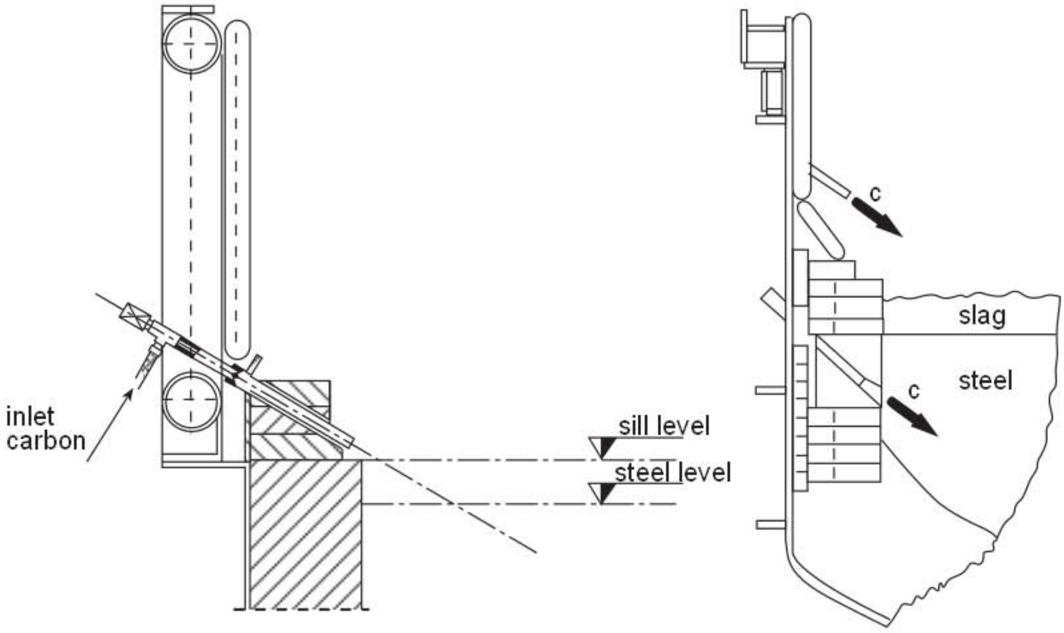


Fig. 10.19 Carbon injector configurations, submerged and above the bath. *(Courtesy of Danieli.)*

Typically, bath temperature measurements are taken manually using a lance with a disposable cardboard thermocouple tip. The lance is connected to an electronic display which converts a voltage signal into a temperature. Combined function units are also capable of measuring bath oxygen levels. The bath sample is usually taken manually using a lollipop type sampler. The sample is cooled and sent to the lab for chemical analysis. Some recent installations have automated bath temperature measurement and sampling using an automated lance. This reduces the time which operators have to spend out on the shop floor and also maintains a consistent sample location.

Many operations are now providing for carbon and lime injection at multiple points around the furnace shell. It was recognized that with injection solely through the slag door, slag foaming tended to be localized. By providing multiple injectors, it is possible to maintain more uniform slag foaming across the entire bath. Several possible configurations are shown in Fig. 10.19.

10.1.1.14 Bottom Stirring

For conventional AC EAF scrap melting there is little natural convection within the bath. As a result high temperature and concentration gradients can exist within the bath. These can lead to increased energy consumption, reduced reaction rates and over or under reaction of some portions of the bath. Bath stirring can help to reduce these gradients and can be accomplished either electromagnetically or by using inert gas injection to stir the bath. Most modern operations opt for the latter.

Typically, the elements used to introduce the gas into the bath can be tuyeres, porous plugs or combinations of porous plugs with permeable refractory rammed into place over top of the plug. Noncontact elements (i.e. those which do not actually contact the bath directly) have the advantage that they can be operated at higher turndown ratios without the fear of plugging. The choice of gas used for stirring seems to be primarily argon or nitrogen though some trials with natural gas and with carbon dioxide have also been attempted. Stirring elements are mounted within the refractory lining of the EAF. The number of elements used can vary greatly based on the design and gas flow rate but is usually in the range of one to six. The size of the furnace and its design will also create different demands for the number of stirring elements to be used.

10.1.1.15 Special DC Furnace Design Considerations

DC furnaces have probably been the biggest innovation in EAF technology in the past ten years. The concept of operating an EAF on DC current is not new but only recently did the costs for rectification units drop to the point where DC furnaces became economical. DC furnaces have several unique requirements over AC furnaces in addition to the obvious differences in electrical power supply which are discussed in the section on furnace electrics. A typical layout for a DC furnace is given in Fig. 10.20. DC furnaces have only one electrode mast arm and a single graphite electrode. This electrode acts as the cathode. Thus the top of the furnace is much less cluttered for the DC case and in general has fewer components to maintain as compared to AC designs. The DC furnace however, requires a return electrode, the anode, to complete the electrical circuit. This anode is commonly referred to as the bottom electrode because it is located in the bottom of the furnace shell. Several different designs exist for the bottom return electrode including metal pin return electrodes with non-conductive refractories, billet electrode, metal fin electrodes, and conductive bottom refractory. These are shown in Fig. 10.21.

In the case of current conducting refractory contact, the refractory lining at the center of the furnace bottom acts as the anode. The bottom has a circular flange which rests inside a circular channel which is welded to the furnace shell. Inside the channel, the flange is supported by fiber reinforced ceramic blocks. The space between the channel, support blocks and flange is filled with a refractory ramming compound. This isolates the bottom electrically from the rest of the furnace shell as shown in Fig. 10.22.

The spherical furnace bottom is constructed of high temperature steel. A circular copper plate is bolted directly to the furnace bottom. Four copper terminals extend down through the furnace bottom from the copper plate and connect to flexible cables which in turn are connected to the bus tubes. The conductive refractory bricks are installed over top of the copper plate. Heat flow from the bottom of the furnace (usually about 15 kWm^{-2}) is removed by forced air cooling. Due to the large surface area of the bottom electrode, the current density tends to be quite low, typically about 5 kAm^{-2}. However, it has been necessary in some installations to use nonconductive patching material in the center of the furnace in order to force the current to distribute more evenly

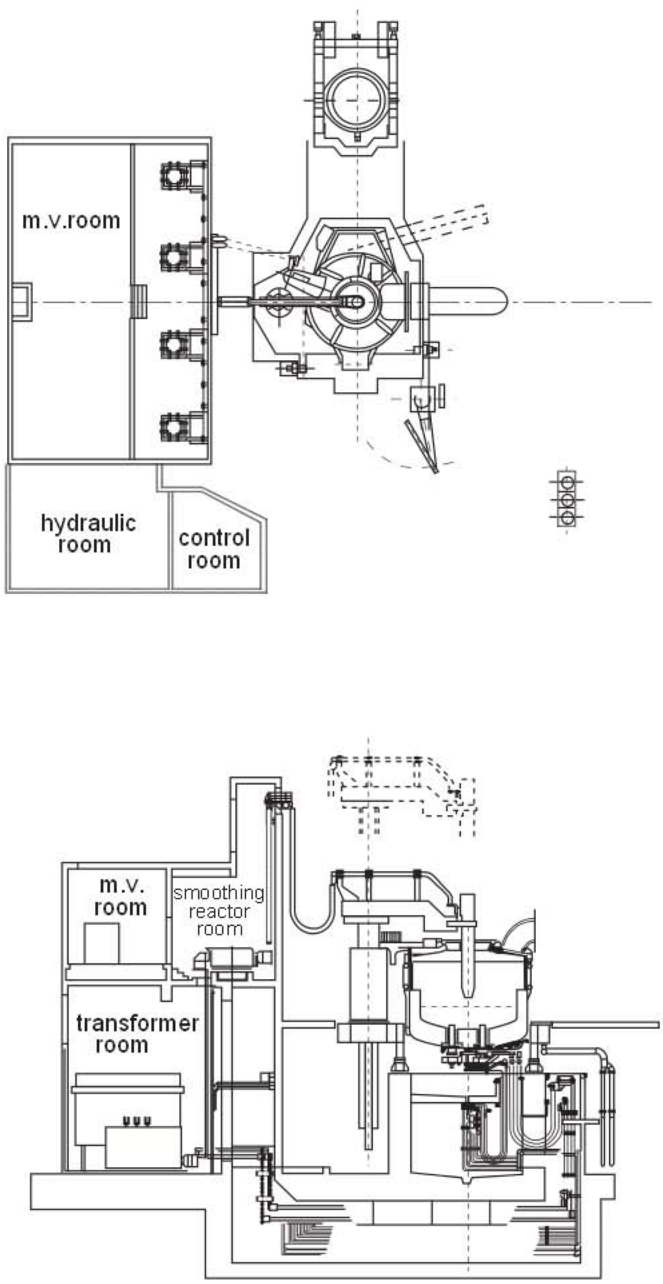


Fig. 10.20 Typical DC furnace configurations. *(Courtesy of Danieli.)*

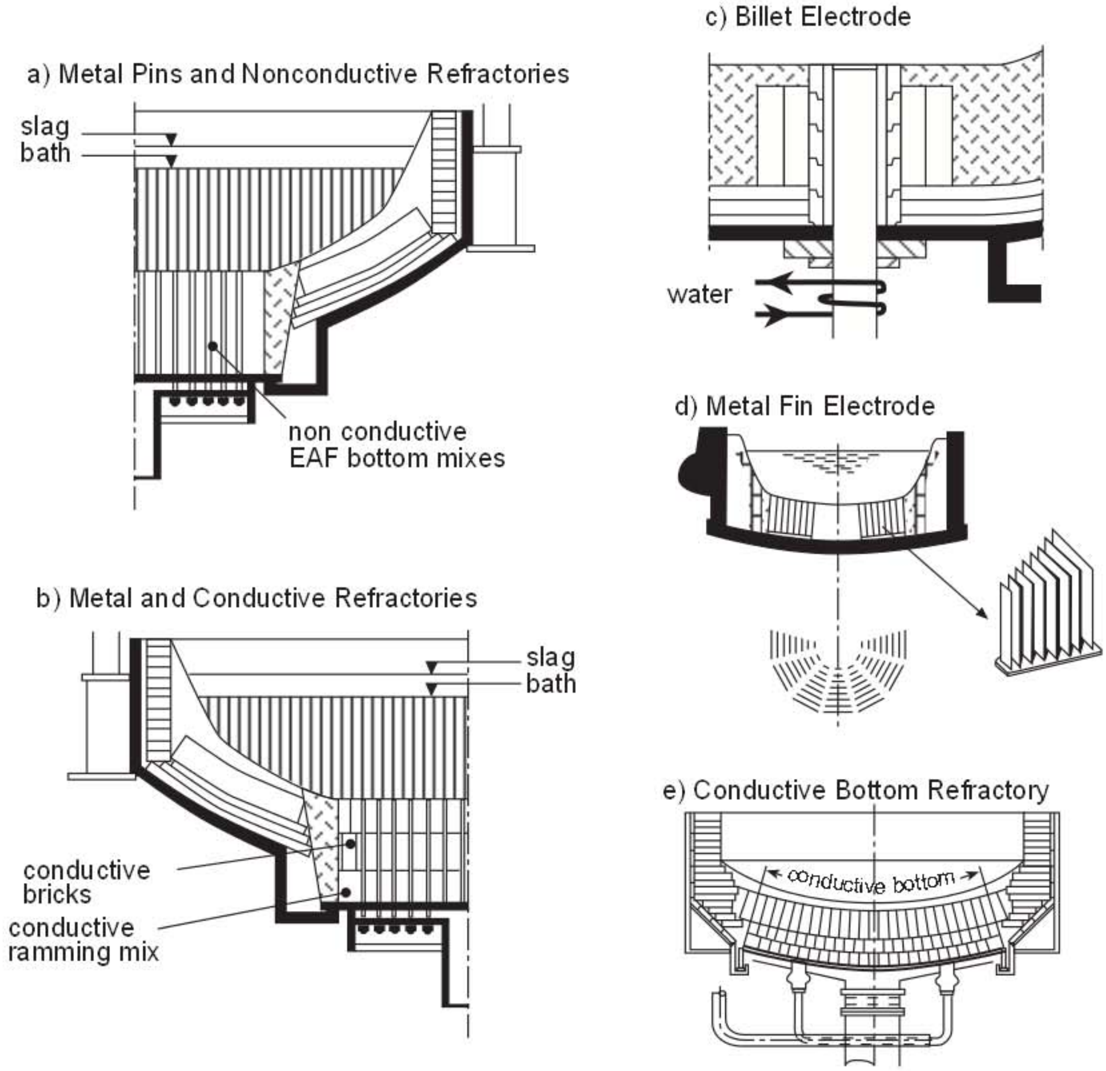


Fig. 10.21 DC furnace bottom anode configurations. *(Courtesy of ABB.)*

over the whole bottom. Failure to achieve proper distribution of the current resulted in hot spots in the center of the furnace.

The billet return electrode configuration employs from one to four large steel billets (on the order of 10–15 cm diameter but can be as large as 25 cm diameter) depending on the size of the furnace. Usually, the design aims for a current of 40–45 kA per bottom electrode. The billets are in contact with the bath at the top surface and will melt back. The degree to which the billet melts back is controlled by water cooling. The billet is inserted into a copper housing through which cooling water is circulated. By providing sufficient cooling, it can be ensured that the billet will not melt back completely. Thermocouples monitor the bottom billet temperature and the cooling water temperature. An insulating sheath isolates the copper housing from the billet. The billet is connected to a copper base. The copper base provides the connection to a power cable. Fig. 10.23 shows the Kvaerner-Clecim billet electrode.

The pin type of return electrode uses multiple metal pins of 2.5–5.0 cm in diameter to provide the return path for the electrical flow. These pins are configured vertically and actually penetrate the refractory. The pins extend down to the bottom of the furnace where they are fixed in position by two metal plates. The bottom ends of the pins are anchored to the lower power conductor plate. The

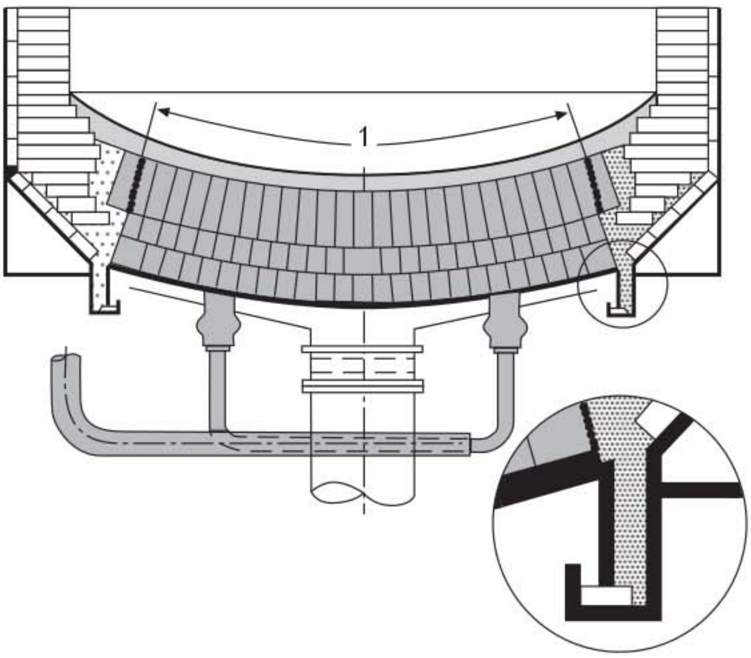


Fig. 10.22 Conductive refractory bottom electrode. *(Courtesy of ABB.)*

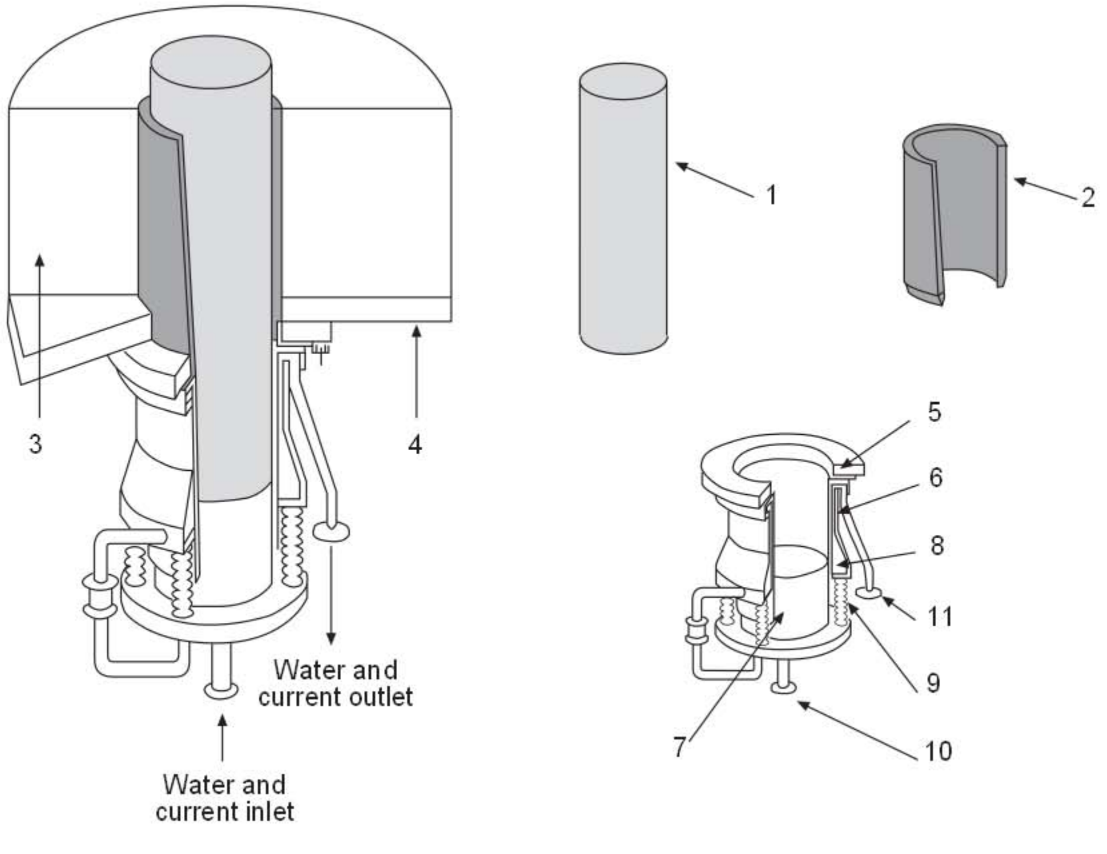


Fig. 10.23 Billet anode. *(Courtesy of Kvaerner Metals.)*

1. steel billet
2. special refractory
3. hearth
4. bottom shell plate
5. insulation material
6. copper sleeve
7. copper cap
8. water jacket
9. compensator
10. water and current inlet
11. water and current outlet

bottom contact plate is air-cooled and is located in the center of the furnace bottom. The top portion of the pins are flush with the working lining in the furnace. The pins are in direct contact with the bath, and melt back as the working lining wears away. A return power cable is attached to the bottom conductor plate.

An extensive temperature monitoring system is provided to track lining wear and bottom electrode life. This enables scheduled change out of the bottom electrode. The integral cartridge design which has evolved allows for quick change out of the bottom electrode over a scheduled eight hour maintenance outage. The sequence of change-out steps is as follows:

1. Following the last heat prior to change-out, the slag is drained from the furnace and refractory surfaces are sprayed with water to accelerate cooling.
2. Electrical connections to the bottom electrode are decoupled and thermocouples are disconnected.
3. The integral cartridge is pushed up by six hydraulic cylinders which are located around the perimeter of the bottom contact plate.
4. Once the cartridge is broken free of the furnace bottom, it can be removed by crane.
5. A new cartridge is then lowered into place and electrical connections are recoupled.
6. A small amount of scrap is charged and an arc is struck to test the system. Following a successful test, the furnace is charged with scrap and melting is commenced at reduced electrical current. Once a liquid heel is established, regular operation is resumed.

A typical anode removal device is shown in Fig. 10.24. Fig. 10.25 shows a typical refractory lining for a UNARC furnace.

The steel fin return electrode uses steel fins arranged in a ring in the furnace bottom to form several sectors. Each sector consists of a horizontal ground plate and several welded on steel fins

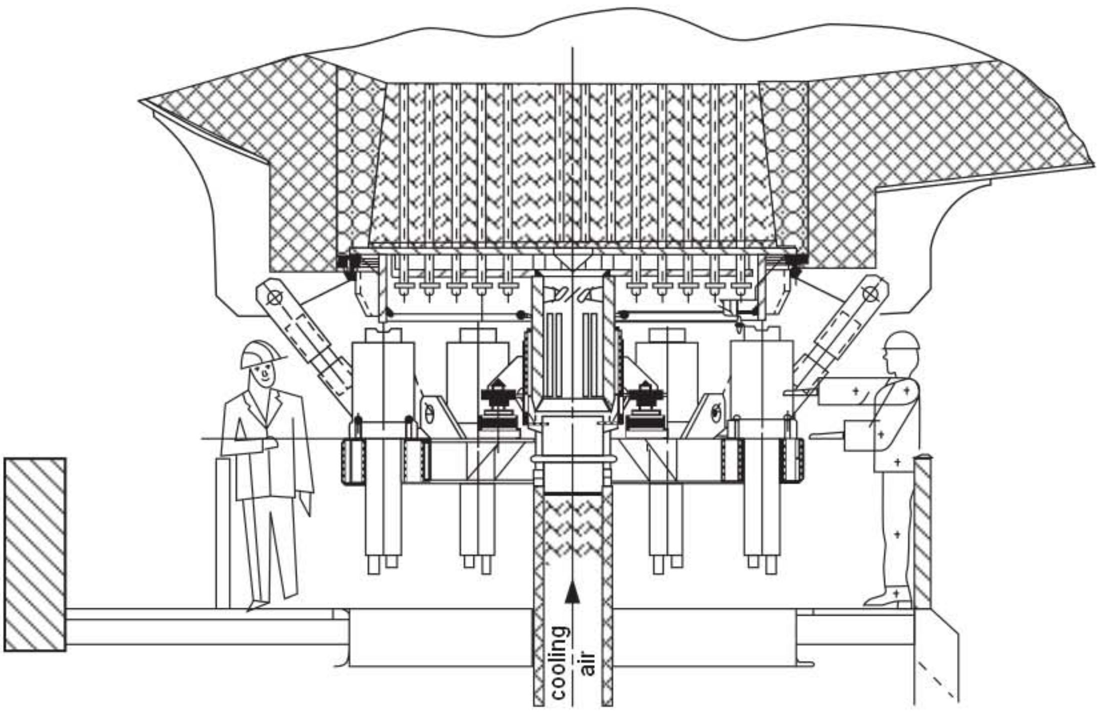


Fig. 10.24 Anode removal device. *(Courtesy of SMS GHH.)*

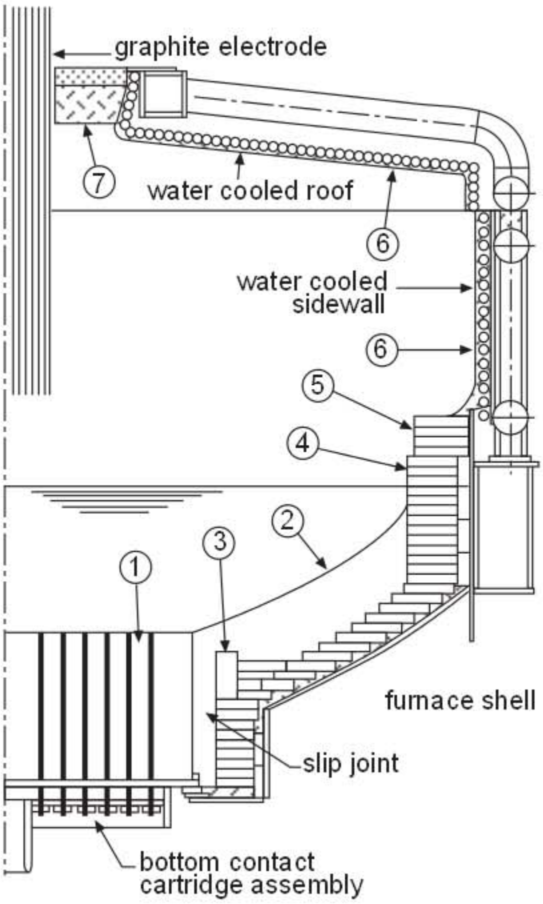


Fig. 10.25 Refractory details of UNARC furnace. *(Courtesy of SMS GHH.)*

which protrude upwards through the refractory. The fins are approximately 0.16 cm thick and are about 9 cm apart. The sectors are bolted onto an air-cooled bottom shell which is electrically insulated from ground and is connected to four copper conductors.

Most DC furnaces are operated with long arcs, typically two to three times those encountered in conventional UHP furnace operations. As a result, many furnace manufacturers specify higher water flow rates for water-cooled panels. Some typical values proposed by Fuchs Systems Inc. are 160 l min^{-1} m^{-2} (3.90 gpm ft^{-2}) for a sidewall panel and 180 l min^{-1} m^{-2} (4.38 gpm ft^{-2}) for a roof panel.

10.1.2 EAF Refractories

Refractories are materials which withstand high temperature without a significant change in chemical or physical properties. Refractory materials are very important to EAF operation as they allow for containment of the liquid steel in the furnace hearth without damaging the furnace structure. Refractories in general are discussed in Chapters 3 and for EAF use are discussed in Section 4.3.

10.1.2.1 Steelmaking Refractories

The electric furnace requires a variety of refractory products. Most important are the refractories in direct contact with the steel. Today's electric furnaces and EBTs generally use magnesite or magnesite carbon products in the steel contact zones. Specialized refractories with good thermal shock resistance are generally used in the roof/delta, the taphole, and the spout or runner of the furnace. The following descriptions give general refractory recommendations.

10.1.2.1.1 Sub-bottom Magnesite or tar-impregnated magnesite brick are generally used in the sub-bottom due to good resistance to steel and slag in the event the working hearth is penetrated. These refractories are expected to last six months, one year, or even several years if the bottom hearth material is well maintained.

10.1.2.1.2 Hearth Most electric furnace operators prefer a monolithic magnesite hearth which is easy to install and maintain. Monolithic hearth materials are generally dry, vibratable high MgO materials which develop strength by sintering in place during burn-in of the bottom. These hearth materials generally have impurity oxides, like iron oxide, to facilitate sintering. Other operators prefer all brick hearths, where the initial construction is totally brick, and monolithic materials are only used to repair holes in the brick hearth after some period of operation. Modern EAF bottom designs often include gas stirring elements which require porous refractory materials or tuyeres through dense refractories to deliver the stirring gas to the molten steel. These stirring gases can cause erosion of the hearth refractories and require regular maintenance.

10.1.2.1.3 Lower Sidewall The lower sidewall generally uses the same brick quality as the sub-hearth, high MgO, plain or tar-impregnated. These brick form the base for the slagline and upper sidewall and are protected by the banks of the hearth material. They are replaced along with the monolithic hearth.

10.1.2.1.4 Slagline The slagline area of most furnaces uses magnesite carbon brick or tar impregnated magnesite brick. Refractory selection in the slagline must be carefully coordinated with expected slag chemistries. Carbon steel shops generate lime rich, $FeO–SiO_2$ slags with a 2:1+ lime:silica ratio, which demand basic refractories. In stainless operations, a more neutral fused grain magnesite chrome brick may be utilized.

10.1.2.1.5 Sidewall/Hot Spots Above the slagline, the refractory lining sees high temperature arc flare, slag splashing, scrap impingement, and flame impingement from burners or lances. Refractoriness, strength, and thermal shock resistance are the critical parameters for electric furnace sidewall refractories. In furnaces with water-cooled panels, magnesite carbon brick is the most common choice, with fused grain magnesite carbon brick in the hot spots.

10.1.2.1.6 Roof and Delta If the roof is water-cooled, a precast delta of high alumina or alumina chrome materials is a likely choice. Greater economy can sometimes be obtained by bricking the electrode ports and ramming a high alumina plastic in the balance of the delta section. Roof deltas are subjected to extreme thermal shock during roof swings for charging. Basic materials would have better resistance to the furnace atmosphere, but cannot generally withstand the thermal cycling. For complete refractory roofs, 70–80% alumina brick annular rings composing about two-thirds of the roof are most common; with high alumina plastic or alumina chrome plastic rammed in the delta section, again with high purity alumina or alumina chrome brick electrode rings.

10.1.2.1.7 Taphole For conventional tapping electric furnaces, a taphole module or taphole block with a predrilled hole is the most common construction. Other operators will ram high alumina or basic ramming mixes around a steel pipe to form their taphole, while others will gun basic gunning mixes around a steel pipe for the initial taphole. Some taphole designs will last the life of the furnace campaign, while other operators regularly maintain their taphole by gunning around an inserted metal pipe. For EBTs, dense magnesite or magnesite carbon shapes form the vertical taphole. These are often encased within larger rectangular shapes to form a permanent taphole well in the nose of the furnace. Again, some EBT tapholes last the life of the furnace while other operators patch the furnace with resin bonded patching mixes or perform regular maintenance by installing new taphole shapes as needed. EBT tapholes require a well-fill sand between heats to prevent steel freezing in the taphole; this sand is usually olivine, magnesite or chromite based.

10.1.2.1.8 Spout or Runner The spout or runner of conventional electric furnaces may be brick shapes, rammed plastic, or a large, preformed spout shape. Due to the extreme thermal shock on this runner or spout, high alumina materials are preferred. Occasionally, a combined basic/high alumina system will be utilized to maximize performance.

10.1.2.1.9 Refractory Considerations for DC Furnaces DC electric furnaces have special refractory requirements due to the fact that the return electrode is usually installed in the bottom of the furnace (some DC furnaces use an alternative arrangement with two graphite electrodes).

In the case of a current conducting bottom, the refractory lining at the center of the furnace bottom acts as the anode. A copper plate is usually connected below the conductive refractory and the return copper bus bar is connected to the plate. In this case special requirements for the refractory are low electrical resistance (preferably <0.5 milliOhms per meter), low thermal conductivity, and high wear resistance.

A typical configuration uses a six inch thick working lining consisting of carbon bonded magnesia mixes containing 5–10 wt% carbon. These materials can be installed either hot or cold. Below the working lining a three layer magnesia carbon brick is installed. The residual carbon content of the bricks ranges from 10–14 wt%. With regular maintenance, this bottom electrode configuration has achieved a bottom life of up to 4000 heats.

The billet return electrode configuration employs from one to four large steel billets (on the order of 10 inches in diameter) depending on the size of the furnace. The billets are embedded in the bottom refractory. The billets are surrounded with a basic refractory brick. The remainder of the hearth is rammed with a special magnesia ramming mix. Magnesia ramming mix is used to maintain the brick area around the electrode. This return electrode configuration has achieved in excess of 1500 heats on the furnace bottom.

The pin type of return electrode uses multiple metal pins of 25–50 cm in diameter to provide the return path for the electrical flow. These pins actually penetrate the refractory down to the bottom of the furnace where they are attached to a metal plate. Dry magnesia ramming mix is used for the entire hearth lining. This mix is rammed between the metallic pins. Alternatively magnesia carbon brick can be used in the area around the anode. This helps to improve the furnace bottom life but is more costly. Typical bottom life ranges from 2000 to 4000 heats depending on the refractory materials used.

The steel fin return electrode uses steel fins arranged in a ring in the furnace bottom to form several sectors. Each sector consists of a horizontal ground plate and several welded steel fins which protrude upwards through the refractory. Dry magnesia ramming mix is used between the fins. The hearth is also lined with this material.

The important points to consider during installation are refractory zoning pattern, hearth contour, slagline location, furnace steel capacity, taphole location, taphole size and angle, roof/delta orientation, expansion allowances, burner port location, slag door construction, bottom stirring elements, and DC furnace bottom electrode.

10.1.2.2 Electric Furnace Lining Installation

Typical procedures for a complete new electric furnace or EBT lining follow.[7]

10.1.2.2.1 Sub-bottom and Monolithic Hearth The furnace must be level and the shell cleaned from all debris prior to starting. It helps to locate the exact center of the furnace and to punch a mark in the bottom center of the shell. The preferred construction is rectangular magnesite brick laid on flat using a basic granular material as fill to provide a flat surface against the rounded steel shell. After locating the EBT taphole seating blocks, any bottom stirring elements and allowing for the DC furnace bottom electrode, the first course of sub-hearth brick is laid dry, tight and LEVEL, and a dry magnesite mortar is swept into the brick joints. Additional fill material is placed around the perimeter of this first course and leveled out. The second course is laid at a 45° angle to the first course and dry magnesite mortar again is swept into the joints. This process is repeated for three or four flat courses. (There is an alternate, less preferable, sub-hearth design which lays two to four flat courses which follow the curved contour of the steel shell. This may be used in furnaces where the shell has a small spherical radius and gives more uniform thickness in the monolithic hearth material.) At the proper elevation, key shaped brick are used to begin the first stadium course. It is preferable to start with the largest ring one inch from the shell and work towards the center. To close the ring, a key brick is cut on a brick saw to the exact dimensions required to close the ring. If the cut shape will be less than a half brick, two cut shapes should be utilized. The void at the end of the course up to the shell is filled with granular magnesite material. The next stadium ring is installed in similar fashion. The contour of the stadium hearth shown on the drawing must be carefully followed to leave sufficient room to add the monolithic hearth material at the appropriate thickness.

For EBT furnaces, it is even more critical to follow the refractory bottom drawing exactly. There are partial rings of brick with varying radii extending out into the nose section of the furnace which must be kept level. One way to facilitate this is to drill a hole in the top flat course of bottom brick in the exact center of the furnace and then utilize a broomstick with a nail on it extending up from this center brick as a mandrel to draw circles and arcs for the stadium rings and partial rings extending into the nose.

10.1.2.2.2 All Brick Hearths If a monolithic hearth material is not utilized, the final course or final two courses in the all-brick hearth are laid in rowlock (on edge) or soldier (on end) construction. Rowlock or soldier construction gives much greater brick-to-brick contact and minimizes heaving of the hearth in service. Again, all courses in the hearth and stadium are laid dry and swept with magnesite mortar to fill the joints.

10.1.2.2.3 Slagline and Sidewalls Once the stadium rings are completed, the slagline brick are installed course by course using the same keying up concept utilized for the stadium rings. The slagline should also be installed in excess of one inch away from the steel shell to permit thermal expansion without spalling or heaving the brick. Brick rings or partial rings should be continued up into the sidewall and hot spots until the water-cooled panels or the top of the furnace is reached.

10.1.2.2.4 Door Jambs The door jambs are a critical design area for the refractory lining. Several operators simply utilize regular key shapes in interlocked courses as their door jamb, and they have a successful practice. Other operators utilize special door jamb shapes which have greater surface area for better interlocking between courses and a sharper angle which opens up the door opening and eliminates or reduces refractory damage when slagging off. Still other door jam designs involve brick or precast shape assemblies which are welded or bolted to the steel shell. These are generally installed first and the slagline and sidewall brick laid directly against these assemblies, with sidewall rings keyed up halfway between the door and the taphole.

10.1.2.2.5 Taphole Conventional tilting electric furnaces generally use taphole module shapes set with a crane at the proper elevation prior to bricking the slagline and sidewalls. One alternative is to leave an opening in the sidewall rings and then ram or gun around a steel pipe forming the taphole. This pipe is then melted out on the first heat. Refractory taphole shapes can also be utilized in this same manner with monolithic material holding them in place at the proper elevation and angle.

10.1.2.2.6 Roofs and Deltas The electric furnace roof or the delta section in a water-cooled roof are generally installed in a refractory reline area, and a finished roof is waiting for the furnace to be rebuilt. With water-cooled roofs and precast deltas, a castable or plastic refractory is often placed around the perimeter of the precast shape to lock it in place against the water-cooled roof. For brick refractory roofs, a roof form is required (generally concrete) which creates the appropriate dome shape for the inside contour of the roof. The mandrel(s) are set for the electrodes. Triple tapered electric furnace roof shapes are laid in concentric rings against the roof ring for the outer two-thirds of the roof. Often partial rings of brick are laid in a wedge pattern between electrodes. Electrode ring brick surround each mandrel and are held in place with steel bands. A castable or plastic is cast or rammed into place in the cavity between the outer rings and electrode ring brick.

10.1.2.2.7 Monolithic Hearth Most electric furnace operators use a monolithic hearth material. This is a high magnesite, self sintering product which is granular in nature. After installing any EBT taphole seating blocks, bottom stirring elements, and DC bottom electrode forms; monolithic hearth construction is started. The hearth material comes ready to use in large bulk bags; a crane holds the bulk bag in position over the brick subhearth while the bag is split and the material is shoveled into place. After two or three bulk bags are in the furnace, several workers using shovels or pitch forks repeatedly jab the granular material in order to remove air and densify the refractory hearth. As the material densifies, the workers further compact it with mechanical vibrators or by simply walking on the hearth to achieve the proper contour. Additional bags of hearth material are added and de-aired and densified until the final contour is reached (usually measured with chains or a form). The new hearth sinters in place during the initial heat.

10.1.2.2.8 Heatup Schedule On a new electric furnace lining with a completely new monolithic hearth, steel plates or light scrap are generally lowered by magnet into the bottom of the furnace to provide protection for the unsintered hearth material. After this cushioning scrap is in place the first bucket is charged and the arc is struck, utilizing a long arc to avoid boring down into the new monolithic bottom. The bottom is usually sintered after the first heat, although it is important to inspect the bottom and banks for any holes or erosion due to unsintered material leaving a void in the lining. Since the new electric furnace lining has very little moisture in it, no special precautions are required during the initial heatup, other than using a long arc to avoid eroding the bottom prior to sintering.

Electric furnace refractory linings are maintained by gunning, fettling, and patching with brick.

10.1.2.2.9 Gunning Maintenance Gunning maintenance consists of mixing water with a magnesite based gunning mix and spraying this mixture onto the refractory lining. Gunning is used to maintain hot spots, slagline erosion, tapholes, the door breast area, or any other portion of the lining which experiences selective refractory wear. Gunning material is usually a temporary measure and will have to be regunned in the same place within the next several heats. Gunning maintenance, while temporary, does offer balanced life by evening out the highly selective wear pattern in the electric furnace lining. That is, refractory wear in AC furnaces is usually greater in the sidewall closest to the mast electrode; and gunning this area maximizes overall lining performance. Most refractory gunned maintenance is done with a pressurized chamber batch gun of roughly 900–1800 kg capacity. This gun delivers dry material pneumatically to a water mixing nozzle, and the air pressure sprays the wet gunning mix onto the surface of the lining. The nozzle operator's training and skill is a factor in the quality of the gunned patch, and there is a tendency to empty the gun, which can increase refractory costs. Gunning can be automated by using a mechanical center-throw gunning device which shoots gun material in a circular pattern while suspended from a crane. This mechanical gunning is faster and easier on people, but often wastes material by placing it where it may not be needed.

Basic gunning mixes range from 40–95% MgO in quality. High temperature operations and high power furnaces generally use higher MgO content gun mixes; while moderate temperature operations utilize lower MgO content gun mixes. Carbon steel producers use between 1.4–7.5 kg of gun mix per ton of steel.

10.1.2.2.10 Fettling Maintenance Fettling maintenance is the technique used to patch holes in the monolithic bottom. A rapid sintering version of the granular hearth material, or the original product, are used for fettling. The dry material is shoveled or dropped by crane wherever there is a hole or divot in the monolithic bottom; or a mechanical chute suspended by crane delivers material onto the sloped banks of the hearth. Occasionally the magnet is used to level this patch material, which then sinters in place during the next heat. With this fettling technique, monolithic bottoms often last anywhere from three months to one year or longer.

10.1.2.2.11 Brick Patching After several weeks of operation, gunning maintenance becomes less efficient in maintaining the refractory lining. Most operators will then take the furnace cold and dig out anywhere from 30–80% of the sidewall and hot spots. The rubble is then removed from the furnace and workers install new brick in all areas of the hot spots and sidewall which were removed. Often the same refractory quality and thickness are reinstalled as was used in the initial lining. Alternatively, lesser quality or thinner linings are installed during this patch and slightly less lining life is anticipated from the patch as would be received from a new complete lining. The brick patch is completed by gunning an MgO gunning mix into all the voids and cracks in the patched brickwork.

Usually during a brick patch, the taphole is completely replaced or repaired with ramming mix or gunning material, if not replaced with brick work. Most electric furnace operators will develop a regular brick patch schedule for their electric furnace. This could be one or two intermediate patches for every complete sidewall job. Or, some operators have utilized the continuous patching concept where some brick are replaced every two weeks or even every week, and the complete lining is totally replaced only once or twice per year. These maintenance and patching decisions are usually dictated by the severity of the operating conditions as well as the company's economic maintenance philosophy.

10.1.2.2.12 Miscellaneous Refractory Maintenance Each EAF has unique features or conditions requiring specific refractory maintenance. Roof delta sections must be replaced at failure or on a regular schedule. Tapholes must be replaced or repaired when the tap time gets too short or slag carryover starts. Conventional EAF tapholes are usually replaced with a high MgO gunning mix shot around a steel pipe, while EBT tapholes are knocked out and a new assembly or one-piece tube inserted by crane from above. Bottom stirring elements and bottom electrodes each require specialized maintenance procedures which vary with design.

10.1.2.3 Operating Philosophy

The operational philosophy often has a significant impact on refractory performance. If maximum production is the key operating philosophy, decisions will be aimed at maximum production and less than optimum refractory maintenance can result. Maximizing production usually requires a high quality initial refractory lining that is expected to last a long time without much refractory maintenance. There is a crossover point where a lack of maintenance actually reduces productivity, so the maximum production philosophy often requires a delicate balance on refractory lining design and maintenance. The minimum cost philosophy is also common. Here the initial refractory lining is inexpensive, generic brands which are anticipated to last a long time through patching or gunned maintenance. Minimum cost linings generally have little zoning and often utilize the same brand and lining thickness throughout the furnace. This philosophy can be effective for low intensity operations.

10.1.2.3.1 Predictable Campaign This philosophy generally requires a set period between refractory relines or patches. In this scenario the electric furnace operator schedules full production for a two week or three week period, knowing the furnace lining will make it that long. Then a patch or reline is scheduled and another predictable campaign is begun. This philosophy requires close coordination and optimization of the refractory design with shop operations.

10.1.2.3.2 Minimum Downtime Philosophy This scenario is similar to maximum production, but without the same zeal for total production. Here the refractory lining must deliver predictable operations with very little maintenance, so a high quality initial lining is usually required. There is not much time for downtime or maintenance until the scheduled down turn or down day. Predictable maintenance is acceptable, but must be on a regularly scheduled basis.

10.1.2.4 Future Considerations

Electric furnace refractory requirements have changed significantly in the past several years and are faced with even more change as electric melting technology and processes progress. The increasing use of scrap substitutes, direct reduced iron (DRI) and hot briquetted iron (HBI) requires modified furnace linings. Post-combustion of carbon monoxide and growing use of oxy-fuel burners both place demands on furnace refractory performance. Increased water cooling in areas of the door jambs and burner openings will be required. There will be pressure to reduce refractory consumption of chromium containing products and to ensure that refractories from all steelmaking processes are recycled.

10.2 Furnace Electric System and Power Generation

10.2.1 Electrical Power Supply

An arc furnace requires a very high power. Each ton of steel melted requires on the order of 400 kWh (0.4 MWh), thus to melt 100 tons 40 MWh of electrical energy is required. For a meltshop producing at a rate of one million tons per year it is necessary to produce at the rate of 100 tons per hour or more. To reach this level, 40 MWh needs to be put into the furnace in about half an hour of power-on time to achieve the desired productivity. This means that the average power level in MW is on the order of 80 MW.

EAF meltshops typically require power levels in the 20–200 MW range, (enough for a small town). Such large power levels can only be supplied by the utility from their high voltage network, where voltages in the range 100–500 kV are present. This is a three-phase system (the three-phase choice, rather than some other number of phases, was made early in the 20th century) and for this reason the traditional AC furnace has three electrodes.

Inside the furnace the physics of electric arcs dictate arc voltages in the range 100–600 volts (AC furnaces); above these levels the arcs become too long to be manageable. In order therefore to obtain arc powers of the required MW it is necessary to operate with currents in the kiloampere range; typically 20–80 kA (AC furnaces).

The power flows from the utility's generators through their network and arrives at the steel plant at very high voltage and must therefore be converted to low voltage suitable for arcs. This task is performed by transformers.

10.2.1.1 Transformer System

For a variety of reasons the task of transforming the power from the kV level at the incoming utility line to the voltage level needed by the arcs is usually done in two stages. A first transformer (occasionally two transformers in parallel) steps the voltage down from the high voltage line to a medium voltage level which is generally standardized for each country, Fig. 10.26. In the U.S. this medium voltage is usually 34.5 kV, while in Europe, Japan and other areas the voltages are not very different, often 30–33 kV.

As the steel plant requires electrical power for other sectors, for example a caster or rolling mill, there will be several transformers connected at the 34.5 kV level and thus it is common to have a small 34.5 kV substation within the meltshop. This arrangement is one of the reasons for making the transformation from utility to arc furnace in two steps.

From the 34.5 kV busbar the arc furnace is powered by a special, heavy duty furnace transformer. The secondary voltage of this furnace transformer is designed to allow operation of the arcs in the desired range of arc voltages and currents. However, because there are varying requirements of arc voltage/current combinations through a heat—for boredown, melting and refining—it is necessary to have a choice of secondary voltages. The furnace transformer is equipped with a tap changer for this purpose.

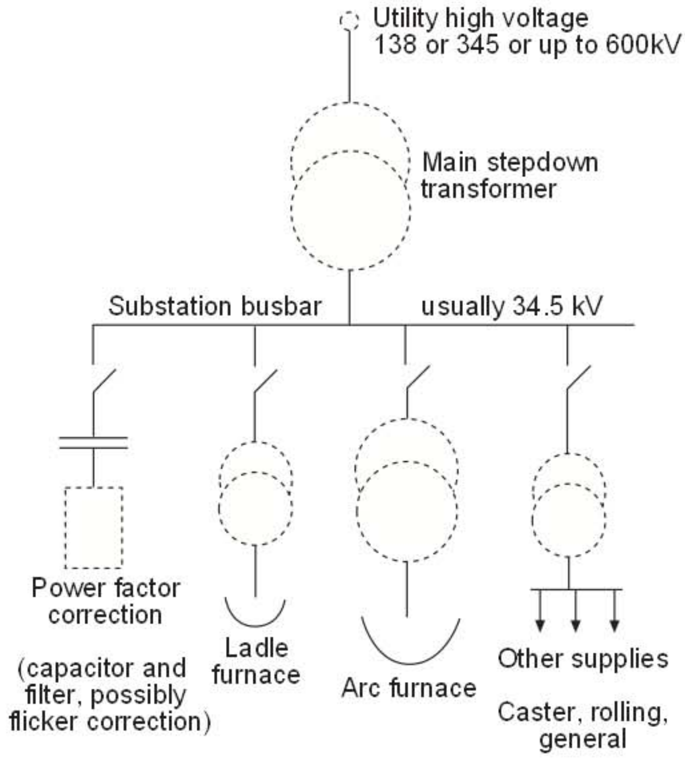


Fig. 10.26 Power transformation from high voltage line to the arc furnace. *(Courtesy of UCAR.)*

10.2.1.2 Transformer Basics

The basic principle of operation of a transformer is illustrated in the sketch of Fig. 10.27. A primary winding is wrapped around one leg of a closed, silicon steel iron core. Around another leg there is the secondary coil of a lower number of turns. The energy flows through the core transported by the magnetic field which is common to both windings. Because of the common field the product of amps × turns is the same for each coil. Thus if there are, for example, 100 turns on the primary coil and ten on the secondary the secondary current will be ten times the primary current.

At the same time the voltage induced in each coil is determined by the number of turns and the magnetic field. Again the magnetic field is common so the voltage ratio of the primary coil to the secondary coil is also determined by the ratio of primary turns to secondary turns.

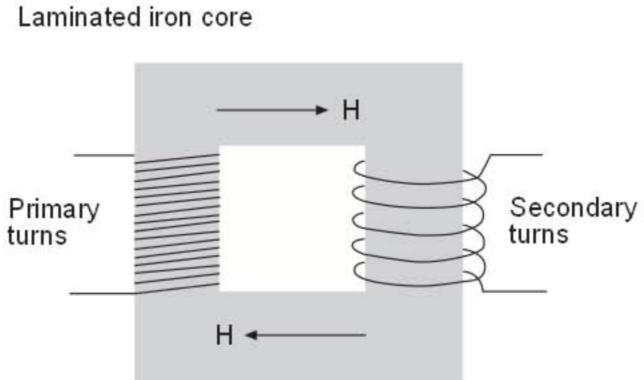


Fig. 10.27 Relationships for a basic transformer. *(Courtesy of UCAR.)*

Mathematically we have:

$$\frac{\text{Primary voltage}}{\text{Secondary voltage}} = \frac{\text{Primary turns}}{\text{Secondary turns}} = \frac{\text{Secondary Amps}}{\text{Primary Amps}} \quad (10.2.1)$$

The arrangement of a three-phase transformer is sketched in Fig. 10.28. The closed iron core consists of three vertical legs, one for each phase. Since it is not necessary to have primary and secondary windings on different legs they are usually wrapped over the same leg. Because the secondary carries a high current it is practical to place it outside of the primary.

10.2.1.3 Core or Shell Designs

As sketched the windings are shown as cylinders enclosing the iron core—the core-type transformer design. An alternative used in some transformers is to wind the coils as spiral pancakes which are arranged like a stack of disks over each leg. This design is known as a shell-type transformer and was chosen by some furnace manufacturers/steelmakers for its lower reactance (see Section 10.2.4 for discussion of reactance). However in latter years the search for low reactance has been replaced by the high reactance design so the use of shell-type transformers is diminishing.

10.2.1.4 Tap Changer

The purpose of a tap changer is to allow a choice of different combinations of volts and amps for different stages of a heat. This is achieved by changing the number of turns of primary coil. (The primary takes lower current so it is simpler to change the number of turns on this coil rather than the high current secondary coil). Basically it takes the form of a motorized box of contacts which switch the primary current to different parts of the coil around the iron core. A schematic diagram of the connection to the primary coil is given in Fig. 10.29.

Most tap changers are designed to operate on-load. This means switching primary current, usually in the 2 kA range, at 34.5 kV. A make-before-break contact movement is used to avoid current interruption. Accordingly these contacts are subject to heavy erosion due to arcing and therefore require preventative maintenance.

Some steelmakers choose an off-load tap changer in order to avoid the heavy duty of on-load switching. However, such a tap changer requires that the steelmaker break the arc by lifting

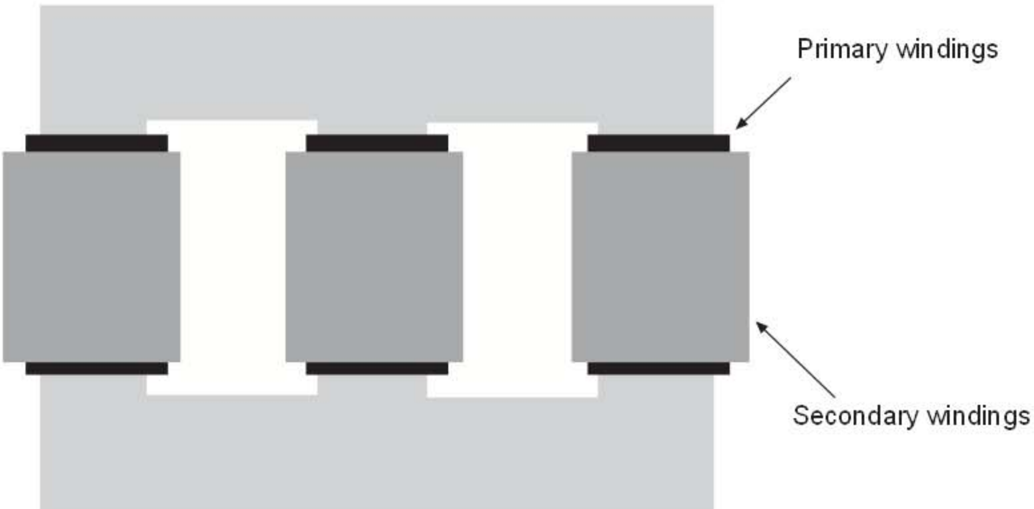


Fig. 10.28 Arrangement of a three-phase transformer. *(Courtesy of UCAR.)*

electrodes and this procedure may take as long as one minute. Today such a delay each time a tap is changed is intolerable and as such this design is becoming rare.

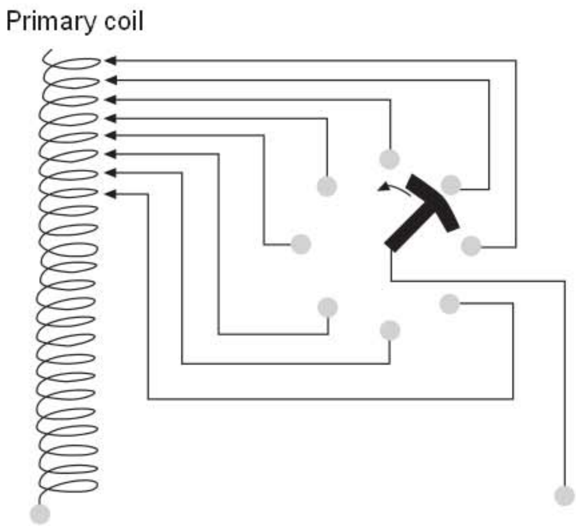


Fig. 10.29 Schematic arrangement of a tap changer. *(Courtesy of UCAR.)*

10.2.1.5 Transformer Generalities

The maximum magnetic field strength within the core is determined by the properties of the silicon steel laminations from which the core is built. This field maximum therefore dictates the size of the core which typically weighs several tons. Several more tons are contributed by the copper windings of which the secondary may be capable of 60 kA operation. Cooling of these coils, heated by Ohmic losses, is achieved by forced oil circulation, the oil itself being cooled by water in a heat exchanger usually outside the transformer tank. Together with its oil, a typical furnace transformer may weigh in the 50–100 ton range.

Transformers are rated in terms of MVA which is a measure of the Ohmic heating of the coils and therefore the temperatures reached by the windings. Persistent overloading of the transformer by the steelmaker will lead to higher than design coil temperatures and a corresponding decrease in lifetime of the insulating material separating coils from each other and the core.

The duty of a furnace transformer is arduous. Arcs are unsteady loads and are often short-circuited and sometimes open-circuited by scrap movements, leading to large current fluctuations. The secondary coils are subject to severe magnetic force shocks which must be absorbed by the insulated clamping system holding the coils in place. Any loosening of these clamps could result in failure of the insulation.

The primary coil sees a variety of voltage spikes due to switching the transformer on or off, which may happen up to 100 times per day. Although the coils are rated at approximately 34.5 kV, between them high frequency voltage surges can reach several times this level.

Because of the critical role of the furnace transformer in steel production there are a variety of alarms and indicators intended to allow monitoring and preventative maintenance.

Tap changers tend to be attached outside of the transformer tank to allow for easy access for contact maintenance.

10.2.2 Furnace Secondary System

The secondary coil terminations from the furnace transformer exit the tank as a set of copper busbars. To step up to the high currents required in the furnace these exit busbars are then connected into a delta closure usually affected outside the transformer, see Fig. 10.30. (Some suppliers have put this delta closure inside the tank to improve cooling.)

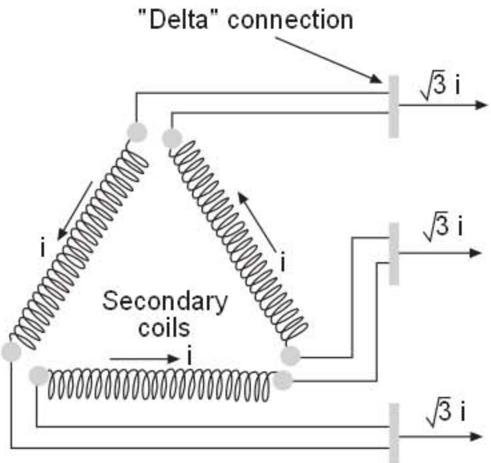


Fig. 10.30 Secondary coil terminations are connected in delta, usually outside the tank. *(Courtesy of UCAR.)*

As there is always a slight risk that the transformer oil may be involved with a fire hazard it is standard practice to place the transformer and some other ancillary equipment in its own building, the transformer house, which simplifies fire control measures associated with the transformer. Thus the high current busbars powering the furnace pass through a wall of this house and it is at this junction where the flexible cables are connected.

10.2.2.1 Cables

Cables are necessary to allow the electrodes to follow the scrap level lowering during meltdown and to allow for furnace movements such as roof swing and tapping. Usually several cables are required on each phase to handle the current.

10.2.2.2 Furnace Arms and Busbars

The mobile ends of the cables are attached to the horizontal arm/busbar system. More and more the manufacturers of furnaces pass the current through an electrically conductive arm to the electrodes rather than through separate busbars. Such conductive arms are either copper clad, steel box constructions or aluminum alloy fabrications. In both cases the whole arm runs at voltage and must therefore be insulated from its supports. In the older separate busbar arrangement both the busbars and the arms were individually isolated electrically from one another.

Current passes to the electrode via the holder. The most common design here is to have one or two fixed contact surfaces, which also act to align the electrode, and a separate pressure pad. The latter is pressed against the electrode in fail-safe fashion with a spring. This spring is compressed hydraulically to open the holder.

10.2.3 Regulation

10.2.3.1 Hydraulic Components

The electrode/arm/mast/cable assembly weighs typically in the range of 20 tons. This is moved vertically for control purposes by a hydraulic cylinder incorporated in the mast. (In some furnaces the movement is effected by an electric motor/cable winch arrangement). As the arc length is dependent, amongst other things, on the level of scrap or liquid under the electrode, and this level changes through the heat, it is necessary to have an automatic control over electrode position—the regulation system.

The flow of the inflammable fluid to the mast cylinder is under the control of a hydraulic spool valve. Flow control is achieved by covering and uncovering ports by displacement of the spool over a stroke in the range of 10 mm. It is pushed by an hydraulic amplifier, a device which generates the necessary power from a separate valve operating with lubricating oil under pressure. An electrical signal enters this amplifying valve at the level of milliamps or a few volts. Thus the system consists of a low power electrical signal, amplified by a hydraulic valve causing displacement of the main spool valve. Fig. 10.31 shows a section of a typical combined amplifier/spool valve.

10.2.3.2 Electronic Control of the Hydraulic Valve

Modern valves incorporate electronic control of several parameters. The most basic parameter is the proportionality between signal level and electrode speed—the gain. Spool valves usually exhibit a dead band over which a signal change produces no flow and to some extent this can be compensated in the electronic system. It is also common practice to control maximum speed electronically rather than through separate flow impedances in the hydraulic lines.

Further signals which can be used by the electronic part of the regulation can be provided from hydraulic pressure and flow rate, allowing speed feedback to be employed.

10.2.3.3 Electrical Control Philosophies

There are significant variations in the way the furnace electrical signals are manipulated in order to achieve the control of arc furnaces. These variations reflect the different philosophies and opinions

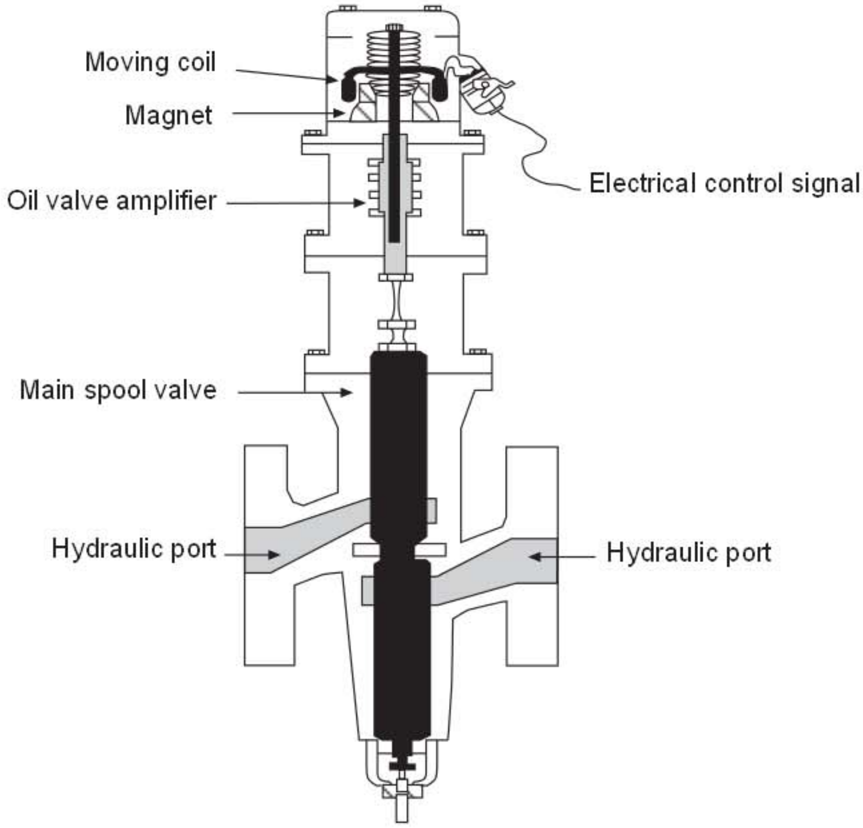


Fig. 10.31 Hydraulic regulator. *(Courtesy of UCAR.)*

amongst suppliers and steelmakers on how best to achieve control. It is the regulation system which influences many important aspects of furnace performance, such as MW input, mean current, arc stability, scrap melting pattern, energy losses to water-cooled panels, energy, and electrode and refractory consumptions. As all these parameters are interrelated in a complex and not well understood manner, it is not surprising that there are many differences of opinion on optimum control strategies.

There is a limit to the acceleration to which the cantilevered mast/arm/electrode can be moved; too high can lead to excessive forces on the system. If, however, the acceleration is too low the time taken to move the electrode becomes excessive. As a general rule the response time (time to reach 90% of full speed) falls between 0.2 and 0.5 seconds. In contrast, arc voltage and current changes are very fast; times are less than one cycle (1/60 or 1/50th second). Because of this inherent mechanical limitation it is not possible to move the system fast enough to correct the fast arc changes; only the lower speed electrical variations can be controlled by the electrode regulation system.

The accepted standard handling of the electrical signals is to form an impedance control. A voltage signal taken from the phase to ground and a current signal are each separately rectified and their DC values are compared back-to-back, as shown in Fig. 10.32. If the voltage and current are each at a desired level—the set point chosen by the

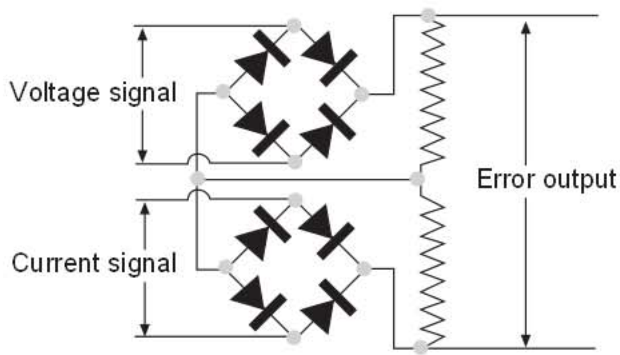


Fig. 10.32 Method of forming a control signal. *(Courtesy of UCAR.)*

steelmaker—the output from this comparison of signals is arranged to be zero. If however the current exceeds this level its signal increases and simultaneously the voltage decreases. Then the two back-to-back voltages do not balance and an output voltage is generated. This signal goes to the regulating valve in such a way to command the electrode to raise, aiming to reduce current. This method of control attempts to hold the ratio of voltage to current constant, hence its description as impedance control.

Alternatives to control of impedance are control over current, arc resistance, arc power or arc voltage. In each case the system attempts to keep the relevant parameter constant. In the cases of arc resistance, arc power and arc voltage controls it is necessary for the supplier to obtain an arc voltage signal. This is not straightforward but the techniques for doing so have been known for some time.[8] The extraction of an arc voltage signal requires that inductive voltages proportional to the rate of change of current be subtracted from the easily accessible secondary phase-to-ground voltages. Rogowski coils are used for this purpose. Indeed such coils, with integration of the output, are commonly used for current measurement.

During the course of a heat the electrical characteristics of the furnace vary. As a consequence the mean electrical parameters will drift with time through the heat if the set points are constant. And the different control philosophies—impedance, current, resistance, etc.—drift through the heat in different ways. Another strategy open to the steelmaker therefore is to vary the set points through the heat in such a way as to achieve real-time optimization.

10.2.4 Electrical Considerations for AC Furnaces

10.2.4.1 Importance of Reactance

In high current systems like arc furnaces reactance plays an important role in the electrical characteristics.

Fig. 10.33 illustrates a high current loop consisting of two bus tubes 15 m (50 ft) long, separated by about 1 m (3 ft). These dimensions are chosen to represent two phases of a three-phase furnace secondary from transformer to arc. If the current is DC the voltage necessary to drive a current of, say, 60 kA around the loop is merely that required to overcome Ohmic resistance. For typical furnace busbars this voltage would only be about 2.5 volts. But if we wish to vary the current we must also overcome the induced voltage generated by the change in magnetic field linking the loop. High currents mean high fields so the induced voltage is also high and is always opposed to the source of current variation. In this example if we wish to drive 60 kA AC current around this loop we would need a much higher voltage—about 300 volts! This inductive impedance to alternating current flow is called reactance.

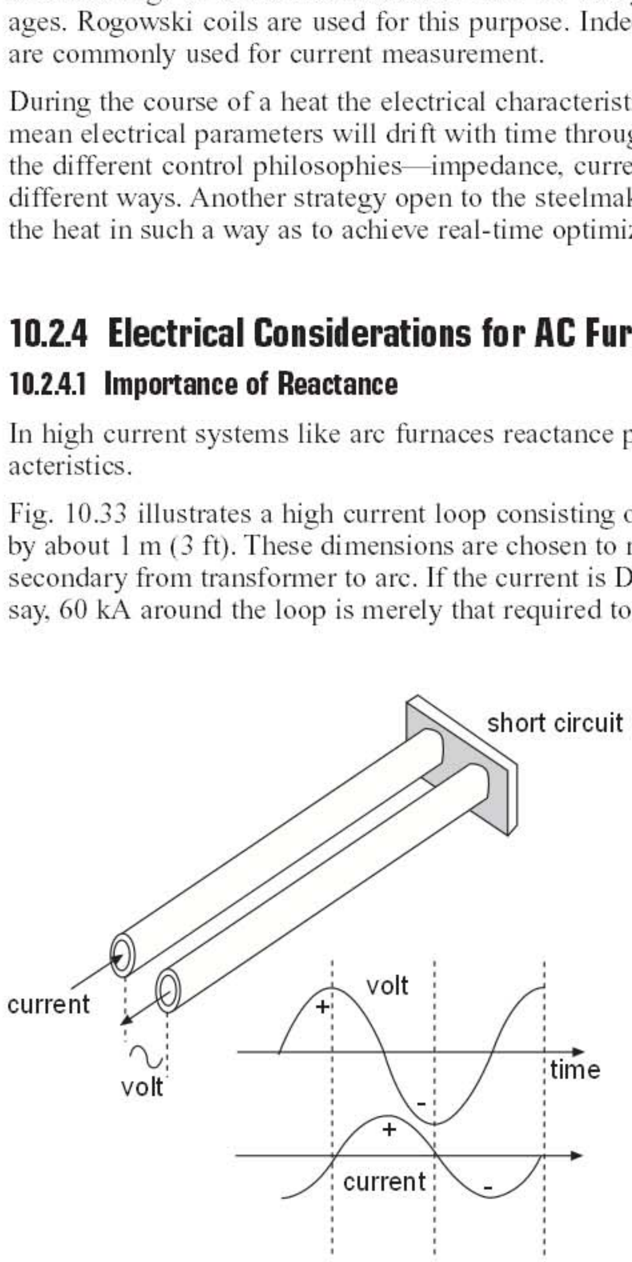


Fig. 10.33 Representation of two phases of an arc furnace secondary current circuit. *(Courtesy of UCAR.)*

For a sinusoidal current waveform the voltage waveform is also sinusoidal but one quarter cycle ahead in time (90 electrical degrees), as shown in Fig. 10.33. This means that the voltage maximum occurs as the current passes through zero. This feature of high voltage around current zero is critical to arc stability when an arc is present.

10.2.4.2 Arc Characteristics

The power flow from the utility's generator, through the net and set of transformers, through cables, arms and electrodes finally reaches the arcs where the useful work of melting and heating occurs. Approximately 95% of the power taken from the utility is released in the arc so its characteristics dominate the functioning of the whole system.

As an electrical element the arc is complicated; it is not a simple fixed Ohmic resistor. Current is transported by free electrons and positive ions in a high temperature plasma in which temperatures may be of the order of 10,000–15,000°C in the hottest regions, dropping to local ambient of, say, 2000°C at the boundary. The diameter of this conducting column depends on current and is of the order of 10–20 cm (4–8 in.), much less than that of the graphite electrode. The voltage depends principally on the length of the column and is of the order of 10 V/cm (25 V/in.); a 300 V arc may therefore typically be 30 cm (12 in.) long.

Because of its high temperature, density is low and for this reason the arc is very easily displaced at high speed by magnetic fields, especially local fields produced by the current flow in pieces of scrap. These effects can result in arc movement over the end of the graphite electrode in milliseconds (ms). In doing so the length of the arc generally changes for geometrical reasons and this in turn results in rapid voltage variations.

Another consequence of the low mass of an arc is that its thermal capacity is also very low. If the current is switched off the ionized plasma column disappears with a characteristic time of about one millisecond. Thus when the current is alternating, as in an AC furnace, the arc has a tendency to cool off each time the current drops to zero. Re-establishment of current flow in the reverse direction may take several hundred volts, depending on ambient conditions, mainly how hot are the graphite and steel arc terminations. This re-establishment of the arc as the current passes through zero is assisted greatly by the presence of reactance in the circuit. The more reactance present the greater is the available voltage at current zero and the smoother is the current polarity transition.

Fig. 10.34 shows arc voltage waveforms under various conditions during a heat. At the beginning when the arc is running between the comparatively cool graphite electrode and cold pieces of scrap, instability of arc voltage is at a maximum. During the course of melting the arc voltage waveform becomes smoother as temperatures rise. A further improvement occurs when the steel termination of the arc is liquid. At the extreme, when the arcs are submerged in foaming slag, the voltage waveforms are almost sinusoidal.

This large excursion in the stability and harmonic content of the arc voltage is reflected in the electrical characteristics of an arc furnace. In the early periods of a heat, effective MW input is lower for a given regulated set point than at later periods. Fig. 10.35 shows generalized electrical characteristics both as MW versus kA and as MW versus MVAR. As stability improves all the curves move to a higher level of MW, tending towards the limit case where the arc voltages would be perfectly sinusoidal and steady. In this hypothetical case of perfectly Ohmic arcs the MW–MVAR characteristic becomes the familiar semicircle, the magnitude of which depends on the transformer voltage and the circuit reactance. Mathematically the effect of real arcs is similar to introducing more reactance into the circuit than physically exists. One can model empirically the changing characteristics by

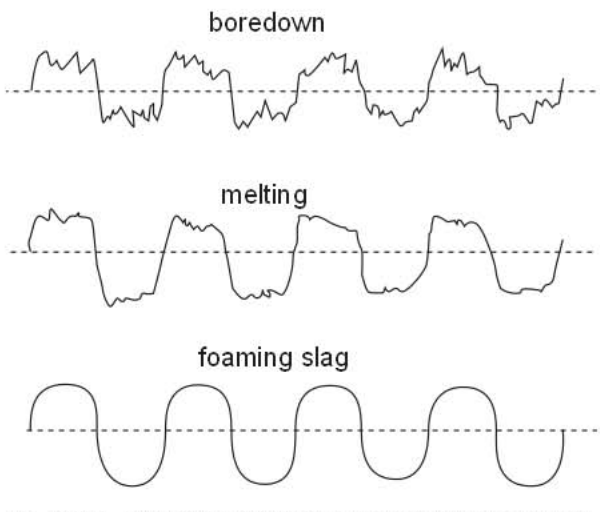


Fig. 10.34 Arc voltage waveforms at three stages of a heat. *(Courtesy of UCAR.)*

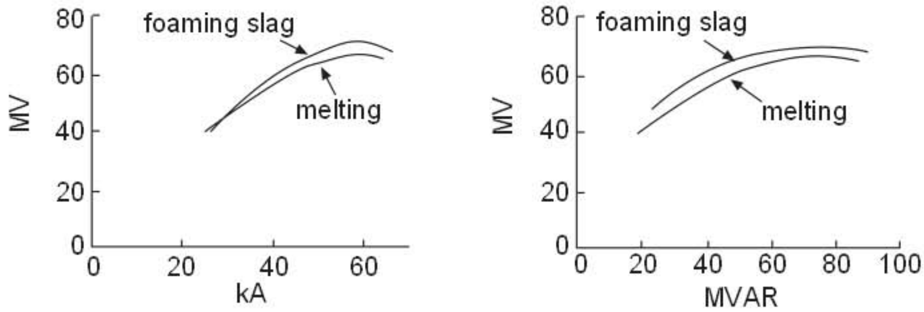


Fig. 10.35 Example of electrical characteristics for an AC furnace. *(Courtesy of UCAR.)*

considering the total circuit to have an operating reactance which is higher than the actual and which decreases with time through the heat.

10.2.4.3 Supplementary Reactance; the High Reactance AC Furnace

The geometrical constraints of the layout of arc furnaces in the 80–160 ton range result in systems which tend to have the total reactance—transformer plus secondary system—falling in a limited range of 2.5–4 milliOhms (per phase). In order to operate with arc voltages in the 400–500 V range this natural reactance level is too low to guarantee smooth transition through current zero during scrap melting. It is now therefore common practice to add supplementary reactance to the circuit. This is achieved conveniently by adding a reactor on the line supplying the primary of the furnace transformer where currents are lower.

Such reactors come in two varieties; one is the air-cored coil design, generally placed outdoors of the meltshop. This design is simple and inexpensive. The alternative is to use an iron-cored reactor, in which the magnetic field is restrained to the restricted volume of the core. Consequently this design takes up less space and is suitable for installation next to the furnace transformer, or, in some cases, within the transformer tank.

The usual magnitude of the additional reactance is such as to push the total reactance per phase into the 5–10 mOhm range (as viewed from the secondary voltage of the transformer). Since the total system impedance is higher in this design of furnace it is necessary to have a corresponding higher transformer voltage to obtain the desired characteristics.

The effect of operating with higher reactance and higher secondary voltage is illustrated in Fig. 10.36. With the increased reactance the secondary voltage waveform is much higher during the

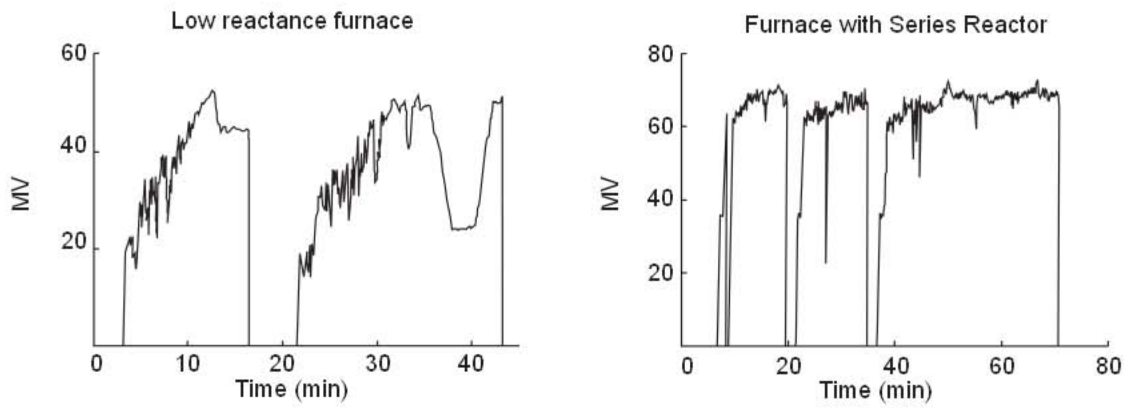


Fig. 10.36 Improvement of stability with additional reactance.

passage through current zero. As a consequence the risk of current interruption is greatly reduced; current waveforms are smoother and MW are therefore increased.

10.2.5 Electrical Considerations for DC Furnaces

10.2.5.1 DC Supply

The required high power again is supplied from a high voltage three-phase AC network. This is converted to DC by rectification of the output of the furnace transformer.

Rectification is achieved by bridge-connected thyristors. A single, three-phase bridge connection is illustrated in Fig. 10.37. The voltage waveforms for different thyristors firing angles are shown in Fig. 10.38 (simplified). This output is described as six-pulse. Normally 12, 18 or 24-pulse supplies are used in arc furnaces, obtained by multiple, parallel transformers electrically displaced one from another so that their individual pulses overlap uniformly. This electrical displacement, of 15, 10 or 7.5°, corresponding to the 12, 18 or 24-pulse systems, is made by various coil connections within the transformer. For this reason the transformers used for DC furnaces are quite different to those for AC and are generally unsuitable for AC furnace operation.

The volt/amp characteristic of a DC supply consists of a weakly declining drop of DC voltage as the DC current increases. The slope of this line is on the order of 1 volt per kA and is determined by the commutating reactance of the transformer/rectifier combination, not by the arc furnace. In order therefore to limit wide current excursions due to widely different arc voltages thyristors are used in preference to diodes. The conducting instant after current zero (firing angle delay) is under the control of the gate terminal. Each thyristor can, in principle, be turned off within half a cycle.

Even so, within the several millisecond delay between an arc voltage change (e.g. a short circuit) and the control of the thyristors, currents could increase significantly. To reduce the rate of rise of the current it is normal to add a reactor within the DC current loop, the natural reactance of the high current DC loop being inadequate.

These reactors are sized to have an inductance in the 100–400 microH range. Since they take the full DC current, Ohmic losses are significant and can only be maintained within acceptable bounds by employing an adequate section of the copper or aluminum making up the coils.

Thyristors are each capable of handling a few kA and a few kV of reverse polarity. An arrangement of series and parallel connected thyristors makes up each leg. Fuses and voltage balancing resistors are used as protective measures. Cooling is affected by de-ionized water.

10.2.5.2 Electrical Characteristics of DC Furnaces

The thyristor control is normally chosen to hold current constant. Thus the AC current before the rectifier is also constant, as is the primary current. Considering powers on the AC primary we see that constant current means MVA is constant. The characteristic of MW as a function of MVAR is therefore a quadrant of a circle for which

$$MW^2 + MVAR^2 = MVA^2 = \text{constant} \qquad (10.2.2)$$

The corresponding DC voltage/DC current relationship is illustrated in Fig. 10.39 where an example of a 1000 V maximum is shown. Generally the slope of the volt/amp line is linear and drops typically 100 V in 100 kA.

Thus at 100 kA, for example, the thyristor control can hold constant current over an arc voltage range from about 900 V down to short circuit by varying the firing angle.

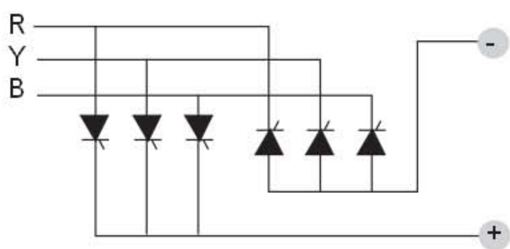


Fig. 10.37 Six-pulse bridge rectifier using thyristors. *(Courtesy of UCAR.)*

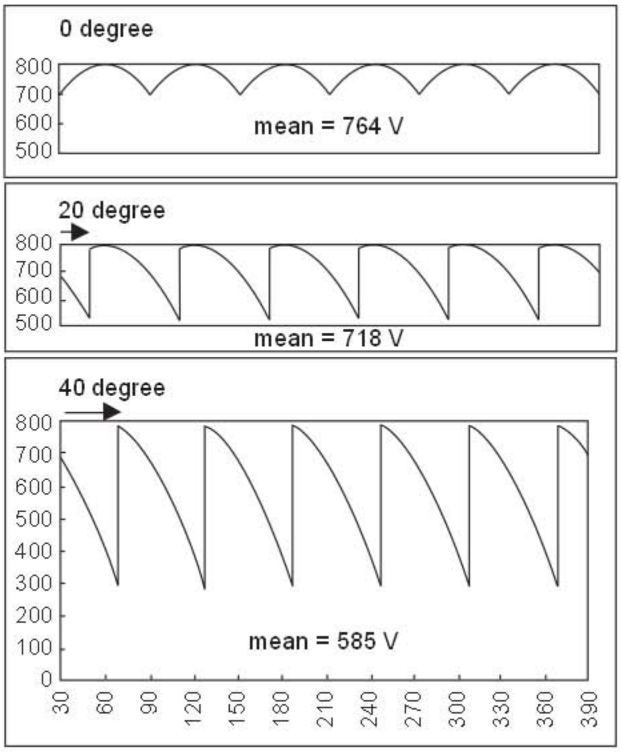


Fig. 10.38 Three-phase bridge waveforms (simplified); effect of firing angle on waveform and mean voltage. *(Courtesy of UCAR.)*

10.2.5.3 Bottom Connections

In order to operate with a single DC arc it is necessary to make an electrical connection—the positive anode—to the steel charge. Various furnace manufacturers have developed several solutions to this problem. The ideas are illustrated in Fig. 10.40.

In one type the anode current is shared amongst many steel rods embedded in a rammed refractory block. The rods, with a diameter of about one inch, could be one meter long and are linked by a copper plate below the furnace shell. The whole anode block may measure 1–2 m in diameter.

A variation on the pin type is to use thin steel sheets, again embedded in refractory.

Another variation is to employ a steel billet of diameter 400 mm (8 in.) passing through an insulated sleeve, leading to a cooled copper connection below the furnace shell.

In all three of these designs (pin, sheet or billet) the top of the steel conductor melts through the course of the heat. It resolidifies during power-off and after scrap charging.

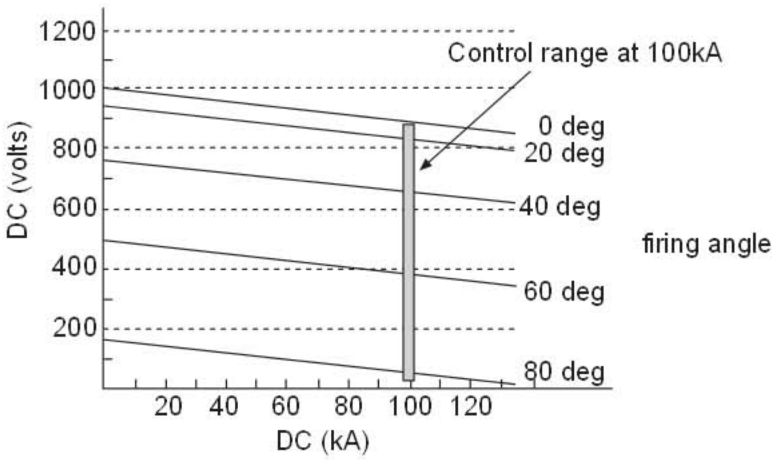


Fig. 10.39 Typical DC volt/amp characteristics; 1000 V example. *(Courtesy of UCAR.)*

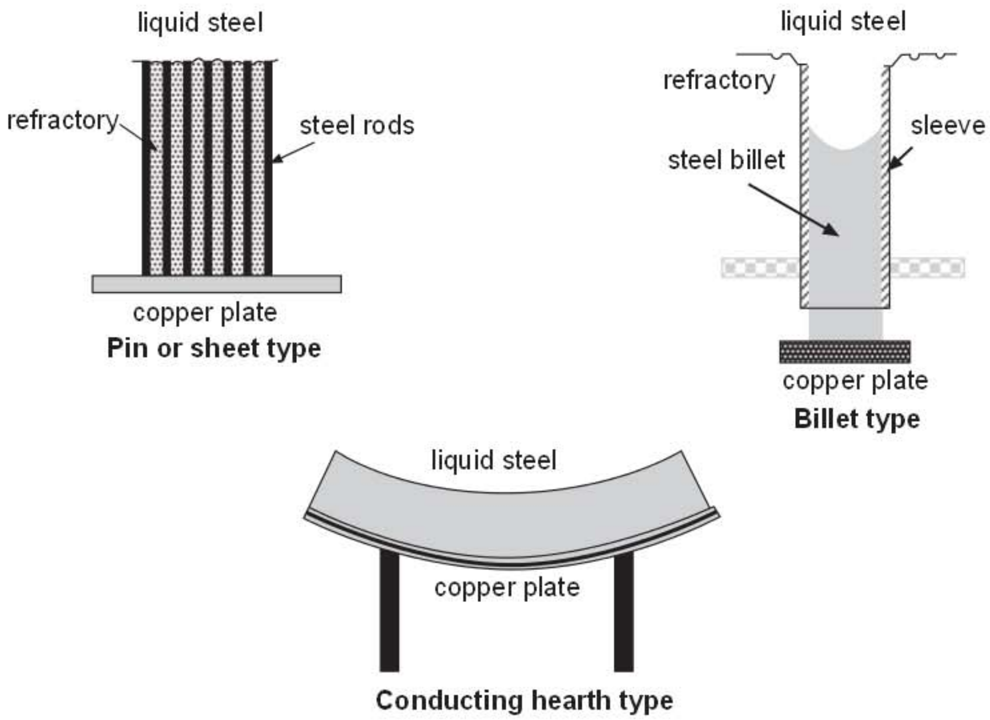


Fig. 10.40 Sketches of types of bottom connection for DC furnaces. *(Courtesy of UCAR.)*

An alternative to the steel-to-steel current designs is one where the current is taken through conductive refractories to a large diameter, copper bottom plate.

In all bottom connection types there must be insulation between the anode connection and the furnace shell. This is to reduce the likelihood of current passing through the shell directly to the anode busbars.

10.3 Graphite Electrodes

Graphite electrodes play an important part in electric arc furnace operation, allowing for the transfer of electrical energy from the power supply to the furnace bath. Electrodes must be capable of withstanding large temperature swings during furnace operation while at the same time providing for continuous and uniform power supply to the process.

As electric furnaces have evolved over the past 20 years, advances in graphite electrode technology have played an important part. As furnace designs were improved to allow higher power input rates, graphite electrode properties were improved to allow greater supply of electrical power to the furnace. Thus as EAF designs and operating practices have evolved, graphite electrode technology has kept pace, allowing for continued advances in EAF steelmaking capability. It is now common for new arc furnace installations to have transformers rated at in excess of 100 MVA. DC furnace operations have also placed additional requirements on electrode characteristics.

10.3.1 Electrode Manufacture

Graphite electrodes are typically made using premium petroleum needle coke, coal tar pitch, and selected proprietary additives. Electrodes are formed in the shape of cylindrical sections that are fit together using screw-in connecting pin sections. Electrodes are available in a variety of diameters, the requirements being dependent on the required current carrying capacity. Typically, diameters greater than 24 inches carry a premium price due to the requirement for different process technologies. Higher quality raw materials, increased energy amounts, extended processing schedules, and the installation of larger, heavy duty equipment are necessary to produce these electrodes.

Processing and manufacture of graphite electrodes consists of the following steps as shown in Fig.10.41:

1. Milling and mixing of petroleum needle coke with coal tar pitch and selected additives.
2. The mixture is then extruded and cut to cylindrical, green electrode sections.
3. The green electrodes are placed in saggers (stainless steel cans) in which the electrodes are covered with a protective packing medium like sand. The filled saggers are loaded onto railcar flatbeds which are moved into large gas fired car bottom kilns where the green electrodes are baked to approximately 800°C. The bituminous, green electrode material is transformed into amorphous, brittle carbon which is abrasive and difficult to machine. This process requires careful control to ensure

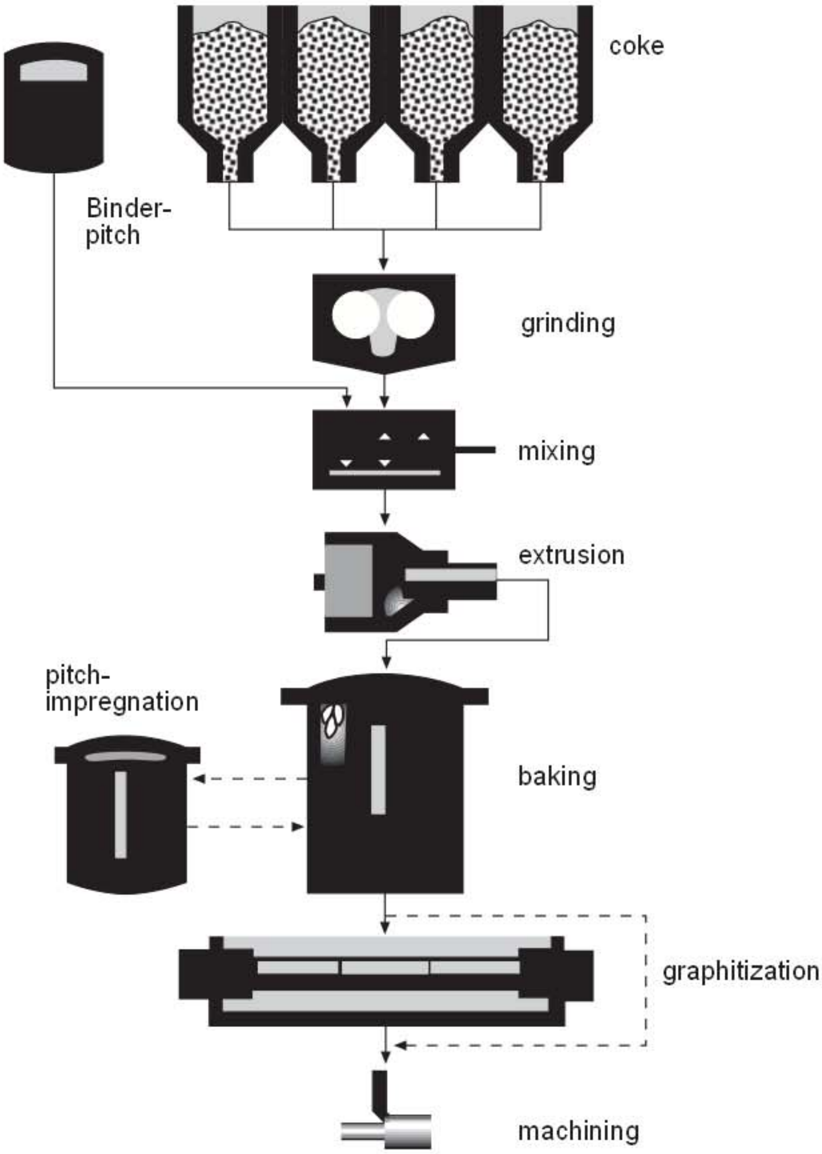


Fig. 10.41 Schematic of graphite production sequence. *(Courtesy of SGL Carbon Corp.)*

that thermal gradients remain small and rapid gas buildup does not occur. Thermally induced stresses or rapid gas buildup can result in cracking, distortion, or excessive porosity which can not be tolerated in the finished product. For this reason, bake cycles are long and take between three to four weeks.

4. The baked carbon sections are impregnated with petroleum pitch in order to increase strength and density. This also improves the end product electrical conductivity.

5. The impregnated carbon sections are again loaded into car bottom kilns and re-baked so that the petroleum pitch is converted to carbon.

6. The re-baked carbon sections are assembled to columns of eight to ten electrodes and loaded into large, electrically powered graphitizing furnaces. Direct current of more than 100kA is passed through the electrode columns heating them to approximately 3000°C. The intense heating causes the crystalline structure to change from the random amorphous form to the ordered layer structure of graphite. This modification increases machinability of the material as well as greatly improving electrical, thermal and mechanical properties. The graphitizing process is very energy intensive and requires more than 3000 kWhr per ton of graphite.

7. Finally, the graphitized sections are machined to the required diameter and length on large lathes. Tapered sockets are machined into each end to accommodate screw-in connecting pins which are used to attach the electrode sections end-to-end. The total production process from extrusion to shipping is quite time consuming and takes approximately three months.

10.3.2 Electrode Properties

Electrode physical properties vary somewhat from manufacturer to manufacturer. However, for most ultra high power (UHP) furnace operations these properties will be similar regardless of manufacturer. Following in Table 10.1 are typical electrode physical properties:

Table 10.1 Typical Electrode Physical Properties.

		Electrode Diameter (in.)		
Property	**Units**	**18–20**	**22–24**	**26–28**
Bulk Density	g/cm^3	1.69–1.76	1.68–1.76	1.68–1.76
Real Density	g/cm^3	2.22–2.25	2.22–2.25	2.22–2.25
Porosity	%	21–25	21–25	21–25
Specific Resistivity	10^{-5} Ohm in.	17.0–21.5	17.0–21.5	17.0–21.5
Bending Strength min.	psi	1200	1150	1000
Bending Strength max.	psi	1500	1450	1400
Young's Modulus	psi x 10^6	1.1–1.5	1.0–1.4	1.0–1.3
Coefficient of Thermal Expansion	10^{-6}/°C	1.25–1.65	1.25–1.65	1.25–1.65
Ash Content	%	0.1–0.6	0.1–0.6	0.1–0.6

10.3.3 Electrode Wear Mechanisms

Considerable research has been carried out over the years to evaluate the factors affecting electrode consumption. Generally speaking, electrode consumption occurs as both continuous consumption and discontinuous consumption. Continuous consumption usually accounts for 90% or greater of the total consumption and results from tip consumption (sublimation) and sidewall oxidation. Discontinuous consumption mechanisms include electrode breakage, butt losses, and electrode tip spalling.

Prior to implementation of electrode water spray cooling, the two continuous consumption mechanisms were believed to be approximately equal. Water spray cooling results in reduced electrode sidewall temperatures above the furnace roof which helps to reduce the amount of sidewall oxidation. Thus a shift to 50% of consumption due to tip consumption and 40% due to sidewall oxidation has been observed in operations using electrode spray cooling.

10.3.3.1 Original Bowman Correlations

Correlations have been developed empirically to estimate electrode consumption for various operating practices. It was found that electrode tip consumption is proportional to the square of the current and to the time over which the tip is consumed, which is the power-on time. The correlation is as follows:

$$C_{TIP} \quad R_{SUB} \cdot \quad \frac{I^2 \cdot \; t_{PO}}{P} \tag{10.3.1}$$

or emphasizing the importance of productivity in tons per hour:

$$C_{TIP} \quad R_{SUB} \cdot \quad \frac{I^2 \cdot \; TU}{p} \tag{10.3.2}$$

where

C_{TIP} = graphite tip consumption (lbs/ton)
R_{SUB} = sublimation rate (lbs/kA^2 per hr)
t_{PO} = power-on time (hrs)
I = current per phase (kA)
P = furnace productivity (tons/heat)
TU = time utilization = t_{PO}/t_{TAP}
p = productivity (tons/hr)

For electrode sidewall consumption, it was found that it is proportional to the oxidizing electrode surface area and to the relevant time period which is the tap-to-tap time:

$$C_{SIDE} \quad R_{OX} \cdot \quad \frac{A_{OX} \cdot \; t_{TAP}}{P} \tag{10.3.3}$$

where

C_{SIDE} = graphite sidewall consumption (lbs/ton)
R_{OX} = oxidation rate (lbs/ft^2 per hr)
A_{OX} = oxidizing electrode surface area (square feet)
t_{TAP} = tap-to-tap time (hrs)
P = furnace productivity (tons/heat)

Below the limit of the water cooling it is sufficiently accurate to approximate the shape of each oxidizing column to a cone, the diameter at the top being the full electrode diameter, D, tapering to the tip diameter, D_t, at the bottom and the mean of these two diameters, D_{AV}. If the length of this cone is L_{OX} then the oxidizing area per column is

$$A_{OX} = \pi D_{AV} L_{OX} \tag{10.3.4}$$

With optimum water cooling the oxidizing length, L_{OX}, is close to the length of the column inside the furnace at flat bath. This increases with furnace size and is typically in the 2–4 m (7–12 ft) range.

Typical average values for oxidation and sublimation rates are:

R_{OX} = average oxidation rate = 8 kg/m^2 per hr (1.64 lbs/ft^2 per hr) for the oxidizing cone of water spray cooled electrodes for operations using around 25 Nm^3 (900 scf) lance oxygen and no post-combustion.

$R_{sub\,AC}$ = average sublimation rate per electrode for AC operation = 0.0135 kg/kA^2 per hr (0.0298 lbs/kA^2 per hr).

$R_{sub\,DC}$ = average sublimation rate per electrode for DC operation = 0.0124 kg/kA^2 per hr (0.0273 lbs/kA^2 per hr).

10.3.3.2 Updated Bowman Correlations

In 1995 Bowman published an update to the above correlations in order to fit modern arc furnace practice.[9] The loss of accuracy of the old model seems to have occurred with the move to longer arcs allowed by the application of water-cooled panels. This suggests that arc length, i.e. arc voltage, is an important parameter which was not included in the old model. Other significant changes in arc furnace practice which influence electrode consumption, particularly oxygen consumption and its method of application needed to be considered in order to establish a new, empirical model for oxidation consumption.

10.3.3.2.1 Tip Consumption The old model for tip consumption, contained current squared and a function for the tip diameter. The link with tip diameter was necessary in order to conserve a relationship with the square of the current, which seemed to be correct on both theoretical and experimental grounds. The relation suggested that with larger electrodes, consumption rates decreased. However, larger electrodes are also associated generally with higher voltages. Thus a link to voltage is used in the new model and the link with electrode diameter has been dropped.

The evidence for a dependence on the square of the current is very strong; the data analysis in the original report showed a very good fit over a very large range of values, from 1–60 kA and almost four orders of magnitude of consumption rate. In order to identify the form of the dependence on arc voltage, the strong effect of the current can be eliminated by dividing measured tip consumption rates by kA^2 and plotting the resulting data as a function of arc voltage. The data for the old model were grouped mostly below the 200 arc voltage limit. Newer data covers higher arc voltages approaching 400 volts. Below an arc voltage of about 250 V there is an increase in the specific consumption rate; above 250 V the rate is more or less constant.

Long periods of arcing onto a slag covered bath are a common feature of operations with more than 70% continuously charged DRI or continuously charged processes such as Consteel. Measurements show that such operations consistently yield figures in the 0.010 kg/kA^2 (0.0221 lbs/kA^2) per hour range. Some furnaces running with 100% scrap also achieve these low figures; others, however, reach up to 0.016 kg/kA^2 (0.0353 lbs/kA^2) per hour. The difference between these two groups of furnaces seems to show up in the angle of the electrode tip. Operations with the higher levels of specific consumption rate are characterized by steep tip angles, up to 45° on meltdown; the lower levels seem to go along with tip angles in the 25–30° range. Bowman offers the following hypotheses to answer the questions why the tip consumption rate depends on tip angle and why the tip angles vary between furnaces.

10.3.3.2.1.1 Dependence of Tip Consumption Rate on Tip Angle For ladle furnaces, increasing tip consumption rate with shorter arcs can be explained by the dissolution of graphite due to splashing by the liquid steel. In the case of melting furnaces, the geometry of the arcing volume around the electrode tip illustrates that for a given average arc voltage the lower part of the tip comes closer to the liquid as the angle increases. Thus, at a tip angle of 40° the gap between graphite and steel is only about two inches at an average arc of 200 volt. In contrast, at 25° it is over three inches at 200 V. At 300 V the corresponding gaps are about five inches and seven inches respectively. The probability of graphite contact with steel is therefore increased with greater tip angles and lower arc voltages.

10.3.3.2.1.2 Variation of Tip Angles between Furnaces Arc blowout, the cause of tip angling, is a magnetic phenomenon. It depends on the proximity of the electrodes, i.e. pitch circle diameter (PCD), the distribution of the ferromagnetic scrap and the current distribution within the charge, between the arcs. It has also been suggested that deep, foaming slag can offer magnetic field protection. For a given current, the parameters which can generate low or high magnetic fields in the arc regions can be summarized in Table 10.2.

Table 10.2 Magnetic Field Generation

Influencing Parameters	High Field Strength	Low Field Strength
PCD	small	large
Bath	shallow	deep
Slag	shallow	deep

Thus a liquid heel operation with maximum time on deep, foaming slag promotes the lower tip angles. Those operations without a hot heel and with shallow slag depths such as stainless steel or acid iron production, are likely to generate steeper tip angles during meltdown.

In long-arc-only operations tip angle tends to stay constant. Tip consumption rates on DC arc furnaces are associated with long arcs and tip angles are usually below 10°. These parameters, as discussed above for AC, should not therefore play a role in consumption.

Based on DC operating data covering the current range 10–130 kA and arc voltages from about 250–550 volts, Bowman found that the specific tip consumption rate is essentially constant at 0.0124 kg/kA^2 (0.0273 lbs/kA^2) per hour, taking into account the experimental errors. No dependence on arc voltage was apparent, as expected from the AC analysis above.

Arc voltage is normally not available on most arc furnaces. Bowman indicates that an accurate value can, however, be calculated from readily available electrical data, in this case MW and kA as follows

$$\text{average arc voltage} = \frac{\text{MW} \cdot 1000}{\sqrt{3} \cdot \text{kA}} - r \cdot \text{kA} \qquad (10.3.5)$$

where the r represents the phase resistance, in milliOhm. Typical values, if not known from a short circuit test, vary with furnace size as shown in Table 10.3.

Table 10.3 Typical Phase Resistance Values

Furnace Electrode Diameter, in.	Phase Resistance, milliOhm
16	0.55
20	0.45
24	0.40

Summarizing, Table 10.4 presents the following specific sublimation rates taking the varying voltage ranges and tip angles into account:

Table 10.4 Sublimation Rates by Furnace Condition

Furnace Condition	R_{SUB}, kg/kA^2 (lbs/kA^2) per hr
AC, arc voltage>250 V	0.0130 (0.0287) × 3 electrodes
AC, tip angle<30°	0.0100 (0.0221) × 3 electrodes
AC, tip angle>45°	0.0160 (0.0353) × 3 electrodes
DC	0.0124 (0.0273)

10.3.3.2.2 Sidewall Oxidation A number of studies have been published which show that there are important differences in the degree to which furnace practice exposes the graphite electrodes to oxidation. These differences are reflected in varying specific rates of oxidation, expressed as kg/m^2 per hr (lbs/ft^2 per hr). Low rates occur with a low oxygen consumption practice and low ingress of air. A continuously charged DRI operation of relatively long duration with a closed furnace and no oxygen lancing is a good example. At the other extreme an operation using large volumes of oxygen, perhaps with burners and post-combustion injection, tends to produce higher specific rates of oxidation. Surface area exposed to oxidation is a major parameter which influences oxidation consumption. Direct water cooling of the electrodes has helped to reduce this area in AC furnaces. In DC furnaces the use of only one electrode further reduces the area significantly.

With such varying oxygen practices in modern operations, it is necessary to assume a varying specific rate of oxidation, kg/m^2 per hr (lbs/ft^2 per hr), based on furnace practice. Oxidation rates are dependent on several features of the use of oxygen (and burners if used) which are often not recorded or not under control: lance angle, lance direction relative to the electrodes, oxygen flow rate and lance diameter, position of the lance tip relative to steel and slag levels, burner excess oxygen levels and duration. In many furnaces the lower parts of the graphite columns are heavily splashed by FeO-rich slag or incompletely reacted oxygen. Post-combustion of CO by injection of oxygen above the slag level can also increase electrode oxidation rates because CO_2 also reacts with graphite.

To account for such varying practices Table 10.5 presents the following range of specific oxidation rates that have been calculated from measurements on various furnaces:

Table 10.5 Specific Oxidation Rates by Practice

Practice	Specific oxidation rate, kg/m^2 (lbs/ft^2) per hr
Ladle furnaces	3–4 (0.62–0.82)
Closed furnace, <5 Nm^3 (<200 scf) per ton of O_2	3–4 (0.62–0.82)
Closed furnace, 5–15 Nm^3 (~200 to ~550 scf) per ton of O_2	5–6 (1.03–1.23)
Closed furnace, 25–45 Nm^3 (~900 to ~1600 scf) per ton of O_2	5–8 (1.03–1.64)
Closed furnace, 25–45 Nm^3 (~900 to ~1600 scf) per ton of O_2 with post-combustion	8–10 (1.64–2.05)

Water cooling of the electrodes above the roof is now standard practice and has helped to reduce wasteful oxidation of electrodes outside the furnace. However, excessive flows of water will cause an increase in energy consumption, the cost of which normally exceeds any gain in graphite savings. Ideally the water should convert to steam within the roof port, providing some protection to the refractory delta insert. The flow rate to achieve this condition depends on escaping flame intensity and current, but typically may require between 380–680 litres (100–180 gal) per hour per one 24 inch electrode. If the flow rate is in excess of the optimum then water is effectively pouring into the furnace, exiting via the extraction system as steam at high temperature.

10.3.4 Current Carrying Capacity

Limits for current carrying capacity have been developed with the objective of minimizing electrode consumption. Generally, exceeding these limits will result in increased consumption via the discontinuous consumption mechanisms, electrode breakage, butt losses, and spalling. Thus the decision to exceed these limits must weigh increased graphite consumption against increased EAF productivity. Current carrying capacities are given in Fig. 10.42.

The current carrying capacity is greater for DC than for AC electrodes. Under AC conditions, the current is forced towards the peripheral region of the electrode. This is known as skin effect. Under DC conditions, the full electrode cross section is available for the current flow. That is why the electrode resistance under DC conditions, R_{DC}, is apparently lower than the resistance, R_{AC}, under AC conditions. Consequently, the current can be increased under DC over that for AC conditions without increasing the resulting Joule's heating in the electrode. Some typical examples are given in Table 10.6.

Besides electrical influences, the current carrying capacity is mainly affected by thermal and mechanical boundary conditions and by the material characteristics of graphite. Again, exceeding these boundary conditions will lead to an increase of the discontinuous consumption processes.

10.3.5 Discontinuous Consumption Processes

Discontinuous electrode consumption has been studied extensively by Lefrank et al. with the objective to improve electrode performance specifically for DC operations.[10] The discontinuous

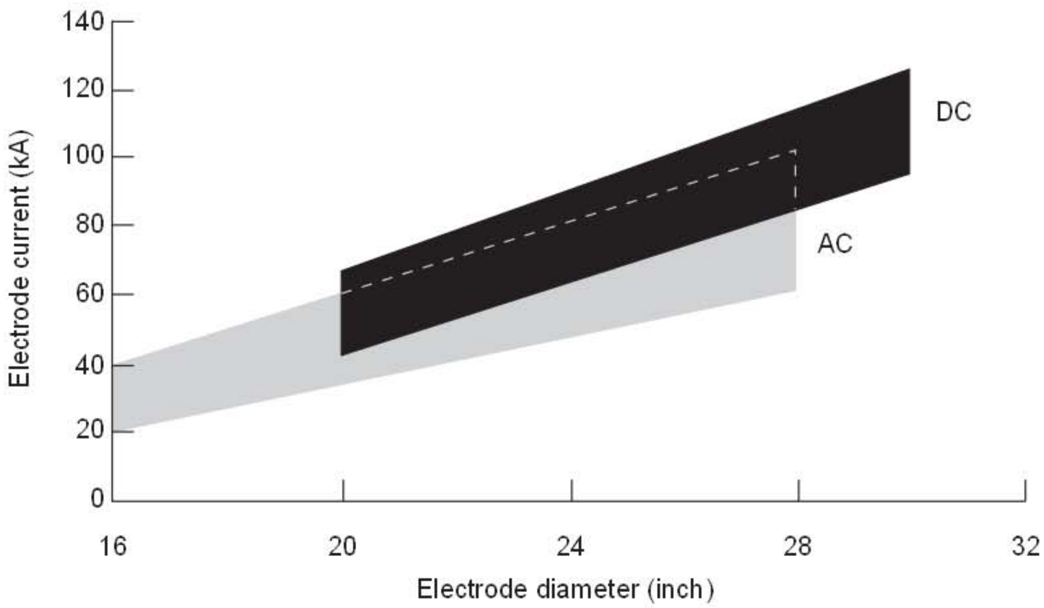


Fig. 10.42 Current carrying capacity vs. electrode diameter for AC and DC furnaces. *(Courtesy of SGL Carbon Corp.)*

Table 10.6 R_{DC} and R_{AC} of 110 in. Electrodes

	Electrode Diameter, (in.)			
	20	24	28	32
R_{DC}, microOhm	70	49	36	27
R_{AC}, microOhm, at 60 Hz	83	65	53	44
I_{DC}/I_{AC}	1.09	1.15	1.21	1.28

consumption processes are results of mechanically or thermally induced stresses which exceed the electrode strength limits. Stress levels of this magnitude occur as electrode tip stresses at the arc foot point or as hoop stresses around the electrode joints. Tip stress leads to tip spalling. Hoop stresses lead to column breakage and butt losses.

10.3.5.1 Butt Losses

Current, temperature, and stress distributions for a 28 inch electrode, operated at 120 kA under AC and DC conditions, are shown schematically in Fig. 10.43.

Under AC conditions, most of the current is flowing through the peripheral region of the electrode. Under DC conditions, more current will flow through the electrode center resulting in an almost evenly distributed, and lower maximum current density than under AC conditions. As a result the DC electrode generates more energy inside the electrode and therefore the temperature gradient is much higher in DC electrodes. Consequently, the hoop stress gradients are much greater for DC conditions. As the electrode diameter and current loads are increased this situation will be amplified further. This phenomenon is a result of the finite graphite thermal conductivity and the reduced capacity of the larger electrode for radiant heat transfer resulting from a reduced surface to volume ratio. Electrode joints are affected even more by the central flow of current in DC operations and potentially, the nipple can overheat resulting in joint opening and socket splitting.

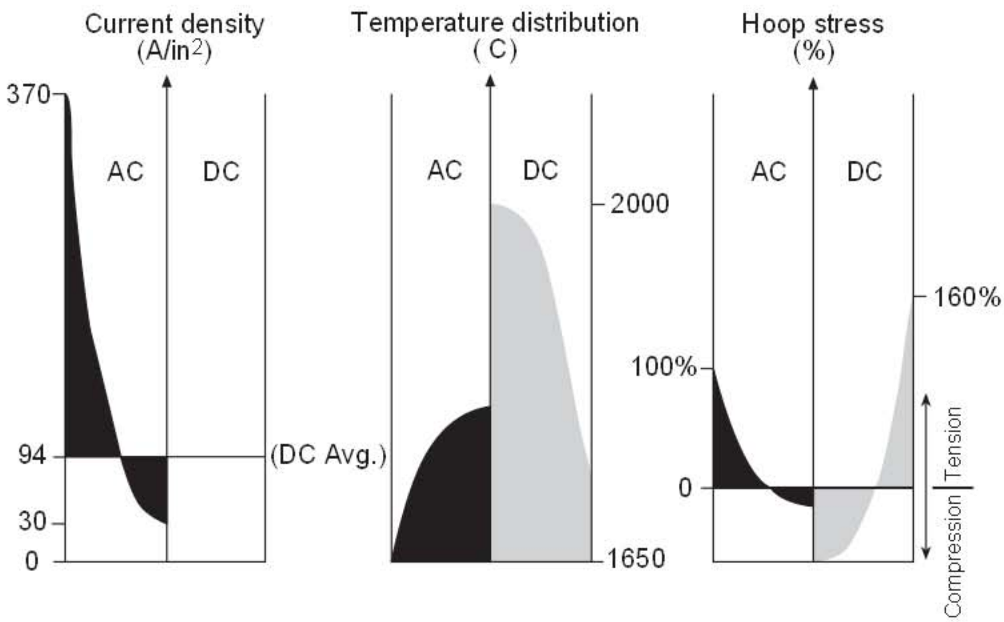


Fig. 10.43 Current, temperature and electrode stress distribution. *(Courtesy of SGL Carbon Corp.)*

10.3.5.2 Tip Spalling

Very high current densities develop at the arc spot on an electrode due to the self-magnetic pinch effect. For DC operations with only one electrode, the tip stresses are higher than for AC operations. The arc spot will move around the tip of the electrode in a random manner but sometimes in DC operation, the arc spot will tend to fix preferentially on one location. This will result in the generation of longitudinal splits and severe butt losses. Improvements in arc deflection control have improved this situation reducing butt losses significantly.

10.3.5.3 Breakage

State of the art AC operations experience few electrode breaks, typically less than two to three per month. However, frequent top joint opening and breakage some time after an electrode addition have been reported by some DC shops. It was hypothesized that a rotational arc movement in some DC operations might generate a clockwise tangential force to the electrode axis leading to the unwinding of the top column joint. For this reason, Lefrank et al. conducted a comprehensive high speed video motion analysis of the arc behavior at major DC furnace operations of different furnace design in the U.S.

The videos show that the DC arc spot is moving with high speed randomly over the electrode tip. The arc spot movement seems to be influenced by the combination of totally random events with controlled, directional forces.

The directional forces are the result of the powerful magnetic fields generated by the DC current loop of the furnace. They can restrict the arc spot to a limited area of the electrode tip and may direct the arc jet preferentially to one furnace side which is shown in Fig. 10.44. Depending on the furnace design, they can also force the arc to rotate either clockwise or counter clockwise for a few hundred milliseconds, but then the arc spot will again move in a totally random and unpredictable way over the electrode tip for prolonged time periods.

The results of this study clearly indicate that a stable DC arc direction or continuous arc rotation did not exist in any of the investigated DC furnaces. Therefore, the phenomenon of top joint opening of DC electrode columns must be generated mainly by the absence of the self tightening,

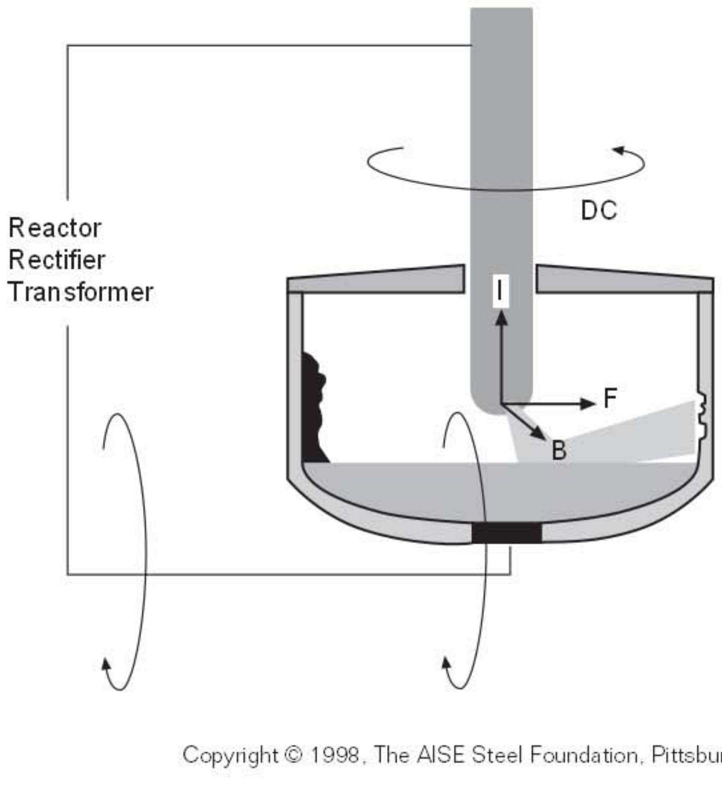


Fig. 10.44 Influence of external magnetic field on DC arc furnace without electromagnetic compensation. *(Courtesy of SGL Carbon Corp.)*

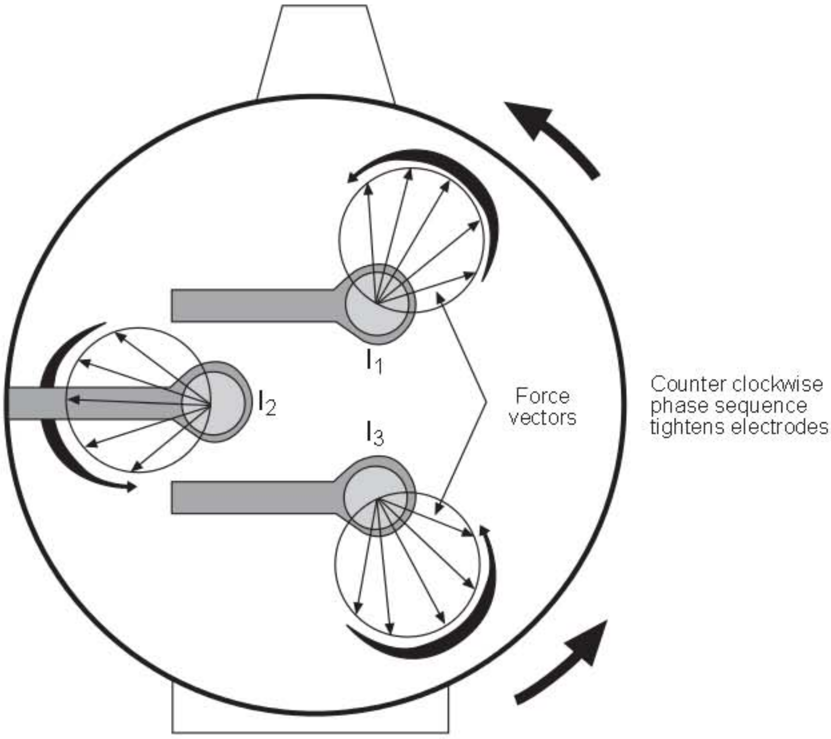


Fig. 10.45 Electromagnetic forces in an AC electric arc furnace. *(Courtesy of SGL Carbon Corp.)*

counter clockwise, electromagnetic force which helps AC operation to maintain tight joints as shown in Fig. 10.45.

Lower powered DC operations suffer specifically from the heat sink effect which is caused by every electrode addition as demonstrated in Fig. 10.46. At low power levels, the necessary thermal nipple expansion supporting the tightness of the top joint does not happen fast enough. The absence of the self-tightening AC effect will be aggravated under these conditions through furnace vibrations, excessive water spray cooling, and, if pitch plugs are used, through joint lubrication by the liquid pitch between 150–250°C.

For higher powered DC operations, the failure mechanism seems to be different. In this case, the contact resistance between the electrode end faces may become too high for the extreme current levels. This phenomenon can be the combined result of the missing self-tightening AC effect, questionable joint design, inadequate tightening torque, or poor electrode addition practice. As a consequence, the current will flow preferentially through the nipple. This can lead to overheating of the nipple to its creep temperature which will result in joint breakage.

A number of counter measures can be taken to prevent these failures. The design of electrodes and nipples should be adjusted by the electrode producer to the demanding DC joint conditions (contact resistance as a function of surface finish, expansion coefficients, and dimensions). DC electrode joints should be locked by nailing after each electrode addition. DC operations should not use the electrode water spray system for at least twenty minutes after each electrode addition. DC shops, with an abnormal number of top joint breaks should apply automated, off- furnace electrode additions or use the automatic on-furnace addition devices available in the marketplace. DC shops should apply the recommended, high tightening torque moments. Table 10.7 presents published torque levels.

10.3.6 Comparison of AC and DC Electrode Consumption

There has been much speculation over the past few years as to the true electrode savings for DC operation compared with AC operation. An analysis by Bowman indicated the following. For a

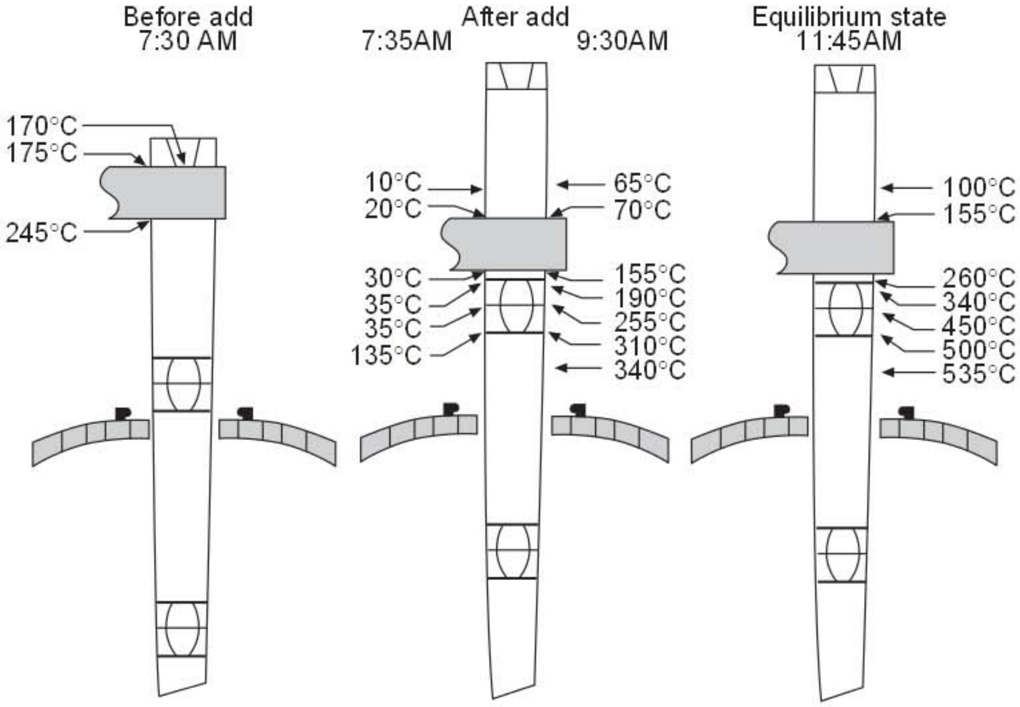


Fig. 10.46 Influence of electrode additions on temperature distribution. *(Courtesy of SGL Carbon Corp.)*

Table 10.7 Recommended Tightening Torque Moments

Electrode Diameter, in.	Recommended Tightening Torque, ft-lbs
24	3000
26	3700
28	4400
30	5500

given size of operation, the DC electrode tip consumption is larger than the combined tip consumption of all three AC electrodes. This is due to the much higher operating current for the DC electrode. DC electrode consumption savings result mainly from reduced sidewall oxidation. Overall savings can be expected to be approximately 25%.

Lefrank et al. have calculated continuous electrode consumption for the DC furnace shops operational in the US in 1995.[10] Then, they compared the calculated consumption figures with the actually observed consumption as shown in Table 10.8. They assumed that the difference between calculated and observed consumption would be equivalent to the amount of discontinuous consumption: breakage, butt losses, and tip spalling. The resulting dependence of discontinuous consumption on average phase current is shown in Fig. 10.47. It is important to note that the data points for the current levels above 100 kA are averages for 28 in. and 30 in., with the 28 in. performing worst and the 30 in. performing best.

With this data, DC operations in the United States can be divided into three major electrode consumption groups:

1. <u>DC furnaces operating below 110 kA</u>: Predominantly continuous, low electrode consumption is achieved with electrodes of 28 in. and smaller diameter. Overall

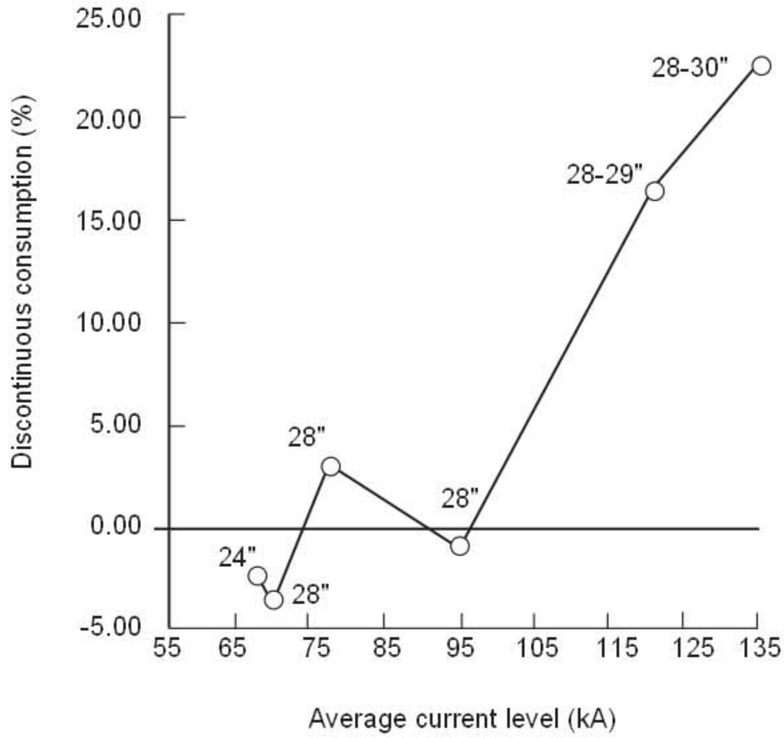


Fig. 10.47 Discontinuous consumption versus kA for DC EAF meltshops. *(Courtesy of SGL Carbon Corp.)*

Table 10.8 Calculated vs. Observed Electrode Consumption for DC EAFs.

EAF Meltshop	Ave. Current, kA	Calculated Side Cons., lbs/ton	Calculated Tip Cons., lbs/ton	Calculated Total Cons., lbs/ton	Measured Total Cons., lbs/ton	Discont. Cons., (%)
A	68	1.62	2.60	4.21	4.10	–2.67
B	70	1.26	1.24	2.50	2.40	–3.91
C	77	1.26	1.56	2.83	2.90	2.62
D	95	0.96	1.96	2.92	2.90	–0.63
E	120	0.82	2.49	3.31	3.85	16.42
F	134	0.81	2.38	3.18	3.90	22.55

electrode savings of 25%, as estimated in Bowman's AC versus DC comparison, are realistic for these operations.

2. DC furnaces operating between 110 and 130 kA: Discontinuous consumption processes increase dramatically with phase currents above 110 kA. Electrodes of 30 in. diameter are necessary to control breaks, butt losses, and spalling. Electrode consumption savings using 28 in. electrodes are marginal at best when compared to high impedance AC shops of similar capacity.

3. DC furnaces operating above 130 kA: 130 kA is currently the maximum recommended current carrying capacity for 30 in. electrodes. DC shops operating above this level are realizing electrode consumption figures higher than high impedance AC shops of similar capacity.

10.3.7 Development of Special DC Electrode Grades

In response to demanding DC furnace conditions, graphite producers are developing special DC electrode grades. SGL Carbon Corp. has published information on a finite element analysis (FEA) model for the evaluation of the influence of graphite property changes on thermal stress generation in electrode columns under DC conditions.[11] Using the information provided by the model along with actual data from DC operations, new electrode nipple systems are being produced for DC applications.

The modifications evaluated have concentrated on:

1. Reduction of joint resistance in combination with high strength nipples with lower thermal expansion coefficients to reduce thermal joint stress and failure
2. Reduction of specific electrode resistivity to lower the generation of thermal stresses resulting from Joule's heating of the electrode center.
3. Enhancement of transverse thermal conductivity to transport the heat from the electrode center faster to the surface.
4. Reduction of coefficients of thermal electrode expansion to lower thermal stress from electrode expansion.
5. Improvement of homogeneity and coarseness of electrodes to increase resistance to crack initiation and propagation.

Shown in Fig 10.48 is an example of the resulting stress levels calculated by the FEA program from the current flow pattern inside the electrode column and the resulting temperature distribution. The material properties of Table 10.9 were used to generate this FEA plot. All columns in this example are loaded with 86 kA.

The FEA results confirm that the highest tangential stress levels are being developed around the bottom joint. Going from the 24 in. electrode with AC graphite properties to the 28 in. electrode with DC graphite properties reduces the joint stress from 2465 to 1160 psi.

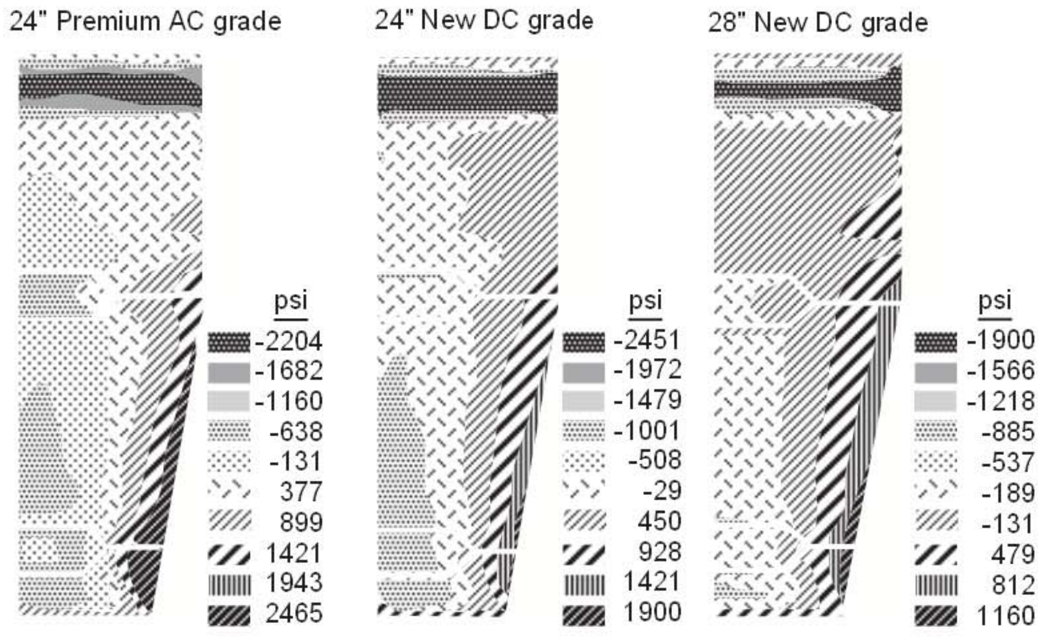


Fig. 10.48 Tangential column stress distributions derived from FEA model. *(Courtesy of SGL Carbon Corp.)*

Table 10.9 Properties of AC and DC Electrodes.

Average Property	Electrodes AC	Electrodes New DC	Nipples AC	Nipples New DC
Apparent density (g/cm^3)	1.71	1.73	1.76	1.78
Special electrical (Ω in. $\times$ 10^{-5}) *	20.5	18.5	13.4	12.6
resistance (Ω in. $\times$ 10^{-5})	33.1	30.3	27.5	26.8
Young's (psi $\times$ 10^6) *	1.35	1.51	2.65	2.54
modulus (psi $\times$ 10^6)	0.7	0.75	0.96	0.94
Coefficient of (ppm/°C) *	0.6	0.5	0.32	0.2
thermal expansion (ppm/°C)	1.4	1.6	1.84	1.7
Thermal (WK^{-1}m^{-1}) *	230	250	315	320
conductivity (WK^{-1}m^{-1})	150	175	175	200

The maximum tangential joint stress from the FEA calculations as a function of phase current is shown in Fig. 10.49. In addition, the strength range for electrode graphite at 2000°C is shown as a bar. Below this level the DC electrodes will mainly be consumed in a continuous way. Above this limit, discontinuous consumption will gradually commence. The FEA data coincide well with electrode performance observations in the field.

It is interesting to note that the FEA model predicts start of discontinuous consumption around 110 kA even for a 30 in. electrode with improved properties. Some DC operations are designed to use operating currents up to 160 kA. Thus, further substantial improvements in electrode material properties will be necessary to control breakage, butt losses, and splitting at such operating currents.

Increasing DC electrode diameters beyond 30 in. would improve the current carrying capacity, but with decreasing efficiency because the thermomechanical stresses in these electrodes will increase exponentially with increasing diameter. Alternatively, the current load for high powered DC operations can be distributed over two electrodes with the objective to reduce electrode stress levels. Two DC arc furnaces with double electrode systems have recently started operation. Reliable performance comparisons are not yet known.

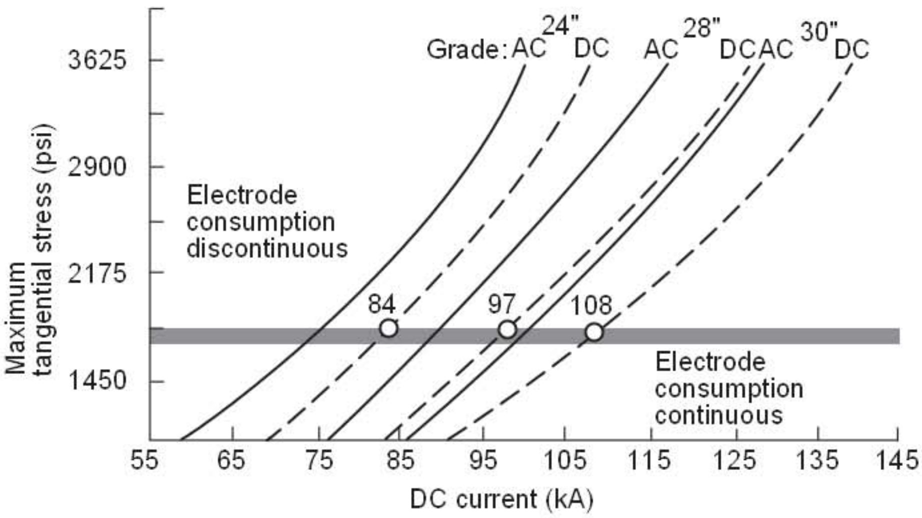


Fig. 10.49 Maximum tangential column stress versus current load. *(Courtesy of SGL Carbon Corp.)*

10.4 Gas Collection and Cleaning

Over the past 30 years electric arc furnace fume systems have evolved considerably from very simple systems aimed at improving the ambient work environment around the furnace to sophisticated systems aimed at controlling not only the emission of particulate but also the control of toxic gases. Modern fume systems are now designed to minimize the formation of toxic gases and to ensure that others are destroyed before exiting the system. Thus gas cleaning in the modern day fume system entails considerably more than trapping and collection of particulate.

Initially, electric arc furnaces operated without any emissions control. In the earlier days, operators merely sought to improve workplace conditions in order to increase productivity. The advent of environmental regulations saw the progress of primary fume control on electric arc furnaces from side draft hoods (SDH), through furnace roof hoods, to rudimentary fourth hole extraction systems, culminating in today's sophisticated direct evacuation systems (DES).

10.4.1 Early Fume Control Methods

As mentioned above, early electric arc furnaces operated without any emissions control and operators merely sought to improve workplace conditions. Most of the EAFs were relatively small in size and in some cases the roof was stationary. As a result many of the early fume control systems involved a roof mounted type of furnace hood. These allowed for effective fume control without requiring elaborate pressure control in the furnace. In addition fume collection did not adversely affect the steel metallurgy. For larger furnaces (>5 metre diameter), furnace hoods were cumbersome and not generally recommended. Typical examples of furnace hoods included the side draft hood, the full roof type hood and the close fitting (donut) hood. Another method that was sometimes used was a mobile canopy hood that is local to the EAF. These are discussed in more detail in the following sections.

10.4.1.1 Side Draft Hood (SDH)

A side draft hood is depicted in Fig. 10.50(a). This type of furnace hood picked up the furnace fume as it exited the electrode ports. Additional branches were sometimes used to collect emissions from the slag door and from the tapping spout. Gas cooling was accomplished by bleed-in air. Typical advantages of this type of hood include: minimal air volume required to give effective control, no interference with electrode stroke, no effect on steel metallurgy, hood does not need to be removed when changing the roof, and complete accessibility to furnace roof components.[12] Disadvantages include: cumbersome for large furnaces, fume escapes at roof ring if there is not a good seal, and difficult to get good capture at B phase due to electrode configuration.

10.4.1.2 Full Roof Type Hood

The full roof type hood configuration is shown in Fig.10.50(b). The full roof type hood encloses the roof entirely allowing for capture of any fume exiting the upper portion of the EAF. This hood design is often less expensive than the side draft design but has several disadvantages: electrode stroke is decreased by 15–24 in. depending on the furnace size, parts of the hood must be removed when the furnace roof is changed, hood covers entire top of EAF making maintenance difficult, openings around the periphery of the hood make fume control difficult, hood has difficulty handling very large offgas volumes, and requires refractory lined or water-cooled sections above the electrode openings in the EAF.[13]

The Pangborn modified full roof hood is shown in Fig. 10.50(c). This has the following advantages over the standard full hood configuration: better utilization of ventilation air (lower volumes than full hood), some fume pickup during furnace charging, less weight than a full hood, and less time required for roof change.[14]

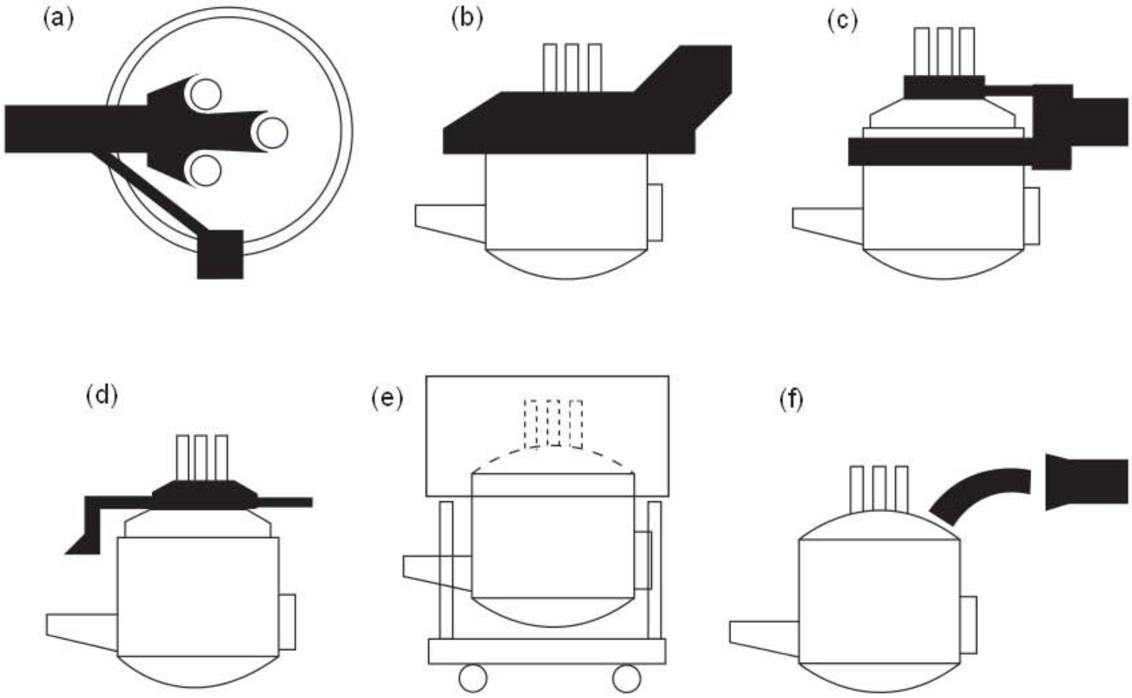


Fig. 10.50 Hood configurations: (a) side draft, (b) full roof, (c) Pangborn modified full roof, (d) close fitting, (e) mobile canopy, (f) snorkel system.

10.4.1.3 Close Fitting Hood

The close fitting hood is shown in Fig. 10.50(d). This type of hood is only really applicable for small furnaces. This hood can only be used if the roof is not raised during oxygen lancing. The hood is generally constructed of stainless steel, heavy reinforced carbon plate or may be water-cooled. The main section has a built-in frame that is mounted to the furnace roof ring. The section over the slag door is permanently attached to the bezel ring. This hood arrangement is relatively tight and gives excellent fume control. Disadvantages include reduction of the electrode stroke and difficulty in accessing the roof. This type of design has been incorporated into ladle furnace roofs for fume control.

10.4.1.4 Mobile Canopy Hood

The mobile canopy hood is as its name suggests, a mobile hood that is moved into position above the furnace to capture fume as it rises from the furnace. This type of hood is only suitable for very small furnace operations (less than 10 tons). A mobile canopy hood is shown in Fig. 10.50(e).

10.4.1.5 Snorkel System

The snorkel system originated in Germany and is used on larger furnaces melting open hearth grade steels.[12] The snorkel system configuration is shown in Fig. 10.50(f). The system employs a fourth hole in the furnace roof followed by a water-cooled elbow. The snorkel is a funnel shaped section of duct that can be retracted with a motor or hydraulic system. The snorkel captures fume as it exits the furnace elbow. The gap between the elbow and the snorkel, 5–120 cm (2–48 in.), allows air to enter the system. This air provides oxygen for combustion and also provides dilution cooling for the system. The fourth hole serves only as a natural pressure release for the furnace. During periods of heavy fume emission, the snorkel is moved closer to the elbow to provide greater suction. The furnace atmosphere remains undisturbed and theoretically operates at a positive pressure at all times.

10.4.2 Modern EAF Fume Control

Essentially all phases of normal EAF operation result in either primary or secondary emissions. Primary emissions are those that are produced during EAF melting and refining operations. Secondary emissions are those that result from charging, tapping and also from escape of fume from the EAF (usually from the electrode ports and/or roof ring). Primary emissions are generally controlled using a direct evacuation system. Secondary emissions are captured using canopy hoods and in some cases auxiliary tapping hoods.

10.4.2.1 Direct Evacuation Systems

The Direct Evacuation System (DES) or fourth hole system as it is commonly known, is probably the most prevalent fume control system in meltshops today. An electric arc furnace (EAF) offgas system is comprised of containment, cooling and collecting elements.[15] Total offgas handling usually involves a direct evacuation system (DES) and a secondary fugitive capture system consisting of a canopy hood in the shop roof.

In a typical DES system, containment components include the fourth hole, system ducting and the fan. Cooling is achieved by elements such as water-cooled ducting, dilution air cooling, evaporative cooling and forced or natural draft heat exchangers.[15] The latter three provide gas cooling without increasing the collector size.

Particulate collection is typically achieved with a baghouse, though scrubbers and electrostatic precipitators are used in some cases.

A good offgas system is one that combines the containment, cooling and collecting elements economically to provide effective fume control within regulatory standards, for both the meltshop and external environments.

A modern DES addresses not only particulate capture but also control of CO, NO_x and VOC emissions.

The main advantages of a DES system are: it requires the lowest offtake volume for furnace fume control, it provides the most effective method for fume capture in UHP steelmaking, it allows low interference with meltshop operations, it requires minimal space on the furnace roof, and it addresses the issues of CO and VOC combustion. The main disadvantages of DES systems are: excessive in-draft leads to increased power requirements and undesired metallurgical effects, and it is difficult to control the furnace freeboard pressure when offgas surges occur. A typical system flowsheet is given in Fig. 10.51.

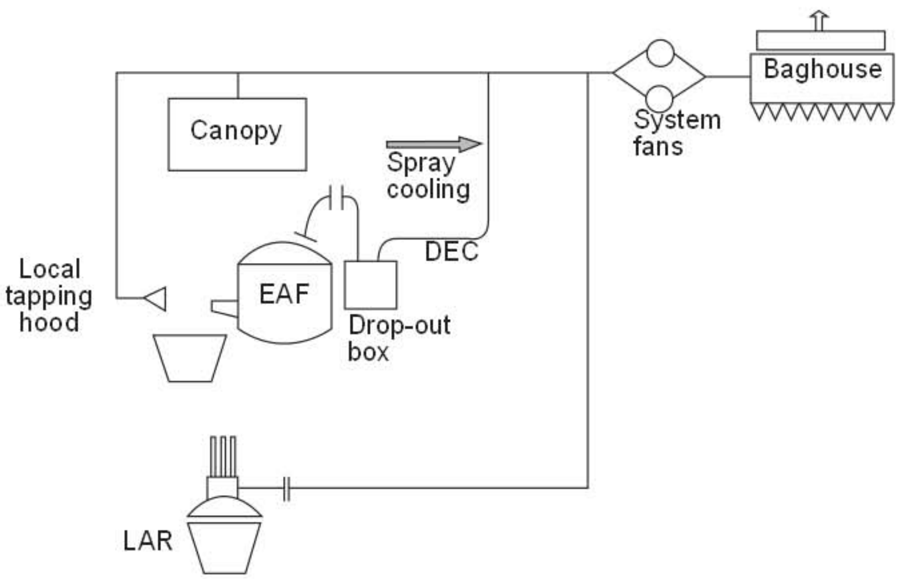


Fig. 10.51 Typical offgas system flowsheets. *(Courtesy of Hatch Associates)*

10.4.2.2 DES Design Methodology

Design of an offgas system requires consideration of several steps including containment of furnace fume, the reaction processes going on in the furnace, and processes through the offgas system itself. In the past there has not been enough collaboration of the furnace operator with the offgas system designer. As a result it has been difficult to provide systems which matched operations within the meltshop. Without a clear understanding of the furnace operations, it is not possible to design an appropriate offgas system.

Essentially, the design of the offgas system involves a detailed mass and energy balance around the EAF to provide the basis for gas quantity and heat content leaving the EAF. There are a variety of furnace process factors that influence the peak offgas flow rate and the peak heat load contained in the offgas. These factors include the charge weight, the scrap mix (its oil, combustible material and moisture contents), furnace power and power-on time, tap-to-tap time, furnace openings and associated air infiltration (slag door, electrode ports, roof ring, auxiliary ports), oxy-fuel burner and oxygen lance rates and duration of use, carbon injection rates, and use of alternative iron sources and feed rate.

Both the gas quantity and its heat content affect the sizing of the DES system. It should be noted that the system fan is a constant volume device. The fan is capable of moving a greater mass of offgas if the offgas is cooler (the cooler the offgas, the lower its volume). At the same time, the fan motor has a limited power rating. As a result, to provide maximum evacuation for a given DES system configuration, it is desired to cool the offgas as much as possible but at the same time ensure that we are staying within the amp current rating of the fan. Thus, it is common to modulate the fan damper position based on the fan motor current and an acceptable setpoint.

In order to keep the gas cleaning system cost economical, it is usual to cool the offgas prior to the baghouse. This allows for use of lower cost bag materials and also keeps the volume of gas to be cleaned to a minimum.

It is important to note that not all of the heat content in the offgas is necessarily sensible heat. If the offgas contains non-combusted species such as CO, a significant amount of energy can be contained in the offgas. This not only represents increased heat removal duty for the offgas system, but also energy lost to the melting process and hence an inefficiency.

The furnace heat cycle includes phases such as flash-off (the first few minutes of melting when the moisture and volatile components of the scrap mix are driven off by the applied heat), melting, which can have several operational phases depending upon the use of oxy-fuel burners, oxygen lancing or DRI additions, and refining, which can be broken down into periods of oxygen lancing, carbon injection and foamy slag practices. It is important to analyze all of these phases to determine the peak offgas flow rate and peak heat load.

Frequently the peak load in terms of gas volume occurs during the first few minutes of melting, when the volatile materials burn and flash-off. However, refining is often seen to result in the highest offgas temperatures during oxygen lancing for carbon removal. Operations where DRI is added continuously can lead to high evolution rates of CO gas. CO evolution rates can be even greater during iron carbide addition. If this CO does not burn in the furnace, high heat loads to the offgas system will result.

Typical peak load offgas characteristics at the fourth hole for effective EAF emission control are an offgas flow rate of 4–6 Nm^3 (150–200 scf) per ton per hour tapped and an offgas temperature of 1370–1925°C (2500–3500°F).

Fourth hole sizes are selected on the basis of a design offgas velocity in the range of 2400–3000 m/min (8000–10,000 ft/min). The gas velocity at the fourth hole is important because it will affect the system pressure drop at the fourth hole and also will affect the amount of material which will carry over from the furnace to the offgas system. In some cases, slag and scrap will be carried over into the offgas system.

A good understanding of the furnace processes and how they affect the offgas characteristics is necessary for the process engineer to design an efficient and cost effective DES. Once the flow rate and heat content for emission control at the fourth hole has been determined for the EAF, the required performance of the DES has been defined. If some of the components of an existing system are to be retained (e.g. system fan, baghouse) then certain limitations on the total allowable system pressure drop will also be fixed. Definition of the heat content of the offgases also fixes the amount of cooling required prior to routing the offgas to a given baghouse filter operation. The fourth hole serves as the union between the furnace and the DES.

In order to control fume emissions, it is necessary to maintain a slight negative pressure within the furnace. This pressure is typically in the range of –0.09 to –0.18 mm Hg (–0.05 to –0.1 in. W.G.). It is important to limit the resultant in-draft to the minimum required to contain fume generated by the melting process. Higher in-draft results in heat losses from the furnace and higher offgas flow rates. Test results presented by Bender indicate that a negative pressure of –0.09 mm Hg (–0.05 in. W.G.) will usually prevent electrode emissions when scrap cave-ins occur.[16] Bender shows, based on measurements at Von Moos Stahl (Swiss Steel AG), that operating with too high a negative pressure can increase the electrical power requirement by approximately 45 kWh/ton. Previous trials on several furnaces in Japan indicated that running the furnace at positive pressure gave power savings of 10–20 kWh per ton.

Furnace openings of significance are usually the electrode ports, the roof ring gap, the slag door and auxiliary ports in the furnace sidewall (decarburization oxygen lance, solids injection etc.).

The minimizing of these openings improves the efficiency of both the furnace and the emission control system. Openings that occur higher up in the furnace are less critical than those which occur low in the furnace due to the pressure profile across the height of the furnace. The pressure tap which is used to measure the pressure used for furnace draft control is located in the furnace roof. If the system controls to the pressure setpoint at the roof level, the negative pressure at the low furnace openings such as the slag door will be much greater. The flowrate of air into the furnace through openings is proportional to the square root of the differential between the furnace pressure and the ambient pressure. As a result the infiltration rate will be much higher at the openings lower in the furnace.

Early in the meltdown phase, furnace openings will not result in much air infiltration because the scrap will act as a resistance to infiltration flow. However, later in the cycle when oxygen lances are inserted through the slag door, considerable air infiltration can result.

The fourth hole serves as the union between furnace operations and the DES. Gases exiting the fourth hole enter the water-cooled elbow mounted on the furnace roof. Generally, there is a gap between the furnace roof and the water-cooled elbow which admits air due to the negative pressure. The air mixing with the offgases provides opportunity for further combustion of offgas and/or dilution cooling. In general the mass of air drawn into this first gap matches the mass of gas flowing out of the fourth hole.

A second gap that allows the furnace to tilt and the furnace roof to swing is normally situated between the water-cooled elbow exit and the entrance to the water-cooled duct. This gap admits additional infiltration air which may provide some further combustion as well as dilution cooling. In some cases a duct section similar to a snorkel is located following the gap. This section is adjustable typically to give a gap of 2.5–12 cm (1–5 in.). This allows better control of the negative pressure in the furnace freeboard and is often necessary in providing sufficient retraction to enable the furnace roof to swing without interference. Again, in general, the mass of air entering the second gap is similar to the total mass exiting the elbow.

Thus the DES typically exhausts a mass flow that is about four times that which actually exits the furnace at the fourth hole. Generally the designer prefers that these final exhaust gases contain 100% of oxygen in excess of the combustion needs for all combustibles generated in the furnace process.

There have been many innovations to the water-cooled elbow design in recent years. While it is recognized that sufficient combustion air needs to be introduced to ensure combustion of all VOCs and CO in the water-cooled duct, it is also necessary to ensure that excessive air is not introduced which could lead to over cooling of the offgas and non-combusted material downstream in the system. Flanges (also known as elephant ears), are sometimes welded around the water-cooled duct and the elbow so that the amount of air infiltration can be better controlled. The water-cooled duct is sometimes sized larger than the elbow so that efficient capture can still be achieved even when the furnace is tilted ±5°. One of the more recent innovations is to use a very steep elbow configuration which will help to reduce slag buildup. Bender recommends that this type of elbow be coupled with a high velocity water-cooled duct entrance followed by a sharp bend downward.[16] This configuration provides for good turbulence for mixing and combustion of the CO and slag can fall out into a dropout box located below. Good mixing of the gas is very important to ensure complete combustion.

Water-cooled ducting is used to cool the offgas and also to contain the offgas combustion reaction heat which would otherwise damage carbon steel ductwork. In some older systems, refractory lined ductwork was used but with improved design and lower costs; water-cooled duct is now the preferred choice. Water-cooled duct is also generally less maintenance intensive. It is important however, to ensure that water-cooled duct is accessible so that routine maintenance can be performed. In many of the newer meltshop installations, the water-cooled duct has a sharp downward bend following the combustion gap. A dropout box is located at the bottom of this bend and then a horizontal run of water-cooled duct is located at grade. This makes it easier to use forklift trucks to remove sections of the duct for maintenance. This also makes the duct accessible for periodic cleanout of any dust buildup which might occur. Some planning is required to ensure that the layout of the ducting does not interfere with other operations.

The gas temperature at the end of the water-cooled ducting is typically 650–760°C (1200–1400°F). Below this range, water-cooled ducting is no longer an efficient method of cooling due to low heat transfer coefficients. Typically offgas flowrates at the water-cooled duct exit for effective EAF emission control are 14–23 Nm^3 (500–800 scf) per ton per hour tapped.

10.4.2.3 Gas Cooling

Prior to entering the carbon steel dry ducting, dilution, evaporative or non-contact cooling is generally used to cool the gas so that the duct wall temperature is well below 425°C (800°F). This generally equates to a gas temperature of about 650–760°C (1200–1400°F).

Evaporative cooling consists of spraying atomized water into the offgas stream. The water will evaporate, absorbing a large quantity of energy. Evaporative cooling is usually carried out following the water-cooled duct. This can be carried out in a spray chamber or in the ductwork. If carried out in ductwork, care must be taken that the water stream evaporates fully before it strikes the duct wall. The evaporation rate is dependent on the water flowrate, the spray angle, quantity of atomizing air, the water droplet size and the presence of contaminant films on the droplet. Evaporative cooling will result in a decrease in offgas volume and is usually the most cost effective method for achieving this because it involves the minimum amount of equipment. The only potential downside to this technology is the possibility of condensation occurring if the offgas becomes saturated. It is important to evaluate the total moisture content of the combined DES and secondary offgases entering the baghouse. If a dewpoint problem exists, condensation will occur which can affect bag performance and life. If this is the case, an alternative to evaporative cooling may have to be selected.

Dilution cooling consists of adding air to the offgas stream to cool it. This will always result in a larger gas volume and increases the required capacity of the fume extraction system. This is the lowest cost method for cooling of the offgas but may adversely affect the cost of system components downstream.

Non-contact cooling is carried out using forced draft coolers or hair pin coolers. The gas is cooled by transferring heat to the wall of the cooler which in turn transfers heat to the outer surface of the tube or duct wall. The net result is to cool the offgas without an increase in offgas volume. Though this is an efficient method for gas cooling which does not result in increased gas volume, it is the most expensive method and requires the greatest amount of equipment.

Further cooling is provided by dilution air drawn in through the canopy hood and mixed with the DES offgases to reduce the combined gas temperature to around 120°C (250°F) before entering the baghouse. The reason for this is to allow polyester filter media to be used. Polyester is an economic filter media commonly used in EAF air pollution control (APC) systems with a design temperature limit of 120°C (250°F) for continuous operation.

Gas velocities in the DES system ductwork are important for many reasons. If the gas velocity is too high, large system pressure drops will result and there is the potential for increased duct wear. If the gas velocity is too low, the particulate in the offgas will settle out in the ductwork. This will also lead to higher system pressure drop and poses a maintenance problem. It has been found in practice that a design velocity of 4000–5000 fpm at standard conditions is optimum for meeting both dust entrainment and economic criteria.

For any configuration, a detailed mass and energy balance must be performed on each DES system component to yield gas flows, temperatures, velocities and pressure losses throughout the entire system. Analysis of various system configurations can be a tedious and time consuming process. As a result, several system designers have incorporated all of the necessary calculations into a computer model. Thus, the effect of changes in process and system design can be evaluated quickly allowing for greater optimization of system design. A typical DES is shown in Fig. 10.52.

10.4.3 Secondary Emissions Control

Control of secondary fume emissions from furnace charging, tapping and slagging operations became strictly regulated subsequent to regulations for primary control. Secondary emissions also result from offgas escaping the furnace around the electrodes and at the roof ring. Initial treatment of secondary emissions were provided by generally inefficient canopy hoods at roof truss level.

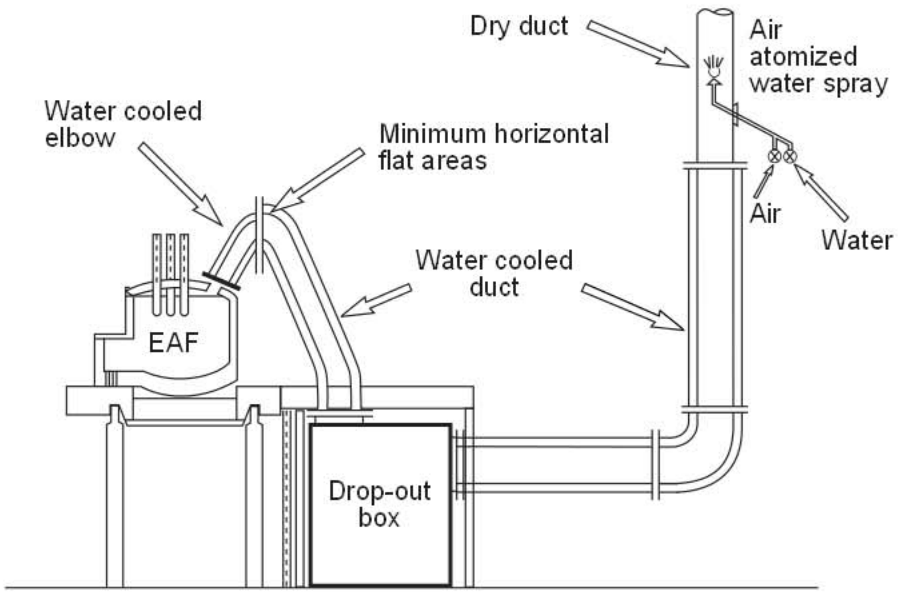


Fig. 10.52 Typical DES configuration. *(Courtesy of Hatch Associates.)*

Modern systems tend to provide much greater evacuation capacity. The sizing of the secondary fume control system is the controlling factor of the overall sizing of the gas cleaning system because secondary extraction tends to be four to six times that required for the DES.

10.4.3.1 Canopy Hoods

Canopy hoods were installed initially to capture primary furnace emissions. As furnace operations became larger, a need arose for control of secondary fume emissions during charging and tapping operations. The canopy hood was chosen to fulfill this need because it could capture these emissions without interfering with operations. Canopy hoods are located in the building trusses above the crane runway. In North America, canopy hoods now form an integral part of modern meltshop fume control systems. The canopy system is throttled back during furnace operation and provides air for dilution cooling of the DES offgases prior to entering the baghouse. During charging and tapping operations the DES does not draw any fume and full extraction capacity is diverted to the canopy hood.

The ability of a canopy hood to capture fume depends on the face area of the canopy, the canopy shape, the canopy volume, the amount of interference presented by the charging/tapping crane, the air flowrate at the canopy face, heat sources within the meltshop (which may cause errant drafts), the height of the canopy hood above the furnace, and the absence of cross drafts within the meltshop.

Many canopy systems are compartmentalized so that one portion of the canopy is used for charging emissions control while another section is used for tapping emissions. With the move towards the use of bottom tapping furnaces, it is impractical to attempt fume control during tapping with a canopy and it is now common to use a side hood for capture of tapping fumes. Hood capture can be greatly affected by cross drafts in the shop and by thermal gradients within the shop. Thus it is important to try to keep doorways closed and to eliminate heats sources as much as possible. One simple way of reducing thermal gradients is to duct ladle preheater gases outside the building. Removal of full slag pots from the building can also provide beneficial effects.

10.4.3.2 Deep Storage Canopy Hoods

Many canopy hood designs are incapable of handling the large plumes generated, particularly during charging. The solution is to provide for temporary storage of this fume until it can be extracted by the fume system. The result is a deep storage type canopy hood which has a storage volume of five to ten times that of a conventional canopy hood.[17] It provides a large storage capacity that is capable of temporarily storing the large surge in fume emissions experienced during furnace charging and allows the fume to be extracted by the gas cleaning system over an extended period of time. This allows the total extraction capacity of the fume system to be lower and yet still meet surge requirements that occur during furnace charging. The design has been verified using physical fluid dynamic modeling and shows greatly improved capture characteristics as compared to conventional technology. There are several successful installations in North America. The Irish Steel installation was the first of its kind in Europe.[17] The concept is illustrated in Fig. 10.53.

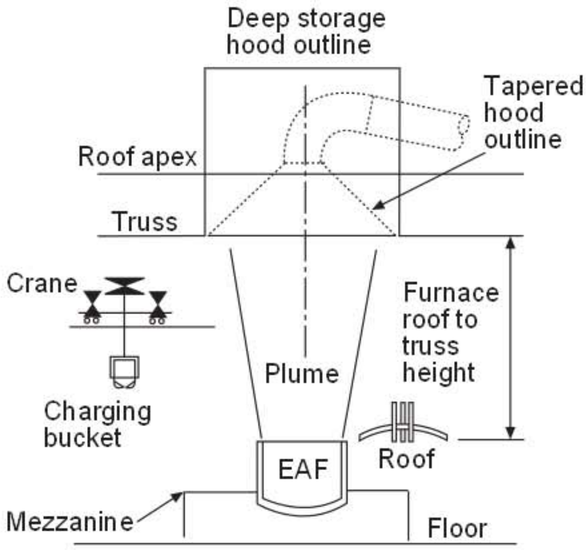


Fig. 10.53 Deep storage canopy hood concept. *(Courtesy of Hatch Associates.)*

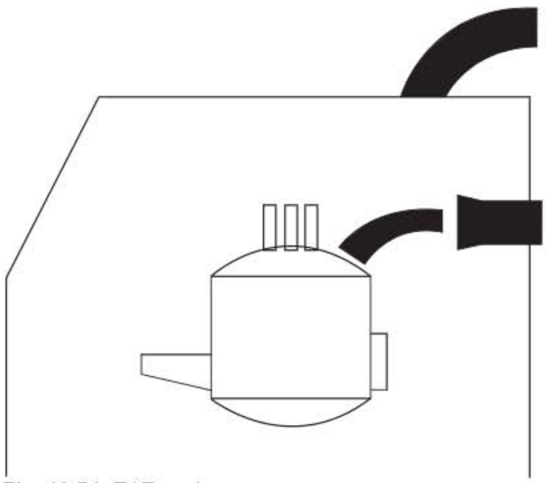


Fig. 10.54 EAF enclosure.

10.4.3.3 Furnace Enclosures

Furnace enclosures (also known as doghouses, snuff boxes or clean houses) are located on the operating floor of the melt shop and totally enclose the furnace. The enclosure generally provides sufficient room for the roof swing and working space around the furnace. A typical configuration is shown in Fig. 10.54. The charge bucket is brought in above the furnace through access doors in one side of the enclosure. The enclosure is low enough that the crane can pass over it. In some cases there is an air curtain sealed slot in the roof of the enclosure that allows the crane cables access when charging to the furnace. Fume is extracted at the roof level adjacent to one of the enclosure walls. Makeup air enters through openings in the operating floor. Tapping ladles are usually brought in on a transfer car. Most modern enclosure systems also incorporate a fourth hole system.

Enclosures provide the benefit that all fume is captured and the extraction volume is considerably less than that required for an equivalent DES/Canopy Hood system. In addition enclosures have no detrimental effects on metallurgy in the EAF and noise levels are reduced substantially (10–25 dB A).[18] It has been reported that energy savings of up to 20 kWh/ton may be achieved due to operation of the furnace at positive pressure. Savings due to lower evacuation volumes have been reported in the range of 10–15 kWh/ton.[4]

Enclosures offer several advantages including cooler, cleaner, quieter and safer meltshop working environments, reduced crane maintenance, lower total emissions, and lower fume system capacity requirements. Some disadvantages include restricted furnace accessibility for furnace maintenance, retrofit applications may be impractical due to the constraints of the existing meltshop layout, and furnace charging operations are sometimes slower than for a conventional meltshop.

In some cases some of the benefits of enclosures can be achieved without fully enclosing the EAF. These are called partial enclosures and consist of free standing walls which do not restrict crane access to the furnace. These baffles help to dampen out noise and minimize the effect of cross drafts within the meltshop.

10.4.3.4 Enclosed Meltshop

An alternative to the furnace enclosure is to close the meltshop and provide total meltshop evacuation. Obviously, the required evacuation capacity for a conventional meltshop layout using this method is much greater than would be required for a canopy/DES configuration. In addition, inadequate extraction could lead to high ambient temperatures in the meltshop and unacceptable working conditions if dust were to drop out within the shop. The solution to these concerns was to provide a compact meltshop layout which provides only enough space to accommodate the EAF and the charging crane. This is shown conceptually in Fig. 10.55. This configuration isolates the furnace melting operations from all other operations within the steelmaking facility. Scrap buckets are transported into the enclosed area using transfer cars. Likewise, the tapping ladle is moved under the furnace on a transfer car. Slag is collected in slag pots which are removed using pot haulers through a dedicated doorway. This emission control configuration has been implemented in several meltshops in Europe and in North America and has been shown to be a cost effective method of providing improved fume control for EAF operation.

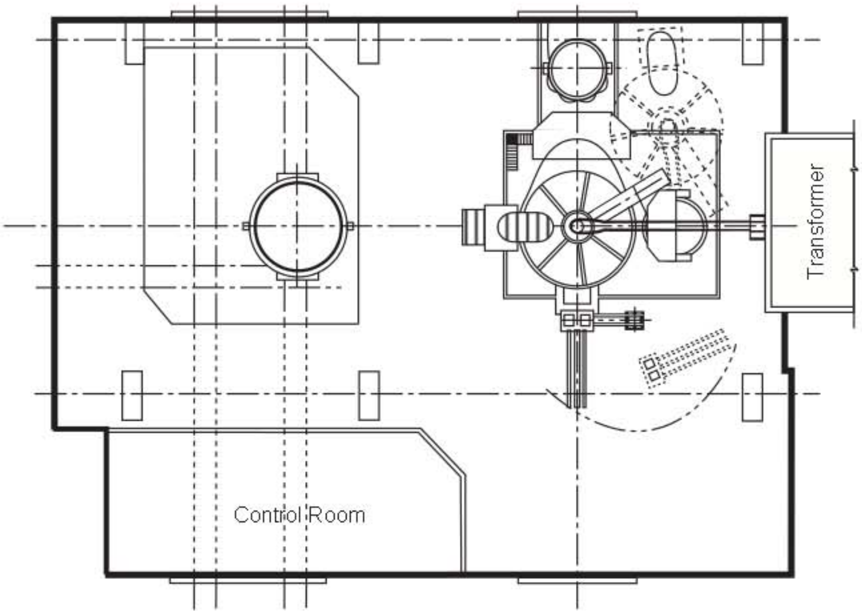


Fig. 10.55 Enclosed EAF meltshop. *(Courtesy of Bender Corp.)*

10.4.4 Gas Cleaning

Following evaluation of primary and secondary gas extraction requirements, the gas cleaning system including system fans, the baghouse and the material handling system can be specified. Some important aspects of equipment selection and design related to specific parts of the gas cleaning system are covered in the following sections.

10.4.4.1 System Control

Many operations within the meltshop have fairly constant exhaust requirements which are independent of the furnace operation. Such sources (ladle furnace, secondary dedusting) can be effectively controlled using booster fans to maintain constant exhaust flows when required, independent of the influence of furnace operations, which may switch large exhaust volumes from one source to another throughout the normal course of furnace operation.

Fume collection systems typically consume 60–100 kWh per ton of good billets produced. As a result, energy conservation has become an important parameter for melt shop profitability. More effort is now placed on identifying potential energy savings in emission control systems. Canopy hood exhaust during charging is by far the largest single exhaust requirement of typical emission control systems. Outside of the charging period, most systems have capacity in excess of that required for DES and other secondary requirements. This is usually applied to the canopies as a simple method of utilizing fan capacity. Some systems achieve significant energy savings by using variable speed fans. In such a configuration fans are turned down during non-charging periods.

Care must be taken, however, because exhausting excess non-charging capacity through the canopies is a way of providing general building ventilation, removing excess heat and fugitive emissions not captured at source. Elimination of this excess capacity may result in higher ambient temperatures and a dirtier shop. Adjustable speed fans are better suited to operations using furnace enclosures, which inherently contain and shield the meltshop from heat and fugitive emissions related to the furnace operations.

10.4.4.2 Fans

Proper specification of fan design parameters is essential to achieve desired fan performance.[19] A typical centrifugal fan performance curve is illustrated in Fig. 10.56. The shape of the system curve, initiating at the origin of the graph, demonstrates the squared relationship between fan static pressure (FSP) and volume flow (i.e. FSP varies directly with the square of Q). The system curve reflects how the system losses change with variations in flow (note this applies to a single damper arrangement in the system because changing damper positions can change the system curve). The fan curve, which intersects the system curve, shows the design FSP produced under different volume flows. The point of intersection of the two curves represents the operating point for the system. If the operating point occurs near the top of the fan curve (point of maximum FSP), unstable surging can result. To avoid this, it is recommended to select an FSP that does not exceed 80% of the maximum FSP. Fans selected near the peak on the pressure curve will be smaller and therefore cheaper but will be at lower efficiency and have no margin.

Due to the system curve relationship between FSP and volume flow, the recommended margins for specifying performance are an additional 10% on design flow and 21% on design FSP. The margins are recommended because fans frequently perform at slightly less than the published fan curve. Some margin should also be allowed for baghouse pressure drop creep above the design performance level.

It is important to remember that the definition of fan static pressure generally used on fan performance curves is not simply the difference in static pressure measured across the fan. One way to define FSP is:

$$FSP = FTP - VP_{OUT} \tag{10.4.1}$$

In equation 10.4.1, FTP is fan total pressure or system total pressure loss and VP_{OUT} is velocity pressure at the fan outlet.

Poor selection and maintenance of baghouses, which can lead to high baghouse pressure drop, can restrict the flow achievable by the fan. A fan selected near the peak pressure output has no chance of meeting the needs of such a system.

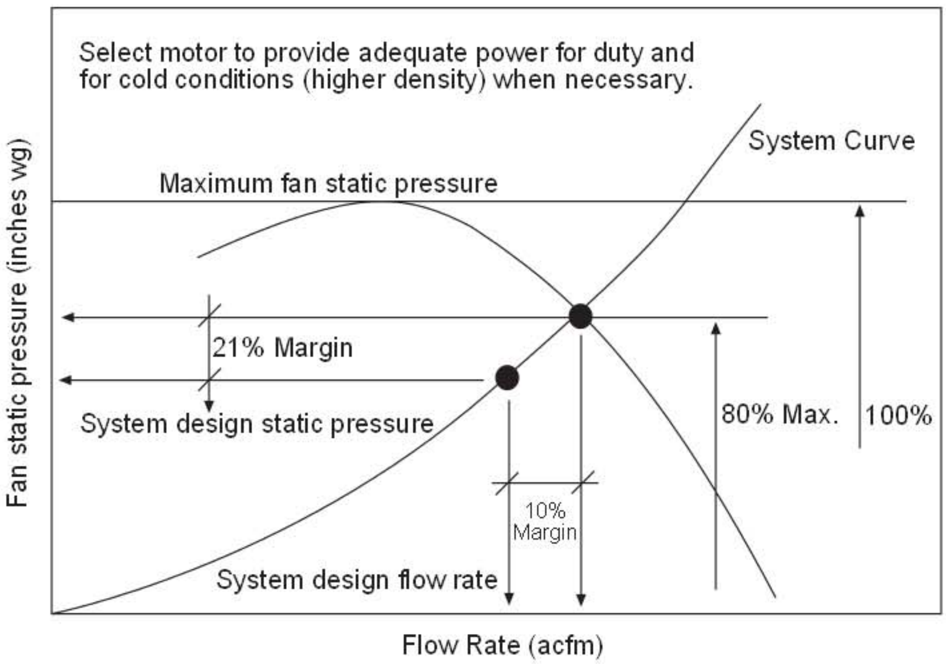


Fig. 10.56 Fan curve and selection margins. *(Courtesy of Hatch Associates.)*

10.4.4.3 Baghouses

A variety of baghouse types and arrangements are available depending on the specific requirements of the particular emission control system. Net refers to the operating condition where one compartment is off-line (such as during online cleaning or maintenance of a compartment). Key baghouse design parameters by baghouse type are given here.

Reverse air baghouses are economical for gas volumes greater than 5600 Nm^3/min. (200,000 scfm).[19] Operating at positive pressure, this design offers cost savings over negative pressure designs. The maximum recommended air to cloth ratio is 2.0–2.5, net. Reverse air baghouses are generally very large in size.

Shaker baghouses are generally similar in application as the reverse air design, but tend to require more maintenance than reverse air designs due to more mechanical movement. It is difficult to achieve the same level of gas cleaning as achieved by reverse air designs. The maximum recommended air to cloth ratio is 1.5–2.0, net.[19]

Pulse jet baghouses are economical for gas volumes less than 5600 Nm^3/min. (200,000 scfm), are of modular construction and operate under negative pressure. The maximum recommended air to cloth ratio is 4.0, net.[19]

These design parameters are general and are meant to be guidelines for baghouse selection. Individual requirements must be determined on a site by site basis.

Reverse air baghouses can be both positive or negative pressure designs. Positive pressure baghouses are cheaper but use dirty-side fans. Dirty-side fan applications have been successfully applied in many operations, but attention is required in impeller selection to minimize wear.

Negative pressure reverse air baghouses are more expensive than positive pressure baghouses because they need to have air-tight construction to avoid leakage. This is generally not an issue with positive pressure baghouses where the baghouse shell encloses the clean-side of the bags and exhausts to atmosphere (only hoppers need to be gas-tight because they see the dirty gas). Also, leakages in positive pressure systems are noticeable, while negative pressure system leakages are not evident and frequently remain undetected. Resulting differences between inlet and outlet flows as high as 15% have been measured.

Large pulse jet baghouses, due to the sheer number of valves and associated equipment, require more maintenance. Their modular construction tends to make them economical for smaller volume requirements (less than 5600 Nm^3/min. or 200,000 scfm).

Insufficient baghouse capacity translates into insufficient cloth area in the baghouse. The type of baghouse and cloth material used determines the maximum flow that can realistically be drawn through the bag filter.

A common design consideration in all baghouses is that the hoppers should never be used for dust storage. Hoppers should be continuously emptied. Allowing dust to build up in the hopper provides an opportunity for dust re-entrainment, increasing the load on the bags, the propensity for fires and potential for spark damage.

Bag damage can occur for a number of reasons including wear, direct impaction of particulate, spark burn, hot flame burn, and high temperature operation. Wear can be a result of poor gas distribution causing bags to swing (for pulse jet baghouses with cage-mounted bags below the tube sheet) or improper tensioning of bags allowing them to rub against each other or compartment walls (for reverse air or shaker baghouses with spring-mounted bags above the tube sheet).

Direct impaction is related to dirty gas distribution, where particulate can follow a direct path from the gas inlet to the bags. The way in which the dirty gas stream is introduced to the bags is a critical aspect of hopper design. Direct impaction of dirty gas on the bags can result in bag wear and premature bag failure. One method of avoiding this problem is to design a high profile hopper with a series of vertical baffle plates as illustrated in Fig. 10.57. This forces an extreme change in direc-

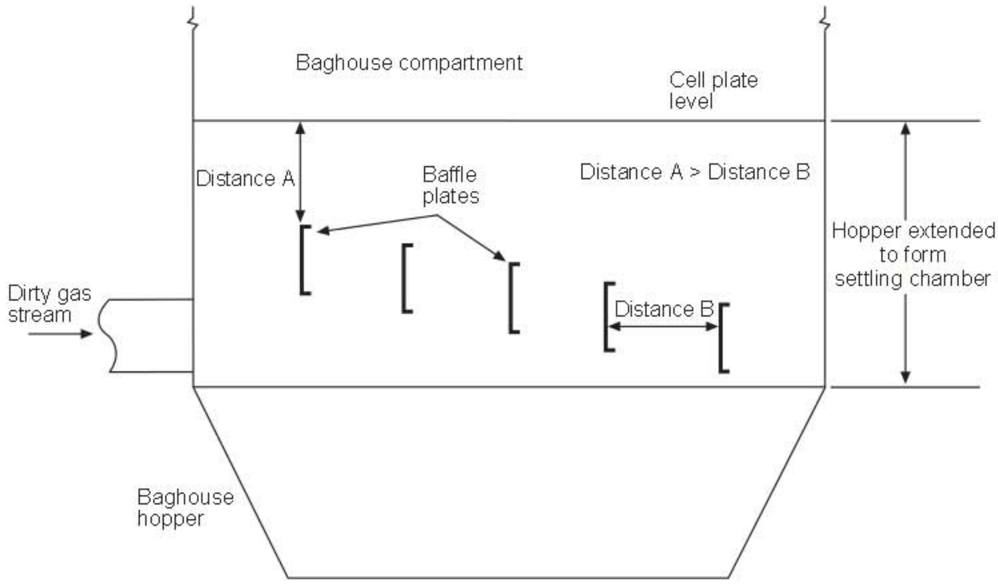


Fig. 10.57 Recommended baghouse hopper baffle plate arrangement. *(Courtesy of Hatch Associates.)*

tion for the particulate causing it to lose its momentum before reaching the bags. This method of dirty gas distribution can be achieved with minimal pressure drop.

Spark burns are normally associated with particulate still combusting as it reaches the baghouse. Hopper baffle plates can help reduce this problem as it is normally associated with larger particulate. An adequately sized dropout box can also remove these large particulate from the gas stream prior to the baghouse inlet.

Hot flame burn can occur when the duct length between the baghouse and canopy is relatively short. Without major modifications to the system configuration, protection from fireballs reaching the baghouse can be achieved using thermocouples tied in to control of emergency dilution dampers.

High temperature operation for extended periods can gradually weaken bag fabric leading to premature failures.

10.4.4.4 Material Handling

Material handling in emission control systems is achieved with a variety of equipment types.

Pneumatic conveying systems can be either dense or dilute phase design. For EAF dust transport, it is recommended to use dilute phase. The pneumatic lines should be a minimum 15 cm (6 in.) diameter.[19] Well designed systems can provide dependable service with minimal maintenance requirements.

Air slide/air lift systems operate by fluidizing the dust to be conveyed. In the case of slides, the equipment includes an enclosed chute mounted diagonally (minimum 7° slope) to allow the dust to flow down the length of the slide as it is fluidized.[19] The chute configuration increases the overall height of the baghouse. Air lifts incorporate low velocity vertical conveying of the collected particulate in a pipe over a fluidizing ejector. Such systems have low wear and low maintenance.

Mechanical conveyors can include any such types as screw conveyors, drag chain conveyors and bucket elevators. It is important that these systems be adequately sized (both in terms of quantity throughput and motor power) for surge conditions. Conveyors discharging from baghouse hoppers should be capable of starting under full hopper conditions.

Mechanical conveying systems will often require dedusting exhaust capacity. In order to avoid fugitive emissions (or in the case of negative pressure systems, in-leakage which robs capacity and re-entrains dust) it is important for the equipment to be well sealed. Routine maintenance is essential for maintaining such systems in good working condition.

For convenient operation, storage silos should be sized for a minimum three day storage capacity. This is so that material discharge can be carried out during day time shifts. It also allows the EAF operation to continue over a long weekend without bringing in staff for dust disposal.

Typically dust control is required for clean operation of silos. This can be accomplished by either an independent bin vent filter mounted on the silo, or a direct connection from the silo to the inlet side of the baghouse (in the case of positive pressure baghouses this connection would be on the inlet side of the fans). In either case, the silo is maintained under a negative pressure so that any openings in the silo will see an infiltration of air rather than a puffing out of dust.

Silo dischargers are also candidates for dust control. One effective design is a telescopic discharge which incorporates a concentric flexible discharge with exhaust applied to the outer section. The source of exhaust can be either of the methods described for the silo itself.

10.4.5 Mechanisms of EAF Dust Formation

Currently about 600,000 tons of EAF dust are generated each year in the U.S. The amount of dust generated is usually in the range of 9–18 kg (20–40 lbs) per ton of scrap melted. The reason for the wide range is due to differences in scrap quality, operating practices, dust capture efficiency and methods of introducing raw materials into the EAF. Currently the Center for Materials Production, (a group sponsored by the Electrical Power Research Institute) is conducting a major study on the mechanisms related to fume generation and hopes to be able to identify methods by which fume generation may be reduced. Much of the previous work done in this field deals with dust and fume generation in the BOF. However, as oxygen use in the EAF continues to rise, analogous mechanisms can be expected. Some of the key fume generation mechanisms are expanded upon.

10.4.5.1 Entrainment of Particles

Entrainment of particles occurs as CaO, rust and dirt in the scrap are pulled out of the EAF in the offgas system. It has been shown that the amount of CaO carryover is highly related to the method by which it is fed to the EAF.

10.4.5.2 Volatilization Of Volatile Metals

Certain metals volatilize as the scrap heats up. These include Zn, Pb, Cd, Na, Mn and Fe. These metals tend to form oxides following volatilization and show up as such in the EAF dust. Obviously, one way of reducing the total amount of dust generated would be to ensure that these materials were not included in the scrap charge. However, the recycle of coated scrap is on the increase and as such this portion of the dust generation is likely to increase. The degree of volatilization is likely related to hot spots in the furnace such as those under the arc and those which occur in oxygen lancing and oxy-fuel burner operations.

10.4.5.3 CO Bubble Bursting

CO bubbles are generated during oxygen lancing and slag foaming operations. Some liquid steel is usually associated with these bubbles and when they burst, this steel is ejected into the furnace atmosphere where it forms very fine particles of iron oxide.

10.4.6 Future Environmental Concerns

With the increased focus on environmental concerns, the focus on meltshop fume control has shifted to include not only particulate capture but also to control the emission of toxic gases from the fume system. Several gases have been scrutinized over the past five years and in 1991, the EPA revised its list of hazardous emissions to address control of emissions of these gases. Some of those gases that are associated with electric furnace operations are CO/CO_2, NO_x, VOCs and dioxins. In addition, emission standards for particulate matter continue to become more stringent. These issues are discussed in detail in the following sections.

10.4.6.1 Comparison of Environmental Regulations

Over the past few years environmental regulations have been made much more stringent as the public has become more aware of the hazards posed by various industrial emissions. In Europe the regulations are even tougher than those proposed in North America. This is probably due to a greater population density that results in industrial facilities frequently being located in heavily populated areas. The European Community has recently adopted regulations that closely align to the German Federal Air Emission Directive (17th BImSch V) of 1990. Table 10.10 shows the German standards.

Table 10.10 German Federal Air Emission Directive, 17th BImSch V

Pollutant	Standard
Dust	10 mg/m^3
CO	
NO_x	200 mg/m^3
SO_x	50 mg/m^3
VOCs	
Dioxins	0.1 ng/m^3 total equivalent

It is likely that North American standards will closely match these in the future.

10.4.6.2 Dust Emissions

Typically, electric furnace operations generate approximately 9–18 kg (20–40 lbs) of dust for every ton of steel produced. The preferred method of capture appears to be via a fourth hole system feeding to a baghouse for capture of the dust. Most modern systems claim discharge levels of 5–50 mg/m^3 in the discharge gas. Of a greater concern in most meltshops is the level of fugitive dust emissions to the shop environment. This can be addressed by several methods such as increasing fourth hole extraction (though this can have a negative effect on productivity), increasing canopy hood extraction (this is the most expensive option), and providing a furnace enclosure.

Of the three options, the latter appears to be gaining popularity because it results in a low volume gas cleaning system without side effects on the process.

10.4.6.3 CO

CO and CO_2 result from several operations in the EAF. When oxygen is lanced into the furnace it reacts with carbon to produce CO. If substantial amounts of CO are not captured by the DES system, ambient levels in the work environment may not be acceptable. Typically up to 10% of the CO unburned in the furnace reports to the secondary fume capture system during meltdown.

CO and CO_2 are of concern due to the fact that they are greenhouse gases and contribute to global warming. As oxygen use has increased in the EAF, so have the quantities of CO and CO_2 emitted from furnace operations. CO generation has also increased considerably with the adoption of foamy slag practices. Much of the CO generated within the EAF does not burn to CO_2 until it is within the fume system. This makes the fume system requirements greater due to the increased heat load. In addition, the benefit of CO post-combustion as an energy source for steelmaking is not utilized.

If however, the CO is burned in the freeboard it is possible to recover heat within the furnace thus reducing the heat load that the offgas system must handle. Results are discussed in Section 10.7.5.1.

If greater energy efficiencies can be achieved in the furnace via post-combustion, the net effect is to decrease the generation of CO_2 considerably. The combustion of carbon to CO produces only one-third of the energy of the combustion of CO. Thus even if only 50% of the energy available from post-combustion of CO is transferred back to the bath, a reduction of oxygen lancing into the bath of 50% could be made. By utilizing the energy from the post-combustion of CO, it will not be necessary to lance as much carbon out of the bath. This will lead to reduced CO/CO_2 emissions as well as considerable cost savings.

Other factors which can reduce CO emissions have been identified and include:

1. Use a low velocity water-cooled elbow exit coupled with a high velocity water-cooled duct inlet.
2. Ensure proper combustion air gap setting.
3. Provide accurate and reliable furnace draft control.
4. Provide sufficient DES capacity.
5. Ensure that carbon for slag foaming is injected at the bath/slag interface.
6. Melt as fast as possible.
7. Apply post-combustion to burn the CO in the furnace.
8. Begin oxygen lancing as early as possible—this spreads out CO generation throughout the melting cycle and reduces the maximum heat load on the DES.

10.4.6.4 NO_x

NO_x is formed in furnace operations when nitrogen passes through the arc between electrodes. Some thermal NO_x is also generated from burner use in EAFs. Typical levels of NO_x reported are in the range of 36–90 g NO_x per ton of steel.[18] Thermal NO_x can be addressed using any of the conventional methods. Typically, thermal NO_x is reduced by improving the burner design and providing good mixing of the pre-combustion gases. NO_x resulting from nitrogen passing through the arc can be reduced by reducing the amount of nitrogen available in the furnace. This can be achieved by closing up furnace gaps and by closing the slag door whenever possible. Many operations lance oxygen through the door and as a result large a large volume of air enters the furnace. This can be reduced by providing a shield close to the door or by hanging chains close to the opening. Foamy slag practice can be beneficial since the slag foams up partially blocking the opening. Also, since foamy slag helps to bury the arc, it is more difficult for nitrogen to pass through the arc and be ionized.

Thermal NO_x is also formed in the water-cooled duct following the combustion air gap as any combustible materials burn with the oxygen which has entered the ductwork in the combustion air. If all of the combustible material burns quickly, the gas temperature will reach a level where thermal NO_x is formed. One option which is now being investigated is to close up the combustion gap somewhat and to supply combustion air at various points in the water-cooled duct downstream of the combustion gap through injectors. Thus the combustion will be staged along the first two-thirds of the water-cooled duct and will avoid the temperatures associated with thermal NO_x generation.

10.4.6.5 VOCs

Most scrap mixes contain organic compounds to some extent. When scrap is charged to the furnace, some of these organic compounds burn off contributing to the charging plume. In most cases organic materials continue to burn off for five to ten minutes following charging. If there is insufficient oxygen available for combustion, these hydrocarbons will report to the offgas system where they may or may not be combusted. The total emissions of hydrocarbons appears to be related to the amount of chlorine in the EAF.[20] Scrap preheating tends to produce greater emissions of hydrocarbons because these do not burn due to the gas temperature range and as a result report to the offgas system.

The best methods for reducing VOCs from EAF operations is to either destroy them or to limit the amount entering the operation. If burners are operated with excess oxygen during the flash off period it is possible to burn most of the organic compounds within the furnace. This is beneficial because some of the heat is transferred back to the scrap thus aiding melting operations. It is also beneficial in that the resulting heat load to the offgas is lower.

Some EAF operations try to limit the amount of oily scrap that is charged to the furnace. In some cases the scrap is washed to remove some of the oil and grease. This is particularly true in the case of turnings and borings. If the amount of organic material in the scrap is reduced the level of VOC emissions is also reduced.

Some processes employ scrap preheating followed by an afterburner to destroy any VOCs. This requires additional equipment and increases operating costs but ensures that emissions of VOCs are kept to a minimum.

10.4.6.6 Dioxins

Dioxins and furans have become a major concern over the past few years in the incineration process. Dioxins and furans are combustion byproducts and the prevention of these emissions depends strongly on control of the combustion process. The term dioxins generally refers to 17 specific dioxin and furan congeners that are extremely toxic, the most toxic of which is tetrachloro–dibenzo–p–dioxin, see Fig. 10.58. These compounds are collectively referred to as dioxins and regulations are specified as a tetrachloro–dibenzo–p–dioxin (TCDD) toxic equivalent (TEQ). It is only recently that dioxin emissions in steelmaking operations have been monitored in Europe.

Dioxins are cyclic hydrocarbons that are usually formed in combustion processes. Dioxins do not have any known technical use and are not produced intentionally. They are formed as trace amounts in some chemical processes and during combustion of fuels, especially materials containing chlorinated materials. In incineration operations, dioxins are believed to form at a temperature of approximately 500°C (932°F). Dioxins tend to be formed downstream of incineration operations as the flue gases are cooled. This usually occurs in the temperature range of 250–400°C (482°F–752°F) by reactions between oxygen, water, hydrochloric gas and products of incomplete combustion.

In studies related to the formation of dioxins it was found that several factors had an affect. High carbon and high chlorine concentrations favor dioxin formation. Carbon acts as a sorbent for precursor compounds for dioxin formation. Water vapor helps to inhibit chlorine and thus reduces dioxin formation. Copper acts as a strong catalyst for dioxin formation. Al_2O_3 increases the effectiveness of copper to catalyze the reaction. Fe, Zn, Mn and Cu along with their oxides help to catalyze dioxin formation. Injection of ammonia into the offgas stream helps inhibit dioxin formation. Activated carbon can be used to strip dioxins from the gas stream.

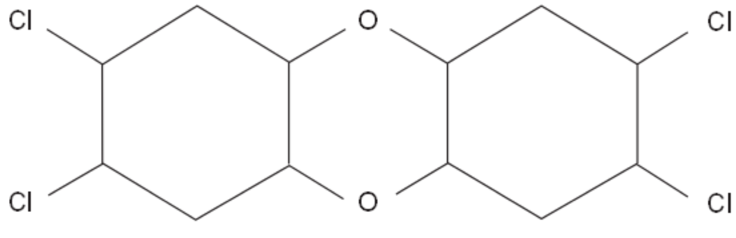


Fig. 10.58 Tetrachloro–dibenzo–p–dioxin structure.

From the above information it is apparent that the components of the offgas stream in the EAF are ideal for the formation of dioxins. In material published by MEFOS it was shown that the biggest contributor to dioxins in scrap steel melting was PVC coated scrap followed by oily scrap.[20] It was also found that the baghouse emissions of dioxins ranged from 20–60% of the offgas stream concentration. This indicates that dioxins were being captured in the baghouse. This is supported by research that indicates that 20–30% of dioxins are bound to particulate.[21] The Swedish paper concludes that the following factors can reduce dioxin emissions: scrap sorting, scrap pretreatment, post-combustion of organics in the furnace, and absorbent injection in the offgas.[20]

Tests conducted at several European steelmakers have indicated that dioxin formation can be controlled by rapid quenching of the offgas through the temperature zone in which dioxin formation usually takes place. In these operations offgas is cooled rapidly from approximately 800°C (1470°F) to less than 300°C (570°F) using spray water. This has been shown to limit the dioxin emissions to 0.1–0.2 ng/Nm^3 TEQ. For most DES systems this would involve spray cooling of the offgas following the water-cooled duct portion of the system. This should not pose problems for system operation as long as the amount of spray cooling does not create dewpoint problems in the baghouse.

10.4.7 Conclusions

Electric arc furnace fume systems have evolved considerably from very simple systems aimed at improving the ambient work environment around the furnace to sophisticated systems aimed at controlling not only the emission of particulate but also the control of toxic gases. Modern fume systems are now designed to minimize the formation of toxic gases and to ensure that others are destroyed before exiting the system. Thus gas cleaning in the modern day fume system entails much more than trapping and collection of particulate matter. With the tighter environmental restrictions expected in the future, it is expected that electric furnace operations will have to look at environmental concerns in conjunction with furnace operations. A variety of factors will affect the costs and processes necessary to maintain production and meet environmental restrictions. Without doubt the cheapest method of meeting emissions restrictions is to prevent these compounds from being present in the first place. Capture and treatment of pollutants is an expensive way of dealing with the problem. Of course, tighter restrictions of the feed materials to the furnace will also have an affect on productivity and operating costs. It will become necessary to weigh environmental considerations with operations concerns in order to find an optimum that allows for low cost steel production while meeting emissions requirements. This means that we must integrate environmental thinking into operations in order to arrive at a low cost, environmentally friendly steel production.

10.5 Raw Materials

The main raw material for EAF steelmaking is steel scrap. Scrap is an energy intensive and valuable commodity and comes primarily from three main sources: reclaimed scrap (also known as obsolete scrap) which is obtained from old cars, demolished buildings, discarded machinery and domestic objects; industrial scrap (also known as prompt scrap) which is generated by industries using steel within their manufacturing processes; and revert scrap (also known as home scrap) which is generated within the steelmaking and forming processes (e.g. crop ends from rolling operations, metallic losses in slag etc.).

The latter two forms of scrap tend to be clean, i.e. they are close in chemical composition to the desired molten steel composition and thus are ideal for recycle. Reclaimed/obsolete scrap frequently has a quite variable composition and quite often contains contaminants that are undesirable for steelmaking. Levels of residual elements such as Cu, Sn, Ni, Cr, and Mo are high in obsolete scrap and can affect casting operations and product quality if they are not diluted. Thus a facility which has a need for very low residual levels in the steel will be forced to use higher quality prompt scrap but at a much higher cost. The alternative is to use a combination of the contaminated obsolete scrap along with what are generally referred to as clean iron units or virgin iron units. These are materials which contain little or no residual elements. Clean iron units are typically in the form of direct reduced iron (DRI), hot briquetted iron (HBI), iron carbide, pig iron, and molten pig iron (hot metal).

It is possible to use lower grade scrap which contains residual elements, if this scrap is blended with clean iron units so that the resulting residual levels in the steel following melting meet the requirements for flat rolled products.

Obsolete scrap is much more readily available than prompt scrap and thus the use of clean iron units is expected to increase as shortages of prompt scrap continue to grow.

In addition to classification of scrap into the above three groups, scrap is also classified based on its physical size, its source and the way in which it is prepared. For example the following categories are those commonly used in North America: No. 1 bundles, No. 1 factory bundles, No. 1 shredded, No. 1 heavy melt, No. 2 heavy melt, No. 2 bundles, No. 2 shredded, busheling, turnings, shredded auto, structural/plate 3 ft., structural/plate 5 ft., rail crops, and rail wheels.

In addition to the residual elements contained in the scrap, there are also several other undesirable components including, oil, grease, paint coatings, zinc coatings, water, oxidized material and dirt. The lower the grade of scrap, the more likely it is to contain greater quantities of these materials. As a result this scrap may sell at a discount but the yield of liquid steel may be considerably lower than that obtained when using a higher grade scrap. In addition, these undesirable components may result in higher energy requirements and environmental problems. Thus the decision for scrap mix to be used within a particular operation will frequently depend on several factors including availability, scrap cost, melting cost, yield, and the effect on operations (based on scrap density, oil and grease content, etc.).

In practice, most operations buy several different types of scrap and blend them to yield the most desirable effects for EAF operations.

10.6 Fluxes and Additives

Carbon is essential to the manufacture of steel. Carbon is one of the key elements which give various steel grades their properties. Carbon is also important in steelmaking refining operations and can contribute a sizable quantity of the energy required in steelmaking operations. In BOF steelmaking, carbon is present in the hot metal that is produced in the blast furnace. In electric furnace steelmaking, some carbon will be contained in the scrap feed, in DRI, HBI or other alternative iron furnace feeds. The amount of carbon contained in these EAF feeds will generally be considerably lower than that contained in hot metal and typically, some additional carbon is charged to the EAF. In the past carbon was charged to the furnace to ensure that the melt-in carbon level was above that desired in the final product. As higher oxygen utilization has developed as standard EAF practice, more carbon is required in EAF operations. The reaction of carbon with oxygen within the bath to produce carbon monoxide results in a significant energy input to the process and has lead to substantial reductions in electrical power consumption in EAF operations. The generation of CO within the bath is also key to achieving low concentrations of dissolved gases (nitrogen and hydrogen) in the steel as these are flushed out with the carbon monoxide. In addition, oxide inclusions are flushed from the steel into the slag.

In oxygen injection operations, some iron is oxidized and reports to the slag. Oxy-fuel burner operations will also result in some scrap oxidation and this too will report to the slag once the scrap melts in. Dissolved carbon in the steel will react with FeO at the slag/bath interface to produce CO and recover iron units to the bath.

The amount of charge carbon used will be dependent on several factors including carbon content of scrap feed, projected oxygen consumption, desired tap carbon, and the economics of iron yield versus carbon cost. In general, the amount used will correspond to a carbon/oxygen balance as the steelmaker will try to maximize the iron yield. Typical charge carbon rates for medium carbon steel production lie in the range of 2–12 kg per ton of liquid steel.

Generally, the three types of carbonaceous material used as charge carbon in EAF operations are anthracite coal, metallurgical coke and green petroleum coke. Most anthracite coal used in North American steelmaking operations is mined in eastern Pennsylvania. This material has a general composition of 3–8% moisture content, 11–18% ash content and 0.4–0.7% sulfur.

The high variation in ash content translates into wide variations in fixed carbon content and in general, EAF operations strive to keep ash content to a minimum. The ash consists primarily of silica. Thus increased ash input to the EAF will require additional lime addition in order to maintain the desired V-ratio in the slag. The best grades of anthracite coal have fixed carbon contents of 87–89%. Low grade anthracite coals may have fixed carbon levels as low as 50%.

Anthracite coal is available in a wide variety of sizes ranging from 4 × 8 in. down to 3/64 in. × 100 mesh. The most popular sizes for use as charge carbon are nut (1 5/8 × 13/16 in.), pea (13/16 × 9/16 in.), and buckwheat (9/16 × 5/16 in.).

Metallurgical coke is produced primarily in integrated steel operations and is used in the blast furnace. However, some coke is used as EAF charge carbon. Generally, this material has a composition of 1–2% moisture content, 1–3.5% volatile material, 86–88% fixed carbon, 9–12% ash content and 0.88–1.2% sulfur.

Usually coke breeze with a size of –1/2 × 0 is used as charge carbon. Coarser material can be used but is more expensive.

Green petroleum coke is a byproduct of crude oil processing. Its properties and composition vary considerably and are dependent on the crude oil feedstock from which it is derived. Several coking processes are used in commercial operation and these will produce considerably different types of coke.

Sponge coke results from delayed coker operations and is porous in nature. It may be used as a fuel or may be processed into electrodes or anodes depending on sulfur content and impurity levels. This material is sometimes available as charge carbon.

Needle coke is produced using a special application of the delayed coker process. It is made from high grade feedstocks and is the prime ingredient for the production of carbon and graphite electrodes. This material is generally too expensive to be used as charge carbon.

Shot coke is a hard, pebble-like material resulting from delayed coker operation under conditions which minimize coke byproduct generation. It is generally used as a fuel and is cost competitive as charge carbon.

Fluid coke is produced in a fluid coker by spraying the residue onto hot coke particles. It is usually high in sulfur content and is used in anode baking furnaces. It can also be used as a recarburizer if it is calcined.

Over the past decade, many operations have adopted foamy slag practices. At the start of meltdown the radiation from the arc to the sidewalls is negligible while the electrodes are surrounded by the scrap. As melting proceeds the efficiency of heat transfer to the scrap and bath drops off and more heat is radiated from the arc to the sidewalls. By covering the arc in a layer of slag, the arc is shielded and the energy is transferred to the bath. Oxygen is injected with coal to foam the slag by producing CO gas in the slag. In some cases only carbon is injected and the carbon reacts with FeO in the slag to produce CO gas. When foamed, the slag cover increases from 10 to 30 cm thick.[22] In some cases the slag is foamed to such an extent that it comes out of the electrode ports. Claims for the increase in energy transfer efficiency range from an efficiency of 60–90% with slag foaming compared to 40% without.[23,24] It has been reported that at least 0.3% carbon should be removed from the bath using oxygen in order to achieve a good foamy slag practice.[25]

The effectiveness of slag foaming is dependent on several process parameters as described in the section on furnace technologies. Typical carbon injection rates for slag foaming are 2–5 kg per ton of liquid steel for low to medium powered furnaces. Higher powered furnaces and DC furnaces will tend to use 5–10 kg of carbon per ton liquid steel. This is due to the fact that the arc length is much greater than low powered AC operations and therefore greater slag cover is required to bury the arc.

Lime is the most common flux used in modern EAF operations. Most operations now use basic refractories and as a result, the steelmaker must maintain a basic slag in the furnace in order to minimize refractory consumption. Slag basicity has also been shown to have a major effect on slag foaming capabilities. Thus lime tends to be added both in the charge and also via injection directly into the furnace. Lime addition practices can vary greatly due to variances in scrap composition. As elements in the bath are oxidized (e.g. P, Al, Si, Mn) they contribute acidic components to the slag. Thus basic slag components must be added to offset these acidic contributions. If silica levels in the slag are allowed to get too high, significant refractory erosion will result. In addition, FeO levels in the slag will increase because FeO has greater solubility in higher silica slags. This can lead to higher yield losses in the EAF.

The generation of slag also allows these materials which have been stripped from the bath to be removed from the steel by pouring slag out of the furnace through the slag door which is located at the back of the EAF. This is known as slagging off. If the slag is not removed but is instead allowed to carry over to the ladle it is possible for slag reversion to take place. This occurs when metallic oxides are reduced out of the slag by a more reactive metallic present in the steel. When steel is tapped it is frequently killed by adding either silicon of aluminum during tapping. The purpose of these additions is to lower the oxygen content in the steel. If however P_2O_5 is carried over into the ladle, it is possible that it will react with the alloy additions producing silica or alumina and phosphorus which will go back into solution in the steel.

Sometimes magnesium lime is added to the furnace either purely as MgO or as a mixture of MgO and CaO. Basic refractories are predominantly MgO, thus by adding a small amount of MgO to the furnace, the slag can quickly become saturated with MgO and thus less refractory erosion is likely to take place.

10.7 Electric Furnace Technology

The electric arc furnace has evolved considerably over the past 20 years. Gone are the days when electric power was the only source of energy for scrap melting. Previously tap-to-tap times in the range of 3–8 hours were common.[26] With advances in technology it is now possible to make heats in under one hour with electrical energy consumptions in the range of 380–400 kWh/ton.[23] The electric furnace has evolved into a fast and low cost melter of scrap where the major criterion is higher productivity in order to reduce fixed costs. Innovations which helped to achieve the higher production rates include oxy-fuel burners, oxygen lancing, carbon/lime injection, foamy slag practices, post-combustion in the EAF freeboard, EAF bath stirring, modified electrical supply (series reactors etc.), current conducting electrode arms, DC furnace technology and other innovative process technologies (scrap preheat, continuous charging etc.).

10.7.1 Oxygen Use in the EAF

Much of the productivity gain achieved over the past 10–15 years was related to oxygen use in the furnace. Exothermic reactions were used to replace a substantial portion of the energy input in the EAF. Whereas oxygen utilization of 9 Nm^3/ton (300 scf/ton) was considered ordinary just 10 years ago, some operations now use as much as 40 Nm^3/ton (1300 scf/ton) for lancing operations.[27,28] With post-combustion, rates as high as 70 Nm^3/ton (2500 scf/ton) have been implemented. It is now common for 30–40% of the total energy input to the EAF to come from oxy-fuel burners and oxygen lancing.[23,28] By the early 1980s, more than 80% of the EAFs in Japan employed oxy-fuel burners.[23] In North America it was estimated that in 1990, only 24% of EAF operations were using such burners.[29] Since that time, a large percentage of North American operations have looked to the use of increased oxygen levels in their furnaces in order to increase productivity and decrease electrical energy consumption. High levels of oxygen input are standard on most new EAF installations.

The IISI 1990 electric arc furnace report indicates that most advanced EAF operations utilize at least 22 Nm^3/ton (770 scf/ton) of oxygen.[18] In addition oxygen supplies 20–32% of the total power input in conventional furnace operations. This has grown with the use of alternative iron sources in the EAF, many of which contain elevated carbon contents (1–3%). In some cases, electrical energy now accounts for less than 50% of the total power input for steelmaking.

One of the best examples of the progressive increase in oxygen use within the EAF is the meltshop operation at Badische Stahlwerke (BSW). Between 1978 and 1990, oxygen use was increased from 9 Nm3/ton to almost 27 Nm^3/ton. Productivity increased from 32 ton per hour to 85 ton per hour while power consumption decreased from 494 kWh/ton to 357 kWh/ton.[30] During this period, BSW developed their own manipulator for the automatic injection of oxygen and carbon. In 1993, BSW installed the ALARC post-combustion system developed by Air Liquide and increased oxygen consumption in the furnace to 41.5 Nm^3/ton. The corresponding power consumption is 315 kWh/ton with a tap-to-tap time of 48 minutes. This operation has truly been one of the pioneers for increased chemical energy use in the EAF.

10.7.2 Oxy-Fuel Burner Application in the EAF

Oxy-fuel burners are now almost standard equipment on electric arc furnaces in many parts of the world. The first use of burners was for melting the scrap at the slag door where arc heating was fairly inefficient. As furnace power was increased, burners were installed to help melt at the cold spots common to UHP operation. This resulted in more uniform melting and decreased the melting time necessary to reach a flat bath. It was quickly realized that productivity increases could be achieved by installing more burner power. Typical productivity increases reported in the literature have been in the range of 5–20%.[22,31–33] In recent years oxy-fuel burners have been of greater interest due to the increase in the cost of electrodes and electricity. Thus natural gas potentially provides a cheaper source of energy for melting. Fig. 10.59 shows oxy-fuel burners in operation in an EAF.

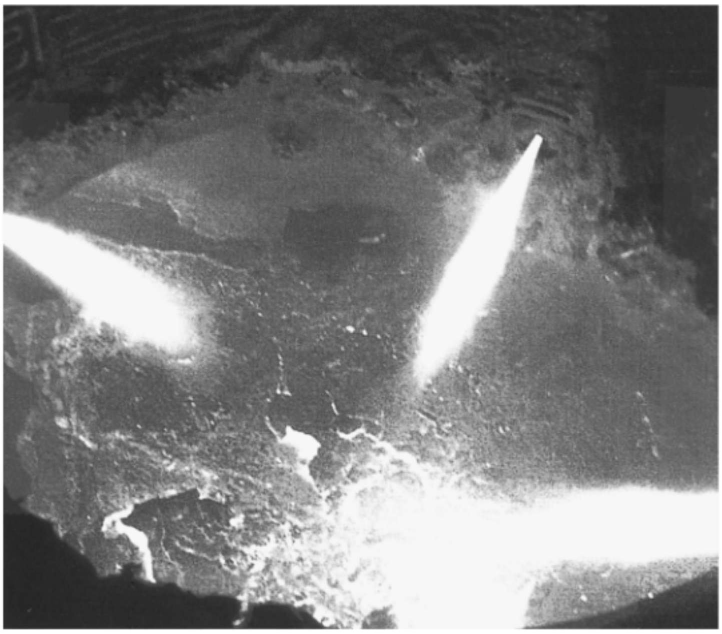


Fig. 10.59 Oxy-fuel burners in EAF operations.

Typically burners are located in either the slag door, sidewall or roof. Slag door burners are generally used for small to medium sized furnaces where a single burner can reach all of the cold spots.[22,33] Door burners have the advantage that they can be removed when not in use. For larger furnaces, three or four sidewall mounted burners are more effective for cold spot penetration. However these are vulnerable to attack from slag, especially if employing a foamy slag practice. In such cases the burners are sometimes mounted in the roof and are fired tangentially through the furnace cold spots.

Oxy-fuel burners aid in scrap melting by transferring heat to the scrap. This heat transfer takes place via three modes; either forced convection from the combustion products, radiation from the combustion products or conduction from carbon or metal oxidation and from scrap to other scrap.[33]

Primarily heat transfer is via the first two modes except when the burners are operated with excess oxygen. Heat transfer by these two modes is highly dependent on the temperature difference between the scrap and the flame and on the surface area of the scrap exposed for heat transfer. As a result oxy-fuel burners are most efficient at the start of a melt-in period when the scrap is cold. As melting proceeds the efficiency will drop off as the scrap surface in contact with the flame decreases and due to the fact that the scrap temperature also increases. It is generally recommended that burners be discontinued after 50% of the meltdown period is completed so that reasonable efficiencies are achieved.[23] An added complication is that once the scrap heats up it is possible for iron to react with the water formed by combustion to produce iron oxide and hydrogen. This results in yield loss and the hydrogen must be combusted downstream in the offgas system. Usually the point at which burner use should be discontinued is marked by a rise in offgas temperature(indicating that more heat is being retained in the offgas).[33] In some operations the temperature of the furnace side panels adjacent to the burner is used to track burner efficiency. Once the efficiency drops below a set point the burners are shut off.[29]

Heat transfer by conduction occurs when excess oxygen reacts with material in the charge. This will result in lower yield if the material burned is iron or alloys and as a result is not generally recommended. However in the period immediately following charging when volatile and combustible materials in the scrap flash off, additional oxygen in the furnace is beneficial as it allows this material to be burned inside the furnace and thus results in heat transfer back to the scrap. This is also beneficial for the operation of the offgas system as it not required to remove this heat downstream.

Fig. 10.60 indicates that 0.133 MW of burner rating should be supplied per ton of furnace capacity.[34] Other references recommend a minimum of 30 kWh/ton of burner power to eliminate cold spots in a UHP furnace and 55–90 kWh/ton of burner power for low powered furnaces.[23]

A burner/lance has been developed by Empco, Fig. 10.15, that has the capability of following the scrap level as it melts. The lance is similar to a BOF design (water-cooled) and is capable of operation at various oxygen to natural gas ratios or with oxygen alone for decarburization. The lance is banana shaped and follows the scrap as its melts back. Thus a high heat transfer efficiency can be maintained for a longer period of time. Typical efficiencies reported for the Unilance are in the range of 65–70%.[35]

Heat transfer efficiencies reported in the literature vary greatly in the range of 50–75%.[24] Data published by L'Air Liquide shows efficiencies of 64–80% based on energy savings. Krupp indicates an energy efficiency based on energy savings alone of 78%,[36] and also gives an indication of why such high efficiencies are reported when looking at energy replacement alone. According to Krupp, an increase of one minute in meltdown time corresponds to an increase in power consumption of 2 kWh/ton. Thus the decrease in tap-to-tap time must be taken into account when calculating burner efficiency. For typically reported energy efficiencies, once this is taken into account burner efficiencies lie in the range of 45–65% in all cases. This is supported somewhat by theoretical calculations by Danieli that indicated that only 20–30% of the energy from the burners went to the scrap;[37] as much as 40–60% of the heat went to the offgas and the remainder was lost to the water-cooled furnace panels.

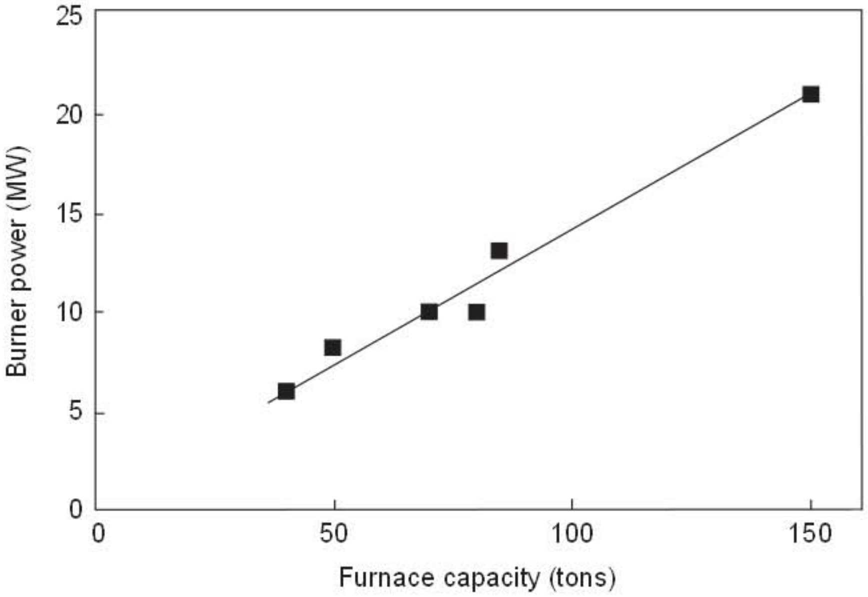


Fig. 10.60 Oxy-fuel burner rating versus furnace capacity.

Fig. 10.61 shows burner efficiency as a function of operating time based on actual furnace offgas measurements.[34] This shows that burner efficiency drops off rapidly after 40–50% of the melting time. By 60% into the melting time burner efficiency has dropped off to below 30%. It is apparent that a cumulative efficiency of 50–60% is achieved over the first half of the meltdown period and drops off rapidly afterwards. As a result, typical operating practice for a three bucket charge is to run the burners for two-thirds of the first meltdown, one-half of the second meltdown and one-third of the third meltdown. For operations with only one backcharge, burners are typically run for 50% of each meltdown phase.[32]

The amount of power input to the furnace has a small effect on the increase in heatload and offgas volume. The major factor is the rate at which the power is put into the furnace. Thus for a low pow-

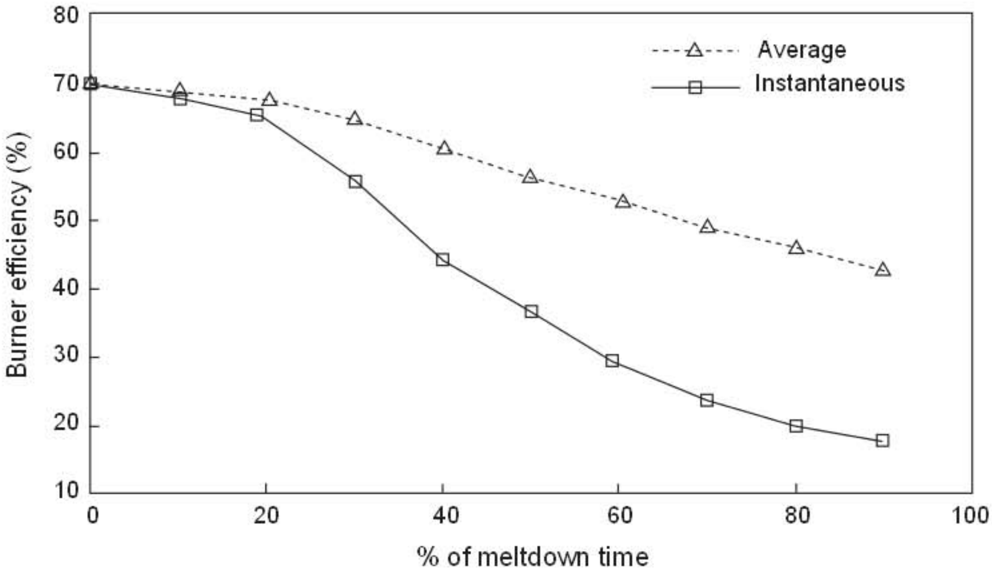


Fig. 10.61 Oxy-fuel burner efficiency versus time into meltdown.

ered furnace(high tap-to-tap time), the total burner input to the EAF may be in the range of 80–100 kWh/ton but the net effect on the offgas evacuation requirements may not change much as the burners are run for a longer period of time. Likewise for a high powered furnace, due to the short tap-to-tap time, the burner input rate may be quite high even though the burners supply only 30 kWh/Ton to the furnace.

Bender estimates that offgas flowrates can increase by a factor of 1.5 and offgas heatload by as high as a factor of 2.5 for operation with burners and high oxygen lance rates.[37]

10.7.3 Application of Oxygen Lancing in the EAF

Over the past ten years oxygen lancing has become an integral part of EAF melting operations. It has been recognized in the past that productivity improvements in the open hearth furnace and in the BOF were possible through the use of oxygen to supply fuel for exothermic reactions. Whereas previously oxygen was used primarily only for decarburization in the EAF at levels of 3–8 Nm^3 per ton (96–250 scf/ton),[23,32] in modern operations anywhere from 10–30% of the total energy input is supplied via exothermic bath reactions.[23,24,28,38,29] For a typical UHP furnace oxygen consumption is in the range of 18–27 Nm^3/ton (640–960 scf/ton).[28] Oxygen utilization in the EAF is much higher in Japan and in Europe where electricity costs are higher. Oxygen injection can provide a substantial power input—a lance rate of 30 Nm^3/min. (1050 scfm)is equivalent to a power input of 11 MW based on the theoretical reaction heat from combustion of C to CO.[23]

Oxygen lances can be of two forms. Water-cooled lances are generally used for decarburization though in some cases they are now use for scrap cutting as well. The conventional water-cooled lance was mounted on a platform and penetrated into the side of the furnace through a panel. Water-cooled lances do not actually penetrate the bath though they sometimes penetrate into the slag layer. Consumable lances are designed to penetrate into the bath or the slag layer. They consist of consumable pipe which is adjusted as it burns away to give sufficient working length. The first consumable lances were operated manually through the slag door. Badische Stahl Engineering developed a robotic manipulator to automate the process. This manipulator is used to control two lances automatically.[22] Various other manipulators have been developed recently and now have the capability to inject carbon and lime for slag foaming simultaneously with oxygen lancing. One major disadvantage of lancing through the slag door is that it can increase air infiltration into the furnace by 100–200%. This not only has a negative impact on furnace productivity but also increases offgas system evacuation requirements substantially. As a result, not all of the fume is captured and a significant amount escapes from the furnace to the shop. This can be a significant problem if substantial quantities of CO escape to the shop due to its rapid cooling and subsequent incomplete combustion to CO_2. Thus background levels of CO in the work environment may become an issue. To reduce the amount of air infiltration to the EAF, some operations insert the lance through the furnace sidewall as shown in Fig. 10.62.

Energy savings due to oxygen lancing arise from both exothermic reactions (oxidation of carbon and iron) and due to stirring of the bath with leads to temperature and composition homogeneity of the bath. The product of scrap cutting is liquid iron and iron oxide. Thus most of the heat is retained in the bath. The theoretical energy input for oxygen reactions in the bath is as follows[23]:

$$Fe + 1/2\ O_2 \rightarrow FeO, \quad \text{heat input} = 6.0\ kWh/Nm^3\ O_2 \qquad (10.7.1)$$

$$C + 1/2\ O_2 \rightarrow CO, \quad \text{heat input} = 2.8\ kWh/Nm^3\ O_2 \qquad (10.7.2)$$

Thus it is apparent that much more energy is available if iron is burned to produce FeO. Naturally though, this will impact negatively on productivity. Studies have shown that the optimum use of oxygen for conventional lancing operations is in the range of 30–40 Nm^3/ton (1000–1250 scf/ton).[27] Above this level yield losses are excessive and it is no longer economical to add oxygen. Typical operating results have given energy replacement values for oxygen in the range of 2–4 $kWh/Nm^3\ O_2$ (0.056–0.125 kWh/scf O_2), with an average of 3.5 $kWh/Nm^3\ O_2$ (0.1 kWh/scf O_2).[24,27,28,31,32,32,40,41] These values show that it is likely that both carbon and iron are reacting. In

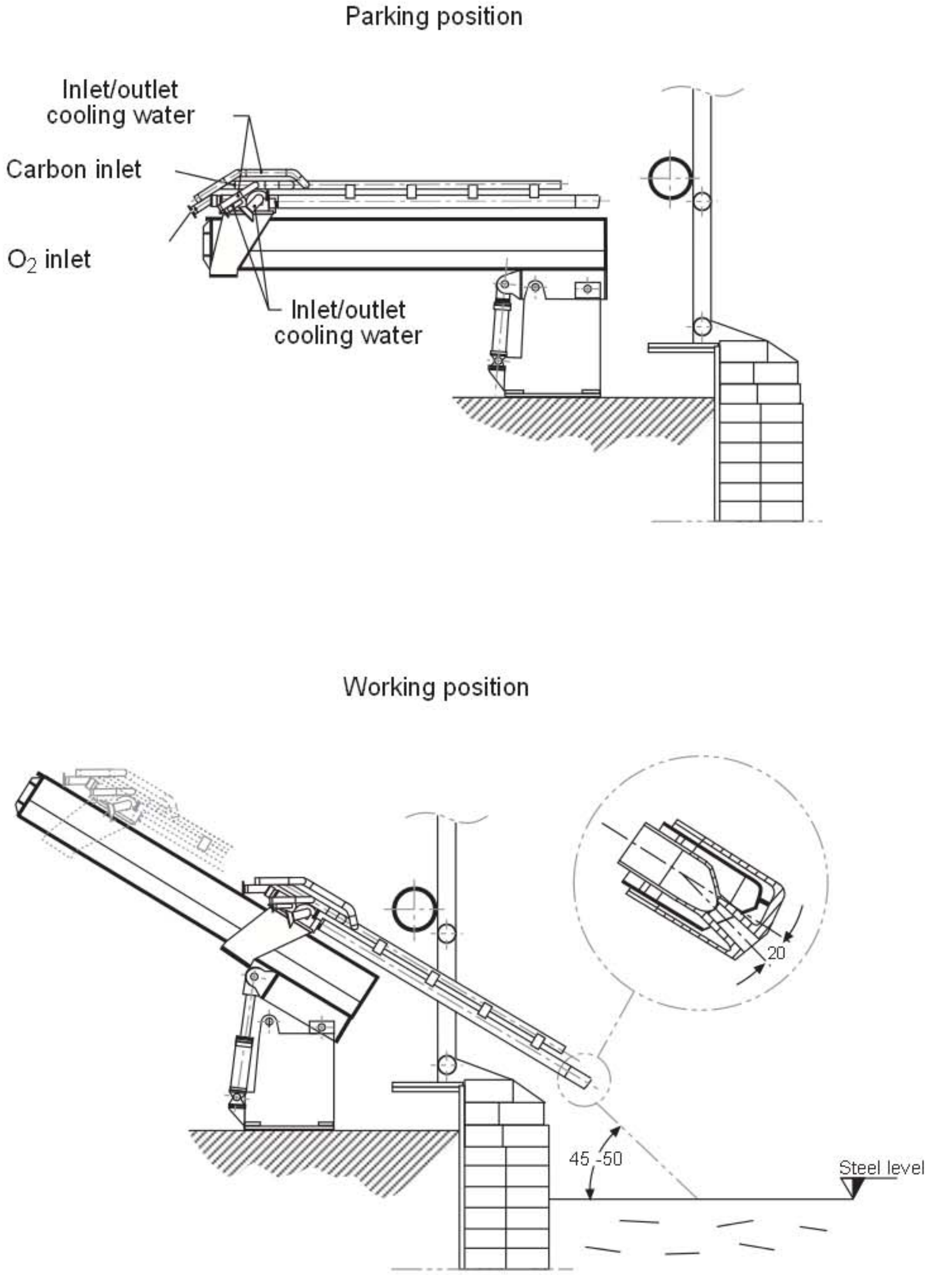


Fig. 10.62 Operation of oxygen lance through the furnace sidewall. *(Courtesy of Danieli.)*

addition, some studies have shown that the oxygen yield (i.e. the amount reacting with carbon) is in the range of 70–80%.[34,41] This would support the theory that both carbon and iron are reacting. During scrap cutting operations, the oxygen reacts primarily with the iron. Later when a molten pool has formed the FeO is reduced out of the slag by carbon in the bath. Thus the net effect is to produce CO gas from the oxygen that is lanced.

Based on the information cited in the preceding section, it can be expected that for every Nm^3 of oxygen lanced, 0.75 Nm^3 will react with carbon to produce 1.5 Nm^3 of CO (based on the average energy replacement value of 3.5 kWh/ Nm^3 O_2). If in addition the stirring effect of the lancing brings bath carbon or injected carbon into contact with FeO in the slag, an even greater quantity of CO may result. That this occurs is supported by data that indicates a decarburization efficiency of greater than 100%. Thus during the decarburization period up to 2.5 Nm^3 of CO may result for every Nm^3 of oxygen injected. Typical oxygen rates are in the range of 30–100 Nm^3/min. (1000–3500 scfm) and are usually limited by the ability of the fourth hole system to evacuate the furnace fume. Recommended lance rates for various furnace sizes are shown in Fig. 10.63 and indicate a rate of approximately 0.78–0.85 Nm^3/ton (25–30 scfm/ton) of furnace capacity. In some newer processes where feed materials are very high in carbon content, oxygen lance rates equivalent to 0.1% decarburization per minute are required. In such cases, the lance rates may be as high as 280 Nm^3/min (10,000 scfm) which is similar to BOF lance rates.

The major drawback to high oxygen lance rates is the effect on fume system control and the production of NO_x. Offgas volumes are greatly increased and the amount of CO generated is much greater. This must be taken into account when contemplating increased oxygen use.

The use of oxygen lancing throughout the heat can be achieved in operations using a hot heel in the furnace. Oxygen is lanced at a lower rate throughout the heat to foam the slag. This gives better shielding of the arc leading to better electrical efficiency. It also gives lower peak flowrates of CO to the offgas system, thus it reduces the extraction requirement of the offgas system.

High generation rates of CO may necessitate a post-combustion chamber in the DES system. If substantial amounts of CO are not captured by the DES system, ambient levels in the work environment may not be acceptable. Typically up to 10% of the CO unburned in the furnace reports to the secondary fume capture system during meltdown.

Operating with the slag door open increases the overall offgas evacuation requirements substantially. If possible oxygen lances should penetrate the furnace higher up in the shell. Another factor to consider is that the increased amount of nitrogen in the furnace will likely lead to increased NO_x.

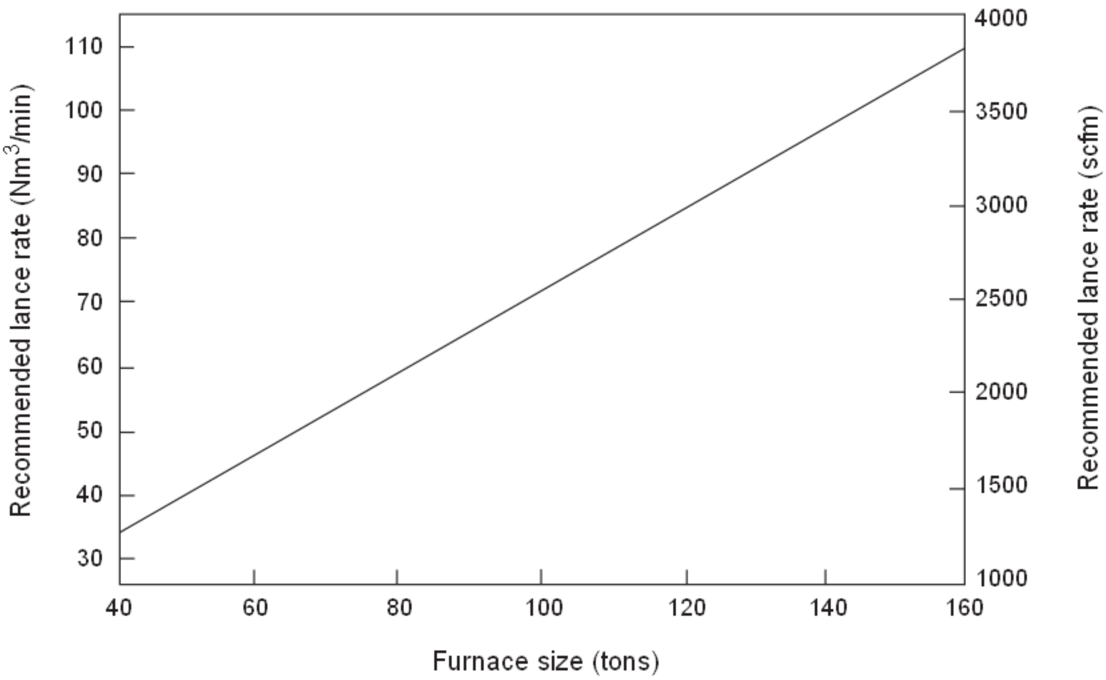


Fig. 10.63 Recommended lance rate versus furnace size.

10.7.4 Foamy Slag Practice

In recent years more EAF operations have begun to use a foamy slag practice. Foamy slag was initially associated with DRI melting operations where FeO and carbon from the DRI would react in the bath to produce CO which would foam the slag. At the start of meltdown the radiation from the arc to the sidewalls is negligible because the electrodes are surrounded by the scrap. As melting proceeds the efficiency of heat transfer to the scrap and bath drops off and more heat is radiated from the arc to the sidewalls. By covering the arc in a layer of slag, the arc is shielded and the energy is transferred to the bath as shown in Fig. 10.64.

Oxygen is injected with coal to foam up the slag by producing CO gas in the slag. In some cases only carbon is injected and the carbon reacts with FeO in the slag to produce CO gas. When foamed, the slag cover increases from four inches thick to twelve inches.[22] In some cases the slag is foamed to such an extent that it comes out of the electrode ports.[42] Claims for the increase in efficiency range from an efficiency of 60–90% with slag foaming compared to 40% without.[23,24] This is shown in Fig. 10.65 and Fig. 10.66. It has been reported that at least 0.3% carbon should be removed using oxygen in order to achieve a good foamy slag practice.[25] If a deep foamy slag is achieved it is possible to increase the arc voltage considerably. This allows a greater rate of power input. Slag foaming is usually carried out once a flat bath is achieved. However, with hot heel operations it is possible to start slag foaming much sooner.

Some of the benefits attributed to foamy slag are decreased heat losses to the sidewalls, improved heat transfer from the arcs to the steel allows for higher rate of power input, reduced power and voltage fluctuations, reduced electrical and audible noise, increased arc length (up to 100%) without increasing heat loss and reduced electrode and refractory consumption.[4,18,24]

Several factors have been identified that promote slag foaming. These are oxygen and carbon availability, increased slag viscosity, decreased surface tension, slag basicity > 2.5 and FeO in slag at 15–20% to sustain the reaction.[4,25,43] Fig. 10.67 and Fig. 10.68 show several of these effects graphically.

The only negative side of foamy slag practice is that a large quantity of CO is produced in the EAF. Bender estimates that offgas flowrates can increase by a factor of 1.5 and offgas heatload by as much as a factor of 2.5 for high slag foaming rates.[37] In many operations, a large amount of carbon is removed from the bath in order to generate chemical energy input for the operation. If this CO is to be generated anyway, the operator is well advised to ensure that slag conditions are such that foaming will result in order to benefit from the CO generation.

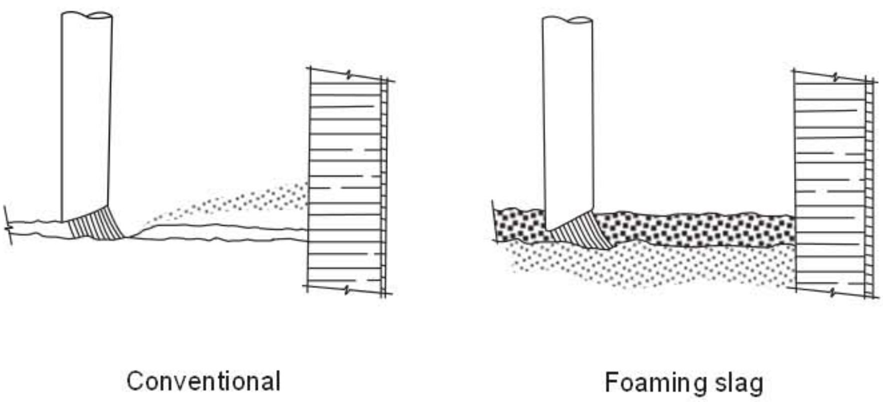
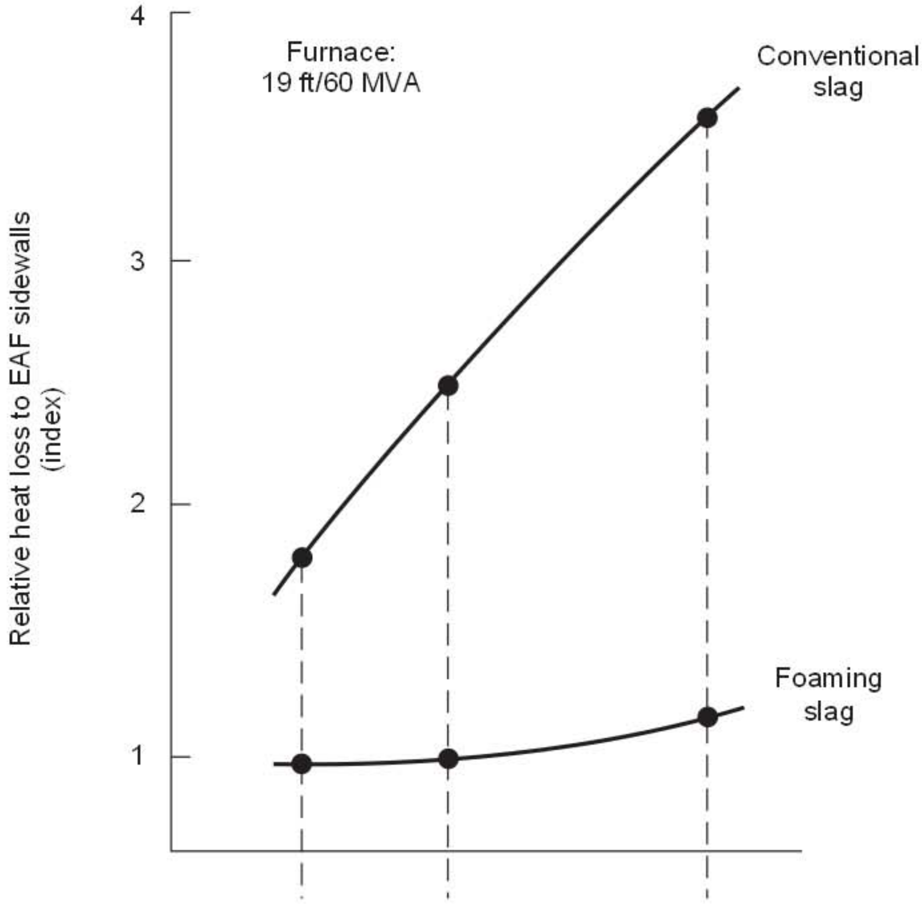


Fig. 10.64 Effect of slag foaming on arc radiation. *(Courtesy of Center for Materials Production.)*



Tap selected (volts)	535	585	685
Power factor (%)	63	76	88
Arc length (in.)	4.4	6.9	10.9
Arc current (kA)	65	59	51
Power (MW)	38	45	53

Fig. 10.65 Effect of slag foaming on heat loss to the furnace sidewalls. *(Courtesy of Center for Materials Production.)*

10.7.5 CO Post-Combustion

The 1990s have seen steelmakers advance further in lowering production costs of liquid steel. Higher electrical input rates and increased oxygen and natural gas consumption has led to short tap-to-tap times and high throughputs. Thus, energy losses are minimized and up to 60% of the total power input ends up in the steel. This has not come without cost, as water-cooled panels and roofs are required to operate at higher heat fluxes. Typically 8–10% of the power input is lost to the cooling water and offgas temperatures are extremely high, with losses of approximately 20% of the power input to the offgas. As EAF steelmakers attempt to lower their energy inputs further they have begun to consider the heat contained in the offgas. One way in which this can be recaptured is to use the offgas to preheat scrap. This results in recovery of the sensible heat but does not address the calorific heat which can represent as much as 50–60% of the energy in the offgas.

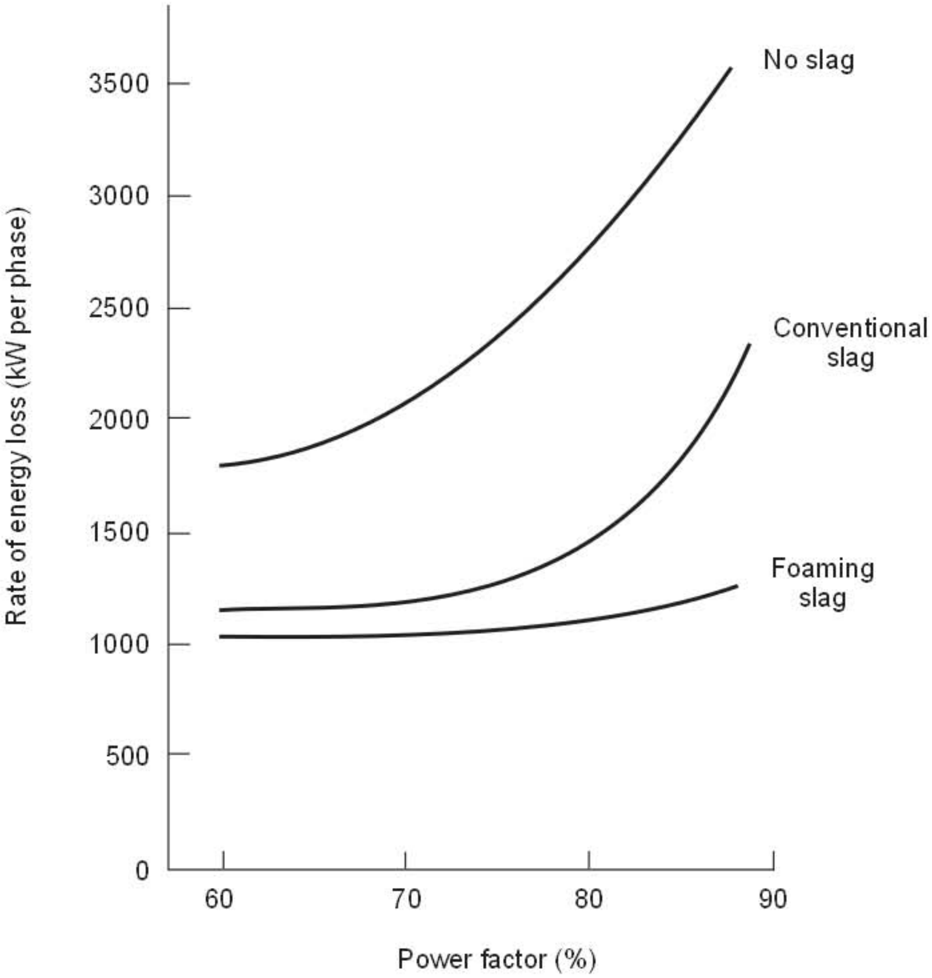


Fig. 10.66 Effect of slag cover on heat loss in the EAF. *(Courtesy of Center for Materials Production.)*

10.7.5.1 Introduction

Generically, post-combustion refers to the burning of any partially combusted compounds. In EAF operations both CO and H_2 are present. CO gas is produced in large quantities in the EAF both from oxygen lancing and slag foaming activities and from the use of pig iron or DRI in the charge. Large amounts of CO and H_2 are generated at the start of meltdown as oil, grease and other combustible materials evolve from the surface of the scrap. If there is sufficient oxygen present, these compounds will burn to completion. In most cases there is insufficient oxygen for complete combustion and high levels of CO result. Tests conducted at Vallourec by Air Liquide showed that the offgas from the furnace could contain considerable amounts of non-combusted CO when there was insufficient oxygen present.[44]

The heat of combustion of CO to CO_2 is three times greater than that of C to CO (for dissolved carbon in the bath). This represents a very large potential energy source for the EAF. Studies at Irsid (Usinor SA) have shown that the potential energy saving is significant and could be a much as 72 kWh/ton.[45] If the CO is burned in the freeboard it is possible to recover heat within the furnace. Some of the expected benefits and concerns regarding post-combustion are given in Table 10.11. As more oxygen is used to reduce electrical consumption, there will be greater need for improved oxygen utilization. In addition, environmental regulations may limit CO_2 emissions. As a result it will be necessary to obtain the maximum benefit of oxygen in the furnace. This can only be achieved if most CO is burned in the furnace.

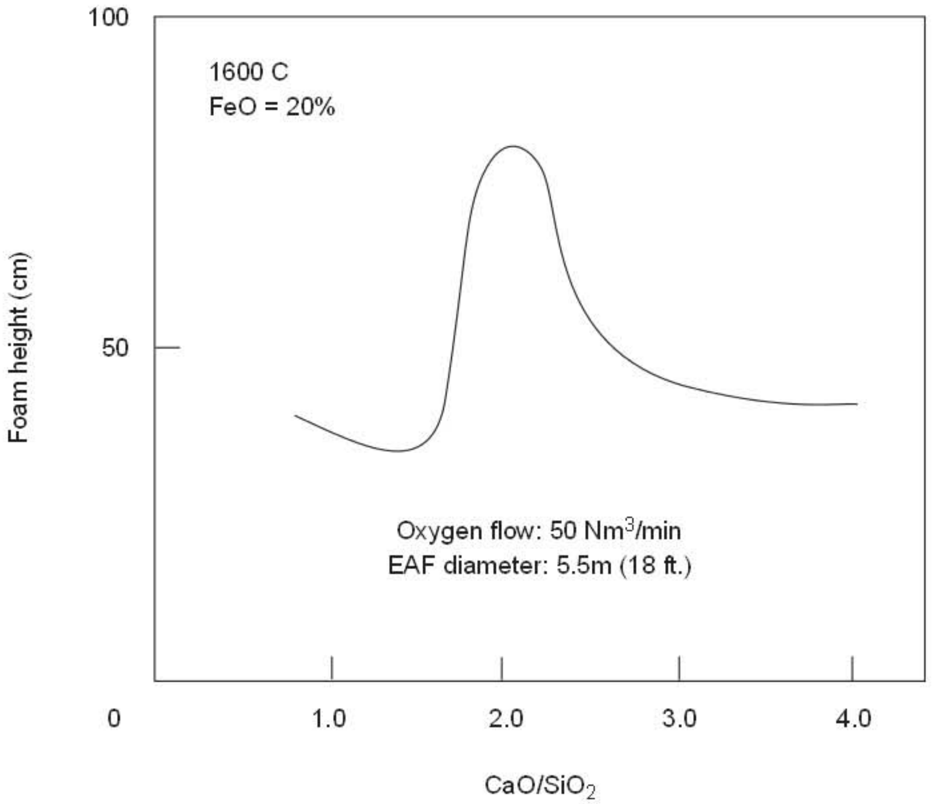


Fig. 10.67 Slag foam height versus slag basicity. *(Courtesy of Center for Materials Production.)*

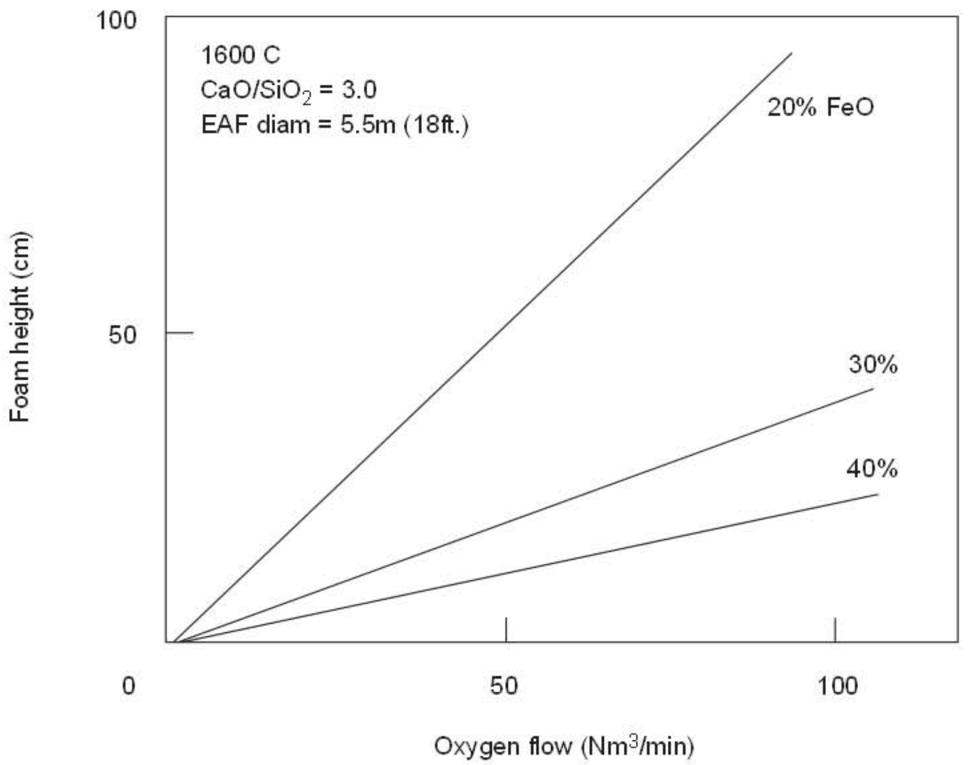


Fig. 10.68 Slag foam height versus oxygen injection rate for various FeO levels in the slag. *(Courtesy of Center for Materials Production.)*

Table 10.11 Benefits and Concerns for Post-Combustion

Benefits	Concerns
Decreased heat load to the offgas system	Increased electrode consumption
Decreased CO emissions to the meltshop and baghouse	Increased heat load to the water-cooled panels and roof
Higher heat transfer due to higher radiation from combustion products	Decreased iron yield
Decreased water-cooled duct requirement	Economics of additional oxygen
Increased utilization of energy from oxygen and carbon	
Reduced electrical power consumption	
Decreased NO_x emissions from the EAF	
Increased productivity without increased offgas system requirements	

In order to maintain consistency of the results presented the following definitions are made for EAF trials:

$$\text{Post combustion ratio (PCR)} \quad \frac{CO_2}{CO + CO_2} \tag{10.7.3}$$

$$\text{Heat transfer efficiency (HTE)} \quad \frac{\text{reduction in kWh to steel}}{\text{theoretical energy of PC for CO}} \tag{10.7.4}$$

10.7.5.2 Post-Combustion in Electric Arc Furnaces

Several trials have been run using post-combustion in the EAF. In some of the current processes, oxygen is injected into the furnace above the slag to post-combust CO. Some processes involve injection of oxygen into the slag to post-combust the CO before it enters the furnace freeboard. Most of these trials were inspired by an offgas analysis which showed large quantities of CO leaving the EAF. Fig. 10.69 shows a typical post-combustion system where oxygen is added in the freeboard of the furnace. Thus combustion products directly contact the cold scrap. Most of the heat transfer in this case is radiative. Fig. 10.70 shows the approach where post-combustion is carried out low in the furnace or in the slag itself. Heat transfer is accomplished via the circulation of slag and metal droplets within the slag. Post-combustion oxygen is introduced at very low velocities into the slag. Heat transfer is predominantly convective for this mode of post-combustion. Some other systems have incorporated bottom blown oxygen (via tuyeres in the furnace hearth) along with injection of oxygen low in the furnace. In fact the first two processes to employ extensive post-combustion as part of the operation (K-ES, EOF) both took this approach. These processes are discussed in detail in Section 10.9 on future developments.

Some of the results that have been obtained are summarized in Table 10.12 along with their theoretical limits. Note that in many cases the energy savings has been stated in kWh/Nm^3 of post-combustion oxygen added. The theoretical limit for post-combustion of CO at bath temperatures (1600°C) is 5.8 kWh/Nm^3 of oxygen. PCRs have been calculated based on the offgas analysis presented in these references. HTE is rarely measured in EAF post-combustion trials and is generally backed out based on electrical energy replacement due to post-combustion. This does not necessarily represent the true post-combustion HTE because the oxygen can react with compounds other than CO.

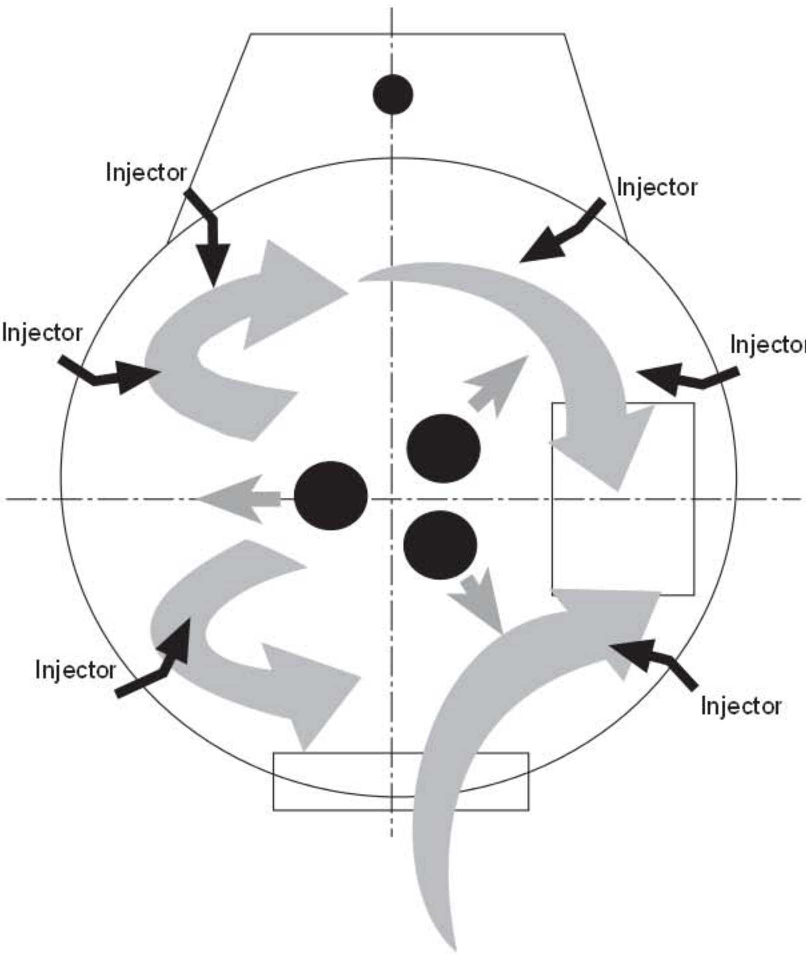


Fig. 10.69 Injection of post-combustion oxygen above the slag. *(Courtesy of Air Liquide.)*

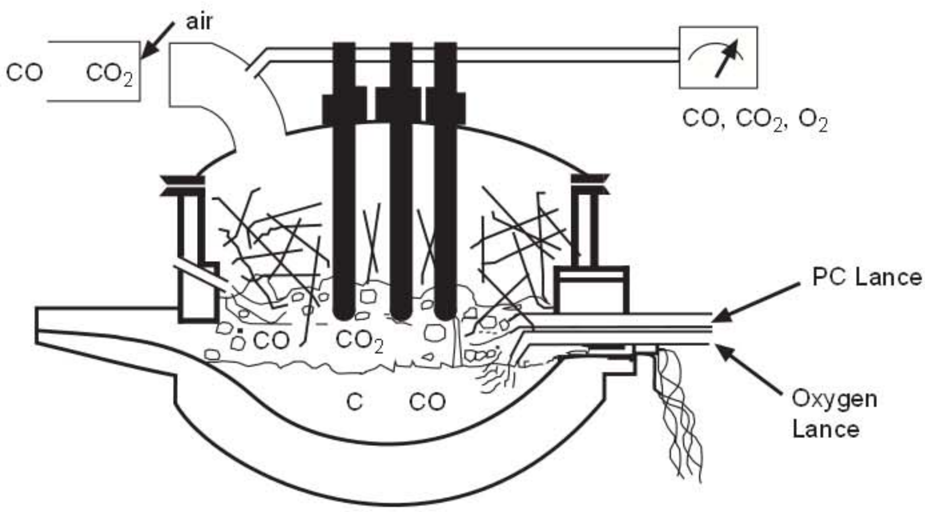


Fig. 10.70 Post-combustion in the slag layer. *(Courtesy of Praxair.)*

· Maximimum degree of post combustion and heat transfer
· PC in scrap melting and flat bath periods
· Unit power savings > 4 kWh/Nm^3 O_2
· 8̃ 10% power savings, 6̃ 8% productivity increase

Table 10.12 Typical Post-Combustion Results for Scrap Based Operations

Location	Method	Net Efficiency	HTE %	PCR %	kWh/Nm³ O_2	Reference
Theoretical		**100%**			5.8	
Acciaierie Venete SpA	K-ES	40–50%		80		46
Commonwealth Steel Co Ltd	Independent	30%				42
Ferriere Nord SpA	K-ES/DANARC	40%	50+	72–75	3–4	47,48,49
Vallourec	Air Liquide	60%		66–71		50,51
Ovako Steel AB	AGA	61%		77–87	4.3	52
Nucor	Praxair	80% (lance)	80	42–75	5.04	53,54,55
		60% (burners)			3.78	53,54,55
UES Aldwerke	Air Products/ Independent	64 %			3.7	56,57
Von Roll (Swiss Steel AG)	Air Liquide				2.53	58
Ferriera Valsabbia SpA	Air Liquide	Burners			1.2–1.6	58
		ALARC PC			3.0–3.5	58
Atlantic Steel	Praxair			60–70	3.7–5.13	60
Cascade Steel	Air Liquide				2.5–3.0	61
Badische Stahlwerke GmbH	Air Liquide	64%		83	3.69	58,59
Cia Siderurgica Pains SA	Korf EOF	60%		80–100		18,62,63

10.7.5.3 Theoretical Analysis of Post-Combustion

Due to the scatter of reported results to date for EAF post-combustion it is necessary to discuss the theoretical analysis to put the results into perspective. In addition, the trials in converters and smelters can help to draw conclusions.

10.7.5.3.1 Post-Combustion and Heat Transfer Mechanisms Heat transfer to a mass of solid scrap is relatively efficient due to its large surface area and the inherent large difference in temperature. However, heat transfer to the bath is more difficult and two methods are proposed. Interaction of metal and slag droplets occurs in the Hismelt process while heat transfer is through the slag phase in deep slag (DIOS and AISI) processes. In bath smelting, post-combustion energy is consumed and is not transferred to the bath. Exact mechanisms are not known at this time.

10.7.5.3.1.1 Post-Combustion in the Furnace Freeboard All of the post-combustion results indicate that high efficiency can only be achieved by effectively transferring the heat from the post-combustion products to some other material. This has been verified in several BOF post-combustion trials where scrap additions during post-combustion resulted in higher post-combustion levels.[64,65] If the combustion products do not transfer their heat, they will tend to dissociate. The best opportunity during scrap melting is to transfer the heat to cold scrap. The adiabatic flame temperature for the combustion of CO with oxygen levels off at 2800°C.[66] The adiabatic flame temperature does not increase proportionally to the amount of CO burned. Thus the amount of heat transferred from the flame does not increase proportionally.[66] The adiabatic flame temperature for oxygen/natural gas combustion is in the range of 2600–2700°C.[67,68] Natural gas combustion produces a greater volume per unit oxygen and therefore more contact with the scrap should occur. Trials with oxy-fuel and oxy-coal burners indicate that efficiencies of 75–80% can be obtained at the start of meltdown, but that efficiency drops off quickly as the scrap heats and an average efficiency of 65% can be expected.[34,69] For post-combustion, the efficiency can be expected to be similar or lower, depending on how long the process is carried out. Burner efficiency over a flat bath is 20 to 30%, setting a lower limit and as a result, heat flux to the furnace panels will increase. Air Liquide has reported PCR = 70% and HTE = 50% for flat bath operations with post-combustion. This equates to a net efficiency of 35%.

Increased electrode consumption is a possibility. High bath stir rates may help to recover more of this energy. If the heat is not transferred from the post-combustion gases then a high PCR can not be expected since some of the CO_2 will dissociate.[70,71,72] During scrap melting oil and grease burn off the scrap. CO and H_2 concentration spikes occur when scrap falls into the bath. The hydrogen from these organics tend to form hydoxyl ions which enhance the rate at which CO can be post-combusted.[67,73,74] The reaction between CO and O_2 is believed to be a branched chain reaction whose rate is greatly enhanced if a small quantity of H_2 is present. Thus higher PCRs in the EAF should be achieved during scrap meltdown. This benefit will not occur for post-combustion carried out in the slag though oxygen injected into the slag will continue to react in the furnace freeboard once it leaves the slag.

Nippon Steel found in their trials that for high offgas temperatures, radiation was the dominant heat transfer mechanism but the net amount of heat that was transferred was low.[75] For temperatures <1765°C, heat transfer by radiation and convection accounted for 30% of the heat transfer. The remainder was due to circulation of materials in the slag.

10.7.5.3.1.2 Post-Combustion in the Slag Layer Praxair has taken the approach of using subsonic or soft blowing of post-combustion oxygen through a multi-nozzle lance. The post-combustion lance is positioned close to the primary oxygen injection lance. The impact of the supersonic primary oxygen jet produces an emulsion of metal droplets in the slag. For post-combustion operations in an EAF, Praxair estimates that 0.25% of the steel bath needs to be emulsified in order to transfer all of the post-combustion heat. Praxair has found no evidence that post-combustion oxygen reacts with the metal droplets or with carbon in the slag. Post-combustion is carried out both during scrap melting and during a flat bath. During the start of scrap melting, post-combustion is difficult to achieve because oxygen lances (especially water-cooled lances) cannot be introduced into the furnace until sufficient scrap has meted back from the slag door. Information supplied by Praxair indicates that they do not attempt to burn all of the CO which is generated. Rather, they attempt to burn only a portion of the CO ranging from 10–50% of that which is generated from primary oxygen lancing.

The injection of oxygen must be localized and well controlled to avoid oxygen reaction with the electrodes. Previous trials for post-combustion in EAF slags have used a piggyback lance over top of the decarburizing oxygen lance. Thus an attempt is made to burn the CO in the slag as it leaves the bath.

The slag layer must be thick in order to avoid the reaction of CO_2 with the bath to produce CO which would make a high PCR difficult to achieve. In converter and bath smelting operations, the slag layer is considerably thicker than in the EAF. Sumitomo proposed that a deep slag layer be used with post-combustion in the upper layer of the slag where the oxygen would not react with metal droplets.[76] The temperature gradient in the slag layer was minimized with side blown oxygen into the slag. From Nippon Steel indications were that moderate bath stirring is necessary to accelerate reduction of iron ore and to enhance heat transfer from post-combustion.[75] A thick slag layer was used in order to separate the metal bath from the oxygen and the post-combustion products. A thick slag layer also helped to increase post-combustion and decrease dust formation. It might prove economical to couple slag post-combustion in the EAF with addition of DRI/HBI following scrap melting operations although DRI addition typically results in much slag foaming. Post-combustion could take place in the upper layer of the slag. Intense stirring would transfer heat back to the slag/bath interface where the DRI/HBI would be melted. This could lead to optimum energy recovery for post-combustion in the slag. Bath smelting results indicate that a net efficiency of about 50% can be achieved when CO is post-combusted in the slag.[75,77,78,79]

During scrap melting operations, post-combustion above the slag low in the furnace freeboard should give the best results. During flat bath operations, it may be advantageous to burn the CO in or just above the slag. However, post-combustion in the slag will be a much more complicated process and will require a greater degree of control over the post-combustion oxygen dispersion pattern in the slag in order to prevent undesirable reactions.

10.7.5.3.2 Electrode Consumption Previous studies have attributed electrode consumption to two mechanisms: sidewall consumption and tip erosion. Tip erosion is dependent on the electrical current being carried by the electrode. Sidewall consumption is dependent on the surface temperature of the electrode and can be controlled by providing water cooling of the electrode (sprays). During the initial stages of meltdown, the scrap helps to shield the electrodes and the hot surface will not react with oxygen present in the furnace. Once the scrap melts back from the electrodes, the hot carbon surface will react with any oxygen that is present. The source of the oxygen can be injected oxygen or oxygen that is present in air that is pulled into the EAF through the slag door.

If the electrodes are not shielded from the combustion products of post-combustion, carbon can react with CO_2 to form CO.[53] The electrode will also react with any available oxygen. Both mechanisms will increase electrode consumption. At high offgas temperatures, the amount of electrode consumption can increase drastically.

Readers are refered to Section 10.3.3 for electrode consumption factors developed by Bowman. These indicate an increase in electrode consumption for operations using large amounts of oxygen and those using post-combustion in the furnace.

10.7.5.3.3 Heatload to the EAF Shell The usefulness of the heat generated by post-combustion will be highly dependent on the effective heat transfer to the steel scrap and the bath. The theoretical adiabatic flame temperature for CO combustion with oxygen (2800°C) is similar to that for combustion of natural gas with oxygen. It is well known that when oxy-fuel burners are fired at very high rates on dense scrap, the flame can blow back against the water-cooled panels and overheat them. If the heat from post-combustion is not rapidly transferred to the scrap, overheating of the panels will also occur. If the combustion products do not penetrate the scrap, then most of the heat transfer will be radiative and the controlling rate mechanism will be conductive heat transfer within the scrap. This will be very slow compared to the radiative heat transfer to the scrap surface and will result in local overheating.

10.7.5.3.4 Iron Yield Oxy-fuel burner use can lead to yield losses and increased electrode consumption as some combustion products react with iron to form FeO. Trials run by Leary and Philbrook on the preheating of scrap with oxy-fuel burners showed that above scrap temperatures of 760°C, 2–3% yield loss occurred.[80] This is supported by work that shows that the thermodynamic equilibrium between iron and CO_2 at temperatures greater that 1377°C is 24% CO_2. Gas temperatures exceeding 1800°C are possible in the EAF and the corresponding equilibrium at this temperature is 8.6% CO_2. Though the gas residence time in the EAF probably will not allow for equilibrium to be reached, some oxidation will occur. If additional carbon is not supplied a yield loss will occur. This is likely to be the case for post-combustion. An iron yield loss of 1% equates to a power input of 12 kWh/ton. This can have a significant effect on the overall post-combustion heat balance resulting in a fictitiously high HTE for post-combustion.

As Fe is oxidized to FeO, a protective layer can form on the scrap. Once the FeO layer is formed, oxygen must diffuse through the layer in order to react with the iron underneath. This will help to protect the scrap from further oxidation if this layer does not peel off exposing the iron. At temperatures above 1300°C the FeO will tend to melt and this protective layer will no longer exist.

10.7.5.3.5 Limits on Potential Gains from Post-Combustion Bender indicates that CO emissions typically range between 0.3 and 2.6 kg/ton.[81] Air infiltration into the furnace will result in natural post-combustion. This is indicated in several EAF post-combustion studies where CO2 levels prior to the trials are in the range of 15–40% and PCR ranges from 30–60%.[44,50,53,57,59] With a post-combustion system, CO levels typically dropped to below 10%. The starting level of CO has a big effect on the efficiency obtained once post-combustion is installed. These results are presented in Table 10.13.

Table 10.13 PCR With and Without Injection of Post-Combustion Oxygen

Location	PCR prior to post-combustion of O_2	PCR after post-combustion of O_2
Vallourec	33–50%	66–71%
BSW	58–61%	83%
Nucor Plymouth	30–50%	42–75%
UES Aldewerke	40–62.5%	
Ferrierre Nord		71–75%
Venete		80%
Co-Steel Sheerness	22–62.5%	
AGA	44–66%	77–87%

CO_2 has a higher heat capacity than CO. As a result for the same offgas temperature, the CO_2 will remove more heat per Nm^3. Thus a portion of the post-combustion energy will be removed from the furnace in the CO_2. For an offgas temperature range of 1200–1400°C the maximum HTE is limited to 93.6–92%. If the slag is heated by 110°C the heat contained is equivalent to 0.9–1.2 kWh/ton (assumes 4–5% slag). These factors should be taken into account.

Some benefits are generally unaccounted for when post-combustion is implemented. A decrease of one minute in tap-to-tap time can save 2–3 kWh/ton.[82] A more recent study indicates a savings of 1 kWh/ton for highly efficient furnaces.[83] This is dependent on the efficiency of the current operation. Decreased cycle times should be considered when backing out the true heat transfer efficiency.

10.7.5.3.6 Environmental Benefits Increased volumes of injected gas will tend to decrease the amount of furnace infiltrated air. A positive pressure operation can save from 9–18 kWh/ton and some savings can be expected to result from a reduction in air infiltration depending on the operation of the offgas system. However, a reduction in NO_x due to reduced air infiltration has yet to be shown in EAF post-combustion operations. If we consider the amount of air infiltration at the start of meltdown we can see that the scrap will help to retard airflow into the furnace. Thus at the start of meltdown, perhaps the effective opening area is only 25% of the slag door area. By the end of meltdown this area is almost entirely open so now the effective opening might be 90% of the slag door area. Thus the amount of air infiltration into the furnace increases drastically over the course of the meltdown period. This will have a big affect on the effectiveness of auxilliary post-combustion oxygen added to the furnace.

Post-combustion helps the fume system capacity because the amount of heat that must be removed by the water-cooled duct is decreased. Several operations report decreased offgas temperatures when using post-combustion.[49,58] CO emissions at the baghouse are a function of offgas system design (sizing of the combustion gap, evacuation rate). Burning of some of the CO in the EAF will help to ensure that less CO enters the canopy system which will help to reduce CO emissions. Lower CO emissions have been reported in several EAF post-combustion operations.[55,58] If CO levels are reduced below 5% in the furnace, it may be difficult to complete CO combustion in the offgas system following addition of dilution air at the combustion gap. Thus controlling CO levels leaving the furnace to approximately 10% is a good practice. Alternatively a post-combustion chamber may be used within the offgas system.

It has been shown that CO tends to react with NO at high temperatures to form nitrogen and carbon dioxide as follows:[74]

$$2CO + 2NO \rightarrow 2CO_2 + N_2 \quad (10.7.5)$$

Thus it is beneficial to have a certain amount of CO present in the furnace in order to reduce the NO emissions. This is another good reason why total post-combustion of the CO is not desireable.

Trials carried out at Dofasco on their K-OBM showed that when excessive slag foaming occurred, particulate emissions from the furnace increased by as much as 59%.[84] Nippon Steel found that a thick slag layer helped to increase post-combustion and decrease dust formation.[75] For post-combustion in the slag in the EAF, high dust generation rates should be expected unless a thick slag layer is used. Excessive stirring of the bath will also lead to increased dust generation if metal droplets react with the post-combustion oxygen.

10.7.5.3.7 Need for a Post-Combustion Chamber If an attempt is made to burn close to all of the CO in the furnace, it is likely that any CO exiting the furnace will not burn at the combustion gap because the concentration will be below the lower flammability limit. If good heat transfer is not achieved in the furnace, some of the CO_2 will dissociate to CO and O_2 and as a result the CO concentration in the offgas will be higher than anticipated. If the combustion gap is not designed accordingly, there will be insufficient combustion air and some of the CO will report downstream to the gas cleaning equipment. If chlorine and metallic oxides (catalyst) are present in the offgas stream, there is a possibility of dioxin formation. The best way to ensure that this does not happen is to install a combustion chamber followed by a water spray quench to cool the gases below the temperature at which they will dissociate and react to form dioxins. This will also help to ensure low levels of CO downstream in the offgas system.

10.7.5.4 Conclusions

The following conclusions are drawn.

1. High levels of PCR in excess of 80% have been demonstrated for EAF operations.
2. An upper limit of 65% can be expected for HTE from post-combustion when there is cold scrap present. For post-combustion above the slag this will drop to 20–30% when there is no scrap present. For post-combustion in the slag an HTE of 80% has been reported. A net efficiency of 50–60% (PCR × HTE) may be achieved if DRI/HBI is added to absorb the energy released by post-combustion or if good heat transfer is achieved via circulation of metal droplets in the slag. During scrap melting operations, post-combustion above the slag should give the best results. During flat bath operations, it appears to be advantageous to burn the CO in or just above the slag.
3. Environmental benefits due to post-combustion have been demonstrated, but additional work needs to be carried out to better understand and optimize these benefits.
4. Potential gains due to post-combustion are highly dependent on the individual EAF operation efficiencies. Most applications of post-combustion in the EAF achieve an energy savings of 20–40 kWh/ton.
5. The economic lower limit for CO level leaving the EAF needs to be established but will likely be in the range of 5–10% based on environmental considerations.
6. If it is attempted to post-combust all of the CO in the EAF, yield losses and increased electrode consumption will occur.
7. Attempts to post-combust all of the CO in the furnace will likely have a negative effect on NO_x levels.

8. Oxygen injection into the bath should start early so that post-combustion can be carried out while the scrap is still relatively cold and is capable of absorbing the heat generated. In order to be most effective decarburization oxygen needs to be distributed throughout the bath. This will help to reduce local iron oxidation in the bath (and therefore the amount of EAF dust generated) and will also distribute the CO that is generated throughout the furnace which will help to maximize energy recovery once it is post-combusted.

9. Staged post-combustion similar to that carried out in the EOF preheat chambers has the greatest potential for capturing the energy generated through post-combustion. This is because the energy from the post-combustion reaction can be transferred to the scrap thus cooling the offgas and avoiding dissociation. A similar effect can probably be achieved in the shaft furnace. In the conventional EAF it will not be possible to recover as much of this energy, and some gas dissociation will likely take place.

10. Post-combustion in the slag can result in yield losses and an increase in the amount of EAF dust generated unless a thick slag layer is used. Alternatively partial post-combustion of the CO in the slag may prove to be more effective than complete post-combustion. The additional fluxes and energy consumed to provide a thick slag layer might offset any savings from post-combustion. In some operations where scrap preheating takes place, the dust is captured on the scrap and a conventional slag cover may be acceptable for post-combustion in the slag.

11. The optimum post-combustion strategy will vary from one operation to the next. Careful deliberation must be undertaken to determine the most cost effective means of applying post-combustion to EAF operations. Frequently, the optimum approach will involve selecting specific portions of the operating cycle in which post-combustion can be applied to give the highest returns. Other issues such as operability and maintenance requirements will also help to determine the complexity of the strategy employed.

Complete post-combustion of CO will be difficult to achieve and is likely uneconomical in light of possible detrimental effects (yield loss, refractory wear, electrode consumption, damage to furnace panels). Thus a complete analysis should be carried out on each installation to determine the best post-combustion practice for that location. Such a program should include CO, CO_2, H_2, H_2O, N_2, NO_x, SO_x gas analysis at the elbow, offgas flowrate and temperature exiting the EAF and at the baghouse, offgas analysis at the baghouse, analysis of baghouse dust before and after post-combustion for increases in iron content, monitoring of iron yield, electrode consumption and other consumables, monitoring EAF cooling water temperatures, slag analysis (check iron levels), and monitoring alloy consumption (check alloy yield).

Post-combustion can be an effective tool for the EAF operator but an economical operating practice must be established based on individual site criteria. There is no universal recipe which can apply for every facility. The way in which post-combustion can be applied is highly dependent on raw materials and operating practices and these must be thoroughly evaluated when implementing a post-combustion practice.

10.7.6 EAF Bottom Stirring

For conventional AC melting of scrap there is little natural convection within the bath. Temperature gradients have been reported in the range of 40–70°C.[43] If there is limited bath movement, large pieces of scrap can take considerably longer to melt unless they are cut up as discussed previously under oxygen lancing operations. Concentration gradients within the bath can also lead to reduced reaction rates and over or under reaction of some portions of the bath.

The concept of stirring the bath is not a new one and records indicate that electromagnetic coils were used for stirring trials as early as 1933.[85] Japanese studies indicated that flow velocities are much lower for stainless steels as compared to carbon steels for electromagnetic stirring.[86] Studies indicate that electromagnetic stirring is capable of supplying sufficient stirring in some cases. However, the cost for electromagnetic stirring is high and it is difficult to retrofit into an existing operation.

Most EAF stirring operations presently in use employ gas as the stirring medium.[87,88] These operations use contact or non-contact porous plugs to introduce the gas into the furnace. In some cases tuyeres are still used. The choice of gas used for stirring seems to be primarily argon or nitrogen though some trials with natural gas and with carbon dioxide have also been attempted.[89] In a conventional EAF three plugs are located midway between the electrodes.[87,88] For smaller furnaces a single plug centrally located appears to be sufficient. In EBT and other bottom tapping operations, the furnace tends to be elliptical and the nose of the furnace tends to be a cold spot. A stirring element is commonly located in this part of the furnace to promote mixing and aid in meltdown. Some operations have also found it beneficial to inject inert gas during tapping to help push the slag back and prevent slag entrainment in the tap stream. For common steel grades the gas flowrate for a contact system is typically 0.03–0.17 Nm^3/min. (1–6 scfm) with a total consumption of 0.1–0.6 Nm^3/ton (3–20 scf/ton).[4] Non-contact systems appear to use higher gas flow rates. Service life for contact systems is in the range of 300–500 heats. Some non-contact systems have demonstrated lives of more than 4000 heats.[90] Fig. 10.71 shows several commercially available stirring elements.

Some of the main advantages attributed to bottom stirring include: reduction in carbon boils and cold bottoms, yield increases of 0.5–1%, time savings of 1–16 minutes (typical is 5 min.) per heat,

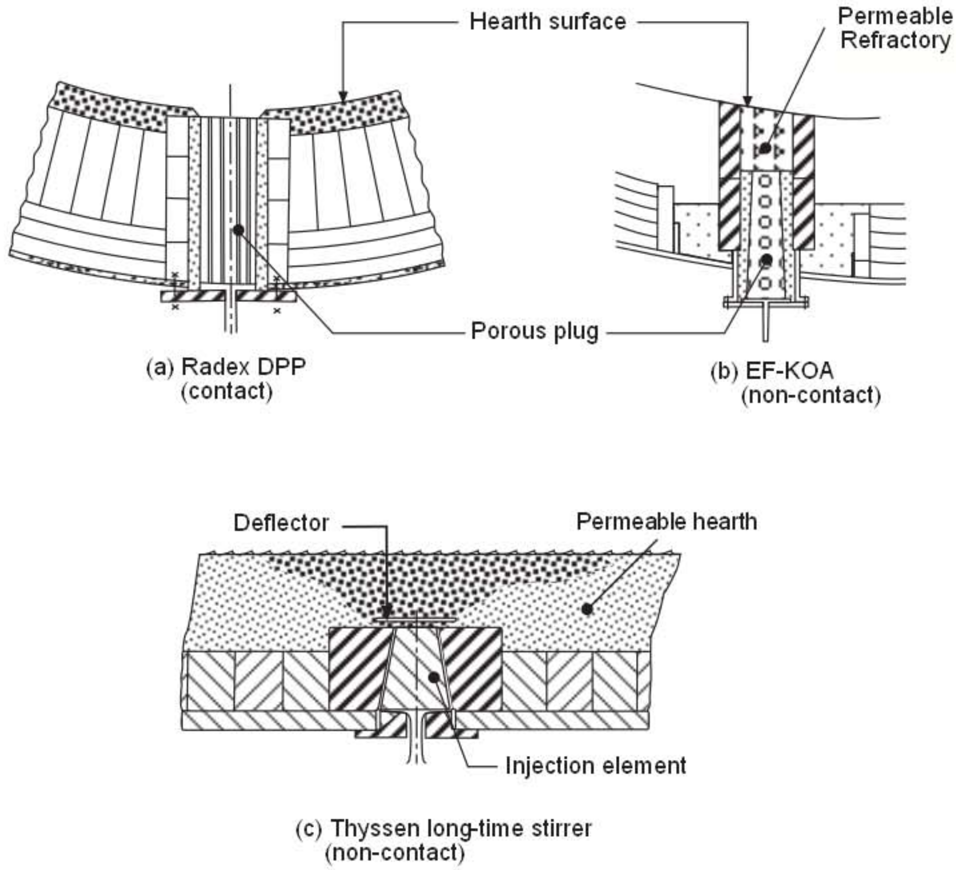


Fig. 10.71 Stirring element configurations. *(Courtesy of Center for Materials Production.)*

energy savings of up to 43 kWh/ton (typical is 10–20 kWh/ton), improved alloy recovery, increased sulfur and phosphorus removal, and reduced electrode consumption.[43,87–91]

In various operations surveyed the cost savings have ranged from $0.90 to $2.30 per ton.[89,90]

10.7.7 Furnace Electrics

In addition to increased oxygen use in the EAF, considerable effort has been devoted towards maximizing electrical efficiency. This has been partly due to the fact that there are practical limitations as to the amount of oxygen used in any one operation (due to environmental concerns). In addition, it has been realized that by using longer arc, lower current operations, it is possible to achieve much more efficient power input to the furnace. Several innovations have contributed to increased efficiencies in this area and are discussed in the following sections.

10.7.7.1 Electrode Regulation

In the days when furnaces melted small sized heats (5–20 tons), electrode regulation was not a major concern. As furnaces became bigger and operating voltages increased however, it became necessary to control the electrodes more closely in order to maximize the efficiency of power input to the furnace. Over the past five years, there have been some substantial advances made in electrode regulation.[92,93] These coupled with advances in furnace hydraulics (thus allowing for faster electrode response) have lead to considerable improvements in EAF operation. For one installation the following benefits were obtained following an upgrade of the electrode regulation system: power consumption decreased by 5%, flicker decreased by 10%, broken electrodes decreased by 90%, electrode consumption decreased by 8.5%, tap-to-tap time decreased by 18.5%, and average power input increased by 8.5 %.[92]

10.7.7.2 Current Conducting Arms

In conventional EAF design, the current is carried to the electrodes via bus tubes. These bus tubes usually contribute approximately 35% of the total reactance of the secondary electrical system.[4] Current conducting arms combine the mechanical and electrical functions into one unit. These arms carry the secondary current through the arm structure instead of through bus tubes. This results in a reduction of the resistance and reactance in the secondary circuit which allows an increase in power input rate without modification of the furnace transformer. Productivity is also increased. Current conducting arms are constructed of copper clad steel or from aluminum.[94–97] Some of the benefits attributed to current conducting arms include increased productivity, increased power input rate (5–10%), reduced maintenance and increased reliability, lower electrode consumption, and reduced electrode pitch circle diameter with a subsequent reduction in radiation to sidewalls.[94–97]

The aluminum current conducting arms, Fig. 10.72, are up to 50% lighter than conventional or copper clad steel arms. Several additional benefits are claimed including higher electrode speeds resulting in improved electrode regulation, reduced strain on electrode column components, and less mechanical wear of components.[94,97]

In comparison to the overall weight of arms and column, the weight of the arms should not be a significant factor.

10.7.8 High Voltage AC Operations

In recent years a number of EAF operations have retrofitted new electric power supplies in order to supply higher operating voltages.[34,98–101] Energy losses in the secondary circuit are dependent on the secondary circuit reactance and to a greater extent on the secondary circuit current. If power can be supplied at a higher voltage, the current will be lower for the same power input rate. Operation with a lower secondary circuit current will also give lower electrode consumption.[34] Thus it is advantageous to operate at as high a secondary voltage as is practical. Of course this is

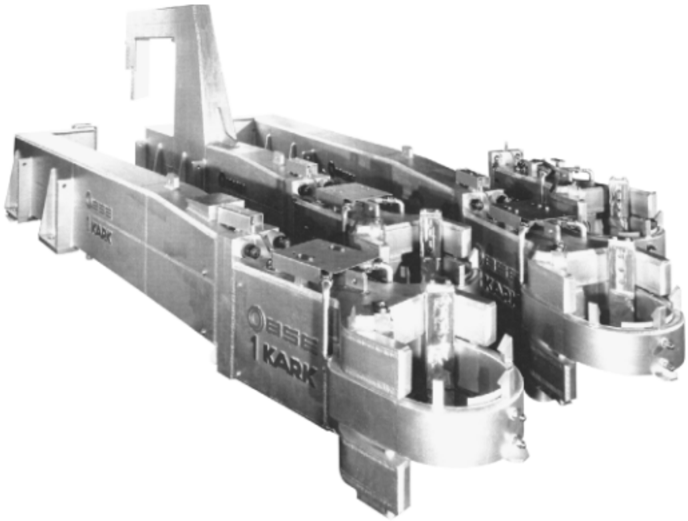


Fig. 10.72 Aluminum current conducting electrode arms. *(Courtesy of Badische Stahl Technology.)*

limited by arc flare to the sidewall and the existing furnace electrics. A good foamy slag practice can allow voltage increases of up to 100% without adversely affecting flare to the furnace sidewalls.[4] Energy losses can be minimized when reactance is associated with the primary circuit.

Supplementary reactance is not a new technology. In the past, supplementary reactors were used to increase arc stability in small furnaces where there was insufficient secondary reactance. However, in the past few years this method has been used to increase the operating voltages on the EAF secondary circuit. This is achieved by connecting a reactor in series with the primary windings of the EAF transformer.[98–100] This allows operation at a power factor of approximately $\sqrt{2}/2$ which is the theoretical optimum for maximum circuit power. This is made possible because the arc sees a large storage device in front of it in the circuit, which in effect acts as an electrical flywheel during operation. The insertion of the series reactor drops the secondary voltage to limit the amount of power transferred to the arc. In order to compensate for this, the furnace transformer secondary voltage is increased into the 900–1200V range allowing operation at higher arc voltages and lower electrode currents. Some of the benefits attributed to this type of operation are a more stable arc than for standard operations, electrode consumption reduced by 10%, secondary voltage increased by 60–80%, power savings of 10–20 kWh/ton, system power factor of approximately 0.72, furnace power factor of approximately 0.90, lower electrical losses due to lower operating current, and voltage flicker is reduced up to 40%.[4,98–101]

10.7.9 DC EAF Operations

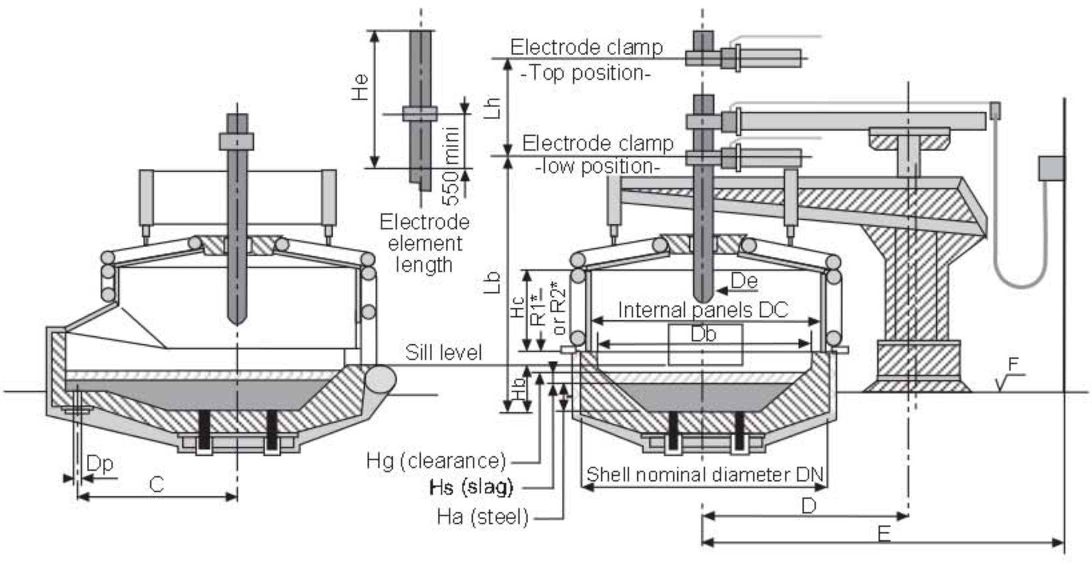
The progress in high power semiconductor switching technology brought into existence low cost efficient DC power supplies. Due to these advances, the high power DC furnace operation became possible. North American interest in DC furnace technology is growing with several existing installations and others that are currently under installation.[102–105] The DC arc furnace is characterized by rectification of three phase furnace transformer voltages by thyristor controlled rectifiers. These devices are capable of continuously modulating and controlling the magnitude of the DC arc current in order to achieve steady operation. DC furnaces use only one graphite electrode with the return electrode integrated into the furnace bottom. There are several types of bottom electrodes: conductive hearth bottom, conductive pin bottom, single or multiple billet, and conductive fins in a monolithic magnesite hearth.[102]

All of these bottom return electrode designs have been proven. The ones that appear to be used most often are the conductive pin bottom where a number of pins are attached to a plate and form the return path and the bottom billet design. The bottom electrode is air cooled in the case of the pin type and water-cooled in the case of the billet design. The area between pins is filled with ramming mix and the tip of the pins is at the same level as the inner furnace lining. As the refractory wears, the pins also melt back. DC furnaces operate with a hot heel in order to ensure an electrical path to the return electrode. During startup from cold conditions, a mixture of scrap and slag is used to provide an initial electrical path. Once this is melted in, the furnace can be charged with scrap.

Some of the early benefits achieved with DC operation included reduced electrode consumption (20% lower than high voltage AC, 50% lower than conventional AC), reduced voltage flicker (50–60% of conventional AC operation) and reduced power consumption (5–10% lower than for AC).[102,106–110]

These results were mainly achieved on smaller furnaces which were retrofitted from AC to DC operation. However, some larger DC furnace installations did not immediately achieve the claimed benefits. Notably, two areas of concern emerged: electrode consumption and refractory consumption.

Several DC furnace operations found that the decrease in electrode consumption expected under DC operation did not occur. Much analysis by the electrode manufacturers indicated that physical conditions within the electrodes was different for AC and DC operations. As a result, for large DC electrodes carrying very large current, an increased amount of cracking and spalling was observed as compared to AC operations. Therefore, it was necessary to develop electrodes with physical properties better suited to DC operation. This is discussed in greater detail in Section 10.3 on electrodes. The economical maximum size for DC furnaces tends to be a function of limitations due to electrode size and current carrying capacity. At the present time the maximum economical size for a single graphite electrode DC furnace appears to be about 165 tons. Larger furnace sizes can be accommodated by using more than one graphite electrode. Fig. 10.73 shows furnace dimensions for a Kvaerner Clecim DC EAF.



Shell nominal diameter DN 6.700 m	Sill level capacity 28.6 m^3	Laboratory total capacity 138 m^3

C	3.850 m	De	0.710 m	Hc	2.700 m	E	11.050 m
D	5.700 m	Dp	0.200 m	He	2.400 m	F	9.500 m
Db	5.648 m	Ha	1.051 m	Hg	0.08 m	Lb	6.305 m
DC	6.825 m	Hb	1.300 m	Hs	0.241 m	Lh	5.600 m

*R1 : without copper panel 0.290 m	*R2 : with copper panels

Fig. 10.73 DC furnace dimensions. *(Courtesy of Kvaerner Metals.)*

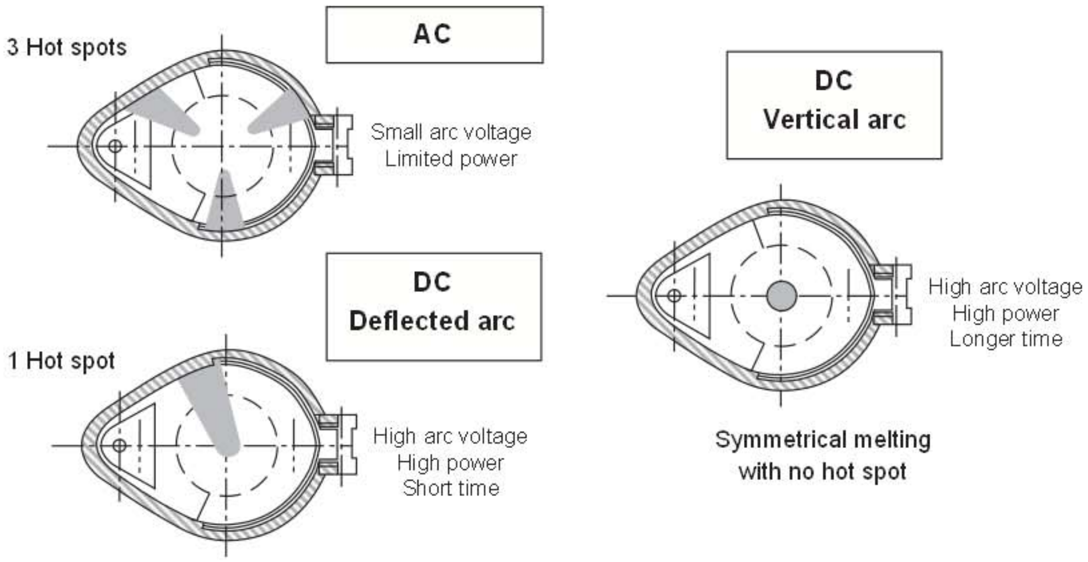


Fig. 10.74 Electric arc behavior. *(Courtesy of Kvaerner Metals.)*

Several of the early DC operations experienced problems with refractory wear and bottom electrode life. These problems were directly related to arc flare within the furnace. The anode design has the greatest influence on the arc flare. In all DC furnaces, the electric arc is deflected in the direction opposite to the power supply due to assymetries in magnetic fields which are generated by the DC circuit. Thus the arc tends to concentrate on one area within the furnace creating a hot spot and resulting in excessive refractory wear. This is shown in Fig. 10.74. Several solutions have been developed to control or eliminate arc flare. All commercial bottom electrode designs are now configured to force the arc to the center of the furnace. In the case of bottom conductive refractory and the pin type bottom, it is necessary to provide split feed lines to the bottom anode or a bottom coil which helps to modify the net magnetic field generated. In the billet bottom design, the amount of current to each billet is controlled along with the direction of anode supply in order to control the arc. The bottom fin design utilizes the fact that electrical feed occurs at several points in order control arc deflection. Quadrants located further from the rectifier are supplied with higher current than those located closer to the rectifier.

Some feel that the possibility for increased automation of EAF activities is greater for the DC furnace.[109,110] This is because with only one electrode, there is increased space both on top and within the furnace. DC furnace installations can be expected to cost from 10–35% more than a comparable AC installation.[106,107] However, calculations on payback indicate that this additional cost can be recovered in one to two years due to lower operating costs.[106,107]

Bowman conducted an analysis comparing AC and DC furnace operations and found that the electrical losses amount to approximately 4% in AC operations and 5.5% in DC operations; the difference in absolute terms is relatively insignificant.[9] The difference in total energy consumption between AC and DC furnaces is likely less than 9 kWh/ton in favor of the DC furnace, however many other variables influence the power consumption and it is difficult to develop accurate figures. DC furnaces experience roughly 25% less electrode consumption than AC furnaces, this correlating to typically 0.4 kg/ton. This difference appears to be greater for smaller AC furnaces. Flicker is approximately 60% lower for DC operations, however, advances in AC power system configurations (additional reactance) may reduce this difference to 40%.

Some typical results which have been presented for large DC EAF operations are electrode consumption of 1–2 kg/ton liquid steel, power consumption at 350–500 kWh/ton liquid steel, tap-to-tap times of 45–120 minutes, and bottom life of 1500–4000 heats. It is important to remember however, that power consumption is highly dependent on operating practices, tap temperature, use of auxiliary fuels, scrap type etc.

10.7.10 Use of Alternative Iron Sources in the EAF

Hot metal production is a standard part of operations in integrated steelmaking. Hot metal is produced in the blast furnace from iron ore pellets. This hot metal is then refined in basic oxygen furnaces to produce steel. However, several operations which were previously integrated operations are now charging hot metal to the EAF. One such installation is Cockerill Sambre in Belgium where up to 40% of the total charge weight is hot metal. This installation gets its hot metal from a blast furnace. In several other operations, hot metal is provided via Corex units, mini blast furnaces, or cupolas. In the case of the Saldahna Steel facility currently under construction in South Africa, the EAF feed will consist of 45% hot metal and 55% DRI.

10.7.10.1 Effect of Feedstocks on EAF Operation

There is a wide range of tabulated effects for various iron alternatives in the EAF. This is primarily due to the fact that within any given product such as DRI or HBI, a number of process parameters may vary quite considerably. These include % metallization, % carbon, % gangue, etc. All of these parameters will have an affect on the energy requirement to melt the material. If there is sufficient carbon to balance the amount of FeO in the DRI, the total iron content can be recovered. Approximately 1% carbon is required to balance out 6% FeO. If insufficient carbon is present, yield loss will result unless another source of carbon is added to the bath.. If excess carbon is present, it can be used as an energy source in conjunction with oxygen injection in order to reduce electrical power requirements. Generally speaking, DRI requires 100–200 additional kWh per ton to melt as compared to scrap melting. If up to 25% DRI is to be used in the charge makeup, it can be added in the bucket. If a larger percentage is to be used it can be fed continuously through the roof. One advantage of DRI is that it can be fed continuously with power on and therefore no thermal losses are incurred by opening the roof. HBI is more dense than DRI and as a result can be charged in the scrap bucket without increasing the number of charges required. DRI tends to float at the slag bath interface while HBI, which has a much higher density, tends to sink into the bath and melt in a manner similar to pig iron. The amount of silica present in the DRI will have a large effect on the economics of steel production from DRI. Silica will attack refractories unless sufficient lime is present to neutralize its effect. In general, a V-ratio of 2.5–3.0 is desired for good slag foaming. Thus the lime requirement increases greatly if silica levels in the DRI are high. Melting power requirements increase accordingly.

In the case of cold pig iron, a power savings should result from using 10–15% pig iron in the charge. This is due to the silicon and carbon contained in the pig iron. These act as a source of chemical heat in the bath when oxygen is injected. Pig iron typically contains up to 0.65% silicon which reacts with oxygen to produce silica which reports to the slag. This requires some additional lime addition in order to maintain the slag basicity. Usually, a maximum of 20% cold pig iron is used in the EAF because it takes longer to melt in than scrap, especially if it is supplied in large pieces. Small sized pieces are preferable. The pig iron can contain up to 4% carbon which results in a very high bath carbon level. Removal of this carbon with oxygen generates much heat but also requires increased blowing times because practical limits exist on the rate at which oxygen can be blown into the steel.

Iron carbide can be charged into the furnace in sacks or it can be injected. Injection is the preferred method of introducing the material into the bath as recovery is maximized. This however creates some practical limitations as to the quantity of iron carbide used since limitations on the injection rate exist. At Nucor Crawfordsville, the maximum rate achieved so far is 2500 kg per minute. Iron carbide dissolves into the steel bath and as the carbon goes into solution, it reacts with FeO or dissolved oxygen in the bath producing a very fine dispersion of carbon monoxide bubbles. These bubbles are very beneficial because they help to strip nitrogen from the bath. Depending on the degree of metallization in the iron carbide, the energy requirements for dissolution of the iron carbide also vary.

The charging of hot metal to the EAF sounds like a simple proposition though it is in fact quite complex. Care must be taken that the hot metal which is charged does not react with the highly oxidized slag which is still in the EAF. Some operations charge hot metal to the EAF by swinging the roof and pouring it into the furnace. This causes very rapid mixing of the hot heel and the highly oxidized slag in the EAF with the hot metal and sometimes explosions do occur. Thus for this mode of operation it is recommended that a slag deoxidizer be added prior to hot metal addition. Typical deoxidizers are silicon fines, aluminum fines and calcium carbide. An alternative method of charging the hot metal to the EAF is to pour it down a launder which is inserted into the side of the EAF. This method requires more time for charging of the hot metal but results in a much safer operation. Paul Wurth has recently developed a side charging system whereby the hot metal can be charged while power is on and thus the charging time is not an issue.

10.7.11 Conclusions

It is apparent that there are many technologies available for improving EAF operating efficiency. The general results for these have been listed in an attempt to provide a starting point for those in the process of upgrading operations. Of course results will vary from one installation to the next. However if a conservative approach is taken and the median of the reported results is used for calculation purposes, the results can be expected to be achievable. It is important to evaluate the effects of certain processes both on furnace operations and on other systems such as fume control. The use of substitute fuels (oxygen and natural gas) may be limited by the capacity of the fume system. If upgrades are to be made, one must also evaluate the need for upgrades to auxiliary systems. The data presented in this section provides a starting point for the person evaluating process changes to improve EAF efficiency. The technologies that have been reviewed are well proven. The interaction between these processes has not been evaluated, though in some cases the blending of these operations can prove to be most beneficial (eg. oxygen lancing, slag foaming and CO post-combustion). When evaluating upgrade requirements for a particular operation, it is necessary to clearly list the objectives and then match these with suitable technologies. It is important to maintain a global perspective regarding overall costs and operations in order to arrive at true optimal operating efficiency in the EAF.

10.8 Furnace Operations

10.8.1 EAF Operating Cycle

The electric arc furnace operates as a batch process. Each batch of steel that is produced is known as a heat. The electric arc furnace operating cycle is known as the tap-to-tap cycle. The tap-to-tap cycle is made up of the following operations: furnace charging, melting, refining, de-slagging, tapping and furnace turnaround. Modern operations aim for a tap-to-tap cycle of less than 60 minutes. With the advance of EAF steelmaking into the flat products arena, tap-to-tap times of 35–40 minutes are now being sought with twin shell furnace operations.

A typical 60 minute tap-to-tap cycle is :

first charge	3 minutes
first meltdown	20 minutes
second charge	3 minutes
second meltdown	14 minutes
refining	10 minutes
tapping	3 minutes
turnaround	7 minutes
Total	**60 minutes**

10.8.2 Furnace Charging

The first step in the production of any heat is to select the grade of steel to be made. Usually a heat schedule is developed prior to each production shift. Thus the melter will know in advance the schedule for the shift. The scrap yard operators will batch buckets of scrap according to the needs of the melter. Preparation of the charge bucket is an important operation, not only to ensure proper melt-in chemistry but also to ensure good melting conditions. The scrap must be layered in the bucket according to size and density in order to ensure rapid formation of a liquid pool in the hearth while also providing protection of the sidewalls and roof from arc radiation. Other considerations include minimization of scrap cave-ins which can break electrodes and ensuring that large heavy pieces of scrap do not lie directly in front of burner ports which would result in blow-back of the flame onto the water-cooled panels. The charge can include lime and carbon or these can be injected into the furnace during the heat. Many operations add some lime and carbon in the scrap bucket and supplement this with injection.

The first step in any tap-to-tap cycle is charging of the scrap. The roof and electrodes are raised and are swung out to the side of the furnace to allow the scrap charging crane to move a full bucket of scrap into place over the furnace. The bucket bottom is usually a clam shell design—i.e. the bucket opens up by retracting two segments on the bottom of the bucket, see Fig. 10.75. Another common configuration is the "orange peel" design. The scrap falls into the furnace and the scrap crane removes the scrap bucket. The roof and electrodes swing back into place over the furnace. The roof is lowered and then the electrodes are lowered to strike an arc on the scrap. This commences the melting portion of the cycle. The number of charge buckets of scrap required to produce a heat of steel is dependent primarily on the volume of the furnace and the scrap density. Most modern furnaces are designed to operate with a minimum of backcharges. This is advantageous because charging is dead time, whereby the furnace does not have power on and therefore is not melting. Minimizing these dead times helps to maximize the productivity of the furnace. In addition, energy is lost each time the furnace roof is opened. This can amount to 10–20 kWh/ton for each occurrence. Most operations aim for 2–3 buckets of scrap per heat and will attempt to blend their scrap to meet this requirement. Some operations achieve a single bucket charge. Continuous charging operations such as Consteel and the Fuchs shaft furnace eliminate the charging cycle.

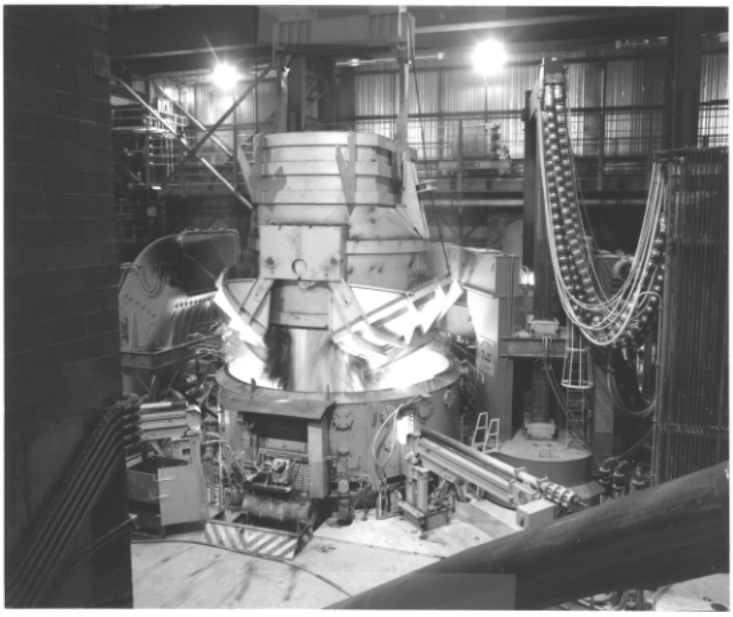


Fig. 10.75 Clam shell bucket charging scrap to the EAF. *(Courtesy of SMS GHH.)*

10.8.3 Melting

The melting period is the heart of EAF operations. The EAF has evolved into a highly efficient melting apparatus and modern designs are focused on maximizing its melting capacity. Melting is accomplished by supplying energy to the furnace interior. This energy can be electrical or chemical. Electrical energy is supplied via the graphite electrodes and is usually the largest contributor in melting operations. Initially, an intermediate voltage tap is selected until the electrodes can bore into the scrap. Usually, light scrap is placed on top of the charge to accelerate bore-in. After a few minutes, the electrodes will have penetrated the scrap sufficiently that a long arc (high voltage) tap can be used without fear of radiation damage to the roof. The long arc maximizes the transfer of power to the scrap and a liquid pool of metal will form in the furnace hearth. Approximately 15% of the scrap is melted during the initial bore-in period. At the start of melting the arc is erratic and unstable. Wide swings in current are observed accompanied by rapid movement of the electrodes. As the furnace atmosphere heats up the arcing tends to stabilize and once the molten pool is formed, the arc becomes quite stable and the average power input increases.

Chemical energy can be supplied via several sources such as oxy-fuel burners and oxygen lancing. Oxy-fuel burners burn natural gas using oxygen or a blend of oxygen and air. Heat is transferred to the scrap by radiation and convection. Heat is transferred within the scrap by conduction. In some operations, oxygen is used to cut scrap. Large pieces of scrap take longer to melt into the bath than smaller pieces. A consumable pipe lance can be used to cut the scrap. The oxygen reacts with the hot scrap and burns iron to produce intense heat for cutting the scrap. Once a molten pool of steel is generated in the furnace, oxygen can be lanced directly into the bath. This oxygen will react with several components in the bath including, aluminum, silicon, manganese, phosphorus, carbon and iron. All of these reactions are exothermic (i.e. they generate heat) and will supply energy to aid in the melting of the scrap. The metallic oxides which are formed will eventually reside in the slag. The reaction of oxygen with carbon in the bath will produce carbon monoxide which may burn in the furnace if there is oxygen available. Otherwise the carbon monoxide will carry over to the direct evacuation system. Auxiliary fuel operations are discussed in more detail in Section 10.7.

Once enough scrap has been melted to accommodate the second charge, the charging process is repeated. After the final scrap charge is melted, the furnace sidewalls can be exposed to high radiation from the arc. As a result, the voltage must be reduced. Alternatively, creation of a foamy slag will allow the arc to be buried and will protect the furnace shell. In addition, a greater amount of energy will be retained in the slag and is transferred to the bath resulting in greater energy efficiency. When the final scrap charge is fully melted, flat bath conditions are reached. At this point, a bath temperature and a chemical analysis sample will be taken. The analysis of the bath chemistry will allow the melter to determine the amount of oxygen to be blown during refining. The melter can also start to arrange for the bulk tap alloy additions to be made. These quantities are confirmed following refining.

10.8.4 Refining

Refining operations in the electric arc furnace have traditionally involved the removal of phosphorus, sulfur, aluminum, silicon, manganese and carbon. In recent times, dissolved gases in the bath have also become a concern, especially nitrogen and hydrogen levels. Traditionally, refining operations were carried out following meltdown , i.e. once a flat bath was achieved. These refining reactions are all dependent upon oxygen being available. Oxygen was lanced at the end of meltdown to lower the bath carbon content to the desired level for tapping. Most of the compounds which are to be removed during refining have a higher affinity for oxygen than carbon. Thus the oxygen will preferentially react with these elements to form oxides which will report to the slag.

In modern EAF operations, especially those operating with a hot heel, oxygen may be blown into the bath throughout the heat cycle. As a result, some of the refining operations occur concurrent with melting.

Most impurities such as phosphorous, sulfur, silicon, aluminum and chromium are partially removed by transfer to the slag. These refining reactions are discussed in detail in Chapter 2. In particular the equilibrium partition ratio between metal and slag are given as functions of slag chemistry and temperature.

The slag in an EAF operation will, in general, have a lower basicity than that for oxygen steelmaking. In addition, the quantity of slag per ton of steel will also be lower in the EAF. Therefore the removal of impurities in the EAF is limited. A typical slag composition is presented in Table 10.14.

Table 10.14 Typical Slag Constituents

Component	Source	Composition Range
CaO	Charged	40–60%
SiO_2	Oxidation product	5–15
FeO	Oxidation product	10–30%
MgO	Charged as dolomite	3–8%
CaF_2	Charged slag fluidizer	
MnO	Oxidation product	2–5%
S	Absorbed from steel	
P	Oxidation product	

Once these materials enter into the slag phase they will not necessarily stay there. Phosphorus retention in the slag is a function of the bath temperature, the slag basicity and FeO levels in the slag. At higher temperature or low FeO levels, the phosphorus will revert from the slag back into the bath. Phosphorus removal is usually carried out as early as possible in the heat. Hot heel practice is very beneficial for phosphorus removal because oxygen can be lanced into the bath while the bath temperature is quite low. Early in the heat the slag will contain high FeO levels carried over from the previous heat. This will also aid in phosphorus removal. High slag basicity (i.e. high lime content) is also beneficial for phosphorus removal but care must be taken not to saturate the slag with lime. This will lead to an increase in slag viscosity which will make the slag less effective for phosphorus removal. Sometimes fluorspar is added to help fluidize the slag. Gas stirring is also beneficial because it will renew the slag/metal interface which will improve the reaction kinetics.

In general, if low phosphorus levels are a requirement for a particular steel grade, the scrap is selected to give a low level at melt-in. The partition ratio of phosphorus in the slag to phosphorus in the bath ranges from 5.0–15.0. Usually the phosphorus is reduced by 20–50% in the EAF. However, the phosphorous in the scrap is low compared to hot metal and therefore this level of removal is acceptable. For oxygen steelmaking higher slag basicity and FeO levels give a partition ratio of 100 and with greater slag weight up to 90% of the phosphorous is removed.

Sulfur is removed mainly as a sulfide dissolved in the slag. The sulfur partition between the slag and metal is dependent on the chemical analysis and temperature of the slag (high basicity is better, low FeO content is better), slag fluidity (high fluidity is better), the oxidation level of the steel (which should be as low as possible), and the bath composition. Generally the partition ratio is 3.0–5.0 for EAF operations.

It can be seen that removal of sulfur in the EAF will be difficult especially given modern practices where the oxidation level of the bath is quite high. If high lime content is to be achieved in the slag, it may be necessary to add fluxing agents to keep the slag fluid. Usually the meltdown slag must be removed and a second slag built. Most operations have found it to be more effective to carry out desulfurization during the reducing phase of steelmaking. This means that desulfurization is commonly carried out during tapping (where a calcium aluminate slag is built) and during ladle furnace operations. For reducing conditions where the bath has a much lower oxygen activity, distribution ratios for sulfur of 20–100 can be achieved.

Control of the metallic constituents in the bath is important as it determines the properties of the final product. Usually, the melter will aim for lower levels in the bath than are specified for the final product. Oxygen reacts with aluminum, silicon and manganese to form metallic oxides which are slag components. These metallics tend to react before the carbon in the bath begins to react with the oxygen, see Table 2.1 in Chapter 2. These metallics will also react with FeO resulting in a recovery of iron units to the bath. For example:

$$Mn + FeO = MnO + Fe \tag{10.8.1}$$

Manganese will typically be lowered to about 0.06% in the bath.

The reaction of carbon with oxygen in the bath to produce CO is important as it supplies energy to the bath and also carries out several important refining reactions at the same time. In modern EAF operations, the combination of oxygen with carbon can supply between 30 and 40% of the net heat input to the furnace. Evolution of carbon monoxide is very important for slag foaming. Coupled with a basic slag, CO bubbles will help to inflate the slag which will help to submerge the arc. This gives greatly improved thermal efficiency and allows the furnace to operate at high arc voltages even after a flat bath is reached. Submerging the arc helps to prevent nitrogen from being exposed to the arc where it will dissociate and become dissolved in the steel.

If the CO is evolved within the bath, it will also remove nitrogen and hydrogen from the steel. The capacity for nitrogen removal is dependent on the amount of CO generated in the metal. Nitrogen levels as low as 50 ppm can be achieved in the furnace prior to tap. Bottom tapping is beneficial for maintaining low nitrogen levels as tapping is fast and a tight tap stream is maintained. A high oxygen content in the steel is beneficial for reducing nitrogen pickup at tap as compared to deoxidation of the steel at tap.

At 1600°C, the maximum solubility of nitrogen in pure iron is 450 ppm. Typically, the nitrogen levels in the steel following tapping are 80–100 ppm. An equation describing the solubility of nitrogen in the steel is given in Chapter 2.

Decarburization is also beneficial for the removal of hydrogen. It has been demonstrated that decarburizing at a rate of 1% per hour can lower hydrogen levels in the steel from 8 ppm down to 2 ppm in ten minutes.

At the end of refining, a bath temperature measurement and a bath sample are taken. If the temperature is too low, power may be applied to the bath. This is not a big concern in modern meltshops where temperature adjustment is carried out in the ladle furnace.

10.8.5 Deslagging

Deslagging operations are carried out to remove impurities from the furnace. During melting and refining operations, some of the undesirable materials within the bath are oxidized and enter the slag phase.

Thus it is advantageous to remove as much phosphorus into the slag as early in the heat as possible (i.e. while the bath temperature is still low). Then the slag is poured out of the furnace through the slag door. Removal of the slag eliminates the possibility of phosphorus reversion.

During slag foaming operations, carbon may be injected into the slag where it will reduce FeO to metallic iron and will generate carbon monoxide which helps to inflate the slag. If the high phosphorus slag has not been removed prior to this operation, phosphorus reversion will occur.

Fig. 10.76 EBT furnace during tapping. *(Courtesy of Fuchs.)*

10.8.6 Tapping

Once the desired bath composition and temperature are achieved in the furnace, the taphole is opened and the furnace is tilted so that the steel can be poured into a ladle for transfer to the next batch operation (usually a ladle furnace or ladle station). During the tapping process bulk alloy additions are made based on the bath analysis and the desired steel grade. Deoxidizers may be added to the steel to lower the oxygen content prior to further processing. This is commonly referred to as blocking the heat or killing the steel. Common deoxidizers are aluminum or silicon in the form of ferrosilicon or silicomanganese. Most carbon steel operations aim for minimal slag carryover. A new slag cover is built during tapping. For ladle furnace operations, a calcium aluminate slag is a good choice for sulfur control. Slag forming compounds are added in the ladle at tap so that a slag cover is formed prior to transfer to the ladle furnace. Additional slag materials may be added at the ladle furnace if the slag cover is insufficient. Fig. 10.76 shows an EBT furnace tapping into a ladle.

10.8.7 Furnace Turnaround

Furnace turnaround is the period following completion of tapping until the first scrap charge is dropped in the furnace for the next heat. During this period, the electrodes and roof are raised and the furnace lining is inspected for refractory damage. If necessary, repairs are made to the hearth, slagline, taphole and spout. In the case of a bottom tapping furnace, the taphole is filled with sand. Repairs to the furnace are made using gunned refractories or mud slingers. In most modern furnaces, the increased use of water-cooled panels has reduced the amount of patching or fettling required between heats. Many operations now switch out the furnace bottom on a regular basis (every 2–6 weeks) and perform the maintenance off-line. This reduces the power-off time for the

EAF and maximizes furnace productivity. Furnace turnaround time is generally the largest dead time in the tap-to-tap cycle. With advances in furnace practices this has been reduced from 20 minutes to less than five minutes in some newer operations.

10.8.8 Furnace Heat Balance

To melt steel scrap, it takes a theoretical minimum of 300 kWh/ton. To provide superheat above the melting point of 1520°C (2768°F) requires additional energy and for typical tap temperature requirements, the total theoretical energy required usually lies in the range of 350–370 kWh/ton. However, EAF steelmaking is only 55–65% efficient and as a result the total equivalent energy input is usually in the range of 560–680 kWh/ton for most modern operations. This energy can be supplied from a number of sources including electricity, oxy-fuel burners and chemical bath reactions. The typical distribution is 60–65%, 5–10% and 30–40% respectively. The distribution selection will be highly dependent on local material and consumable costs and tends to be unique to the specific meltshop operation. A typical balance for both older and more modern EAFs is given in the Table 10.15.

Table 10.15 Typical Energy Balance for EAFs

		UHP Furnace	Low to Medium Power Furnace
	Electrical Energy	50–60%	75–85%
INPUTS	Burners	5–10%	
	Chemical Reactions	30–40%	15–25%
	TOTAL INPUTS	**100%**	**100%**
	Steel	55–60%	50–55%
	Slag	8–10%	8–12%
OUTPUTS	Cooling Water	8–10%	5–6%
	Miscellaneous	1–3%	17–30%
	Offgas	17–28%	7–10%
	TOTAL OUPUTS	**100%**	**100%**

Several factors are immediately apparent from these balances. Much more chemical energy is being employed in the EAF and correspondingly, electrical power consumption has been reduced. Furnace efficiency has improved with UHP operation as indicated by the greater percentage of energy being retained in the steel. Losses to cooling water are higher in UHP operation due to the greater use of water-cooled panels. Miscellaneous losses such as electrical inefficiencies were much greater for older, low powered operations. Energy loss to the furnace offgas is much greater in UHP furnace operation due to greater rates of power input and shorter tap-to-tap times.

Of course the figures in Table 10.15 are highly dependent on the individual operation and can vary considerably from one facility to another. Factors such as raw material composition, power input rates and operating practices (e.g. post-combustion, scrap preheating) can greatly alter the energy balance. In operations utilizing a large amount of charge carbon or high carbon feed materials, up to 60% of the energy contained in the offgas may be calorific due to large quantities of uncombusted carbon monoxide. Recovery of this energy in the EAF could increase energy input by 8–10%. Thus it is important to consider such factors when evaluating the energy balance for a given furnace operation.

The International Iron and Steel Institute (IISI) classifies EAFs based on the power supplied per ton of furnace capacity. For most modern operations, the design would allow for at least 500 kVA per ton of capacity. The IISI report on electric furnaces indicates that most new installations allow for 900–1000 kVA per ton of furnace capacity.[18] Most furnaces operate at a maximum power factor of 0.85. Thus the above transformer ratings would correspond to a maximum power input of 0.75–0.85 MW per ton of furnace capacity.

10.9 New Scrap Melting Processes

Over the past ten years the steelmaking world has seen many changes in operating practices and the utilization of new process concepts in an attempt to lower operating costs and to improve product quality. In addition many new alternatives have been presented as lower cost alternatives to conventional AC EAF melting. Some of the specific objectives of these processes include lowering specific capital costs, increasing productivity, and improving process flexibility.

All of these processes share one or more of the following features in common. Energy from the waste offgas is used to preheat the scrap. Carbon is added to the bath and is later removed by oxygen injection in order to supply energy to the process. An attempt is made to combust the CO generated in the process to maximize energy recovery. An attempt is made to maximize power-on time and minimize turnaround time.

The EOF was one of the first scrap melting processes to use hot metal, post-combustion and scrap preheating and is described in Chapter 13.

Many of these process have their roots associated with scrap preheating. It is only fitting that this be discussed first in order to provide the groundwork for the development of these processes.

10.9.1 Scrap Preheating

Scrap preheating has been used for over 30 years to offset electrical steel melting requirements primarily in regions with high electricity costs such as Japan and Europe. Scrap preheating involves the use of hot gas to heat scrap in the bucket prior to charging. The source of the hot gases can be either offgases from the EAF or gases from a burner. The primary energy requirement for the EAF is for heating of the scrap to its melting point. Thus energy can be saved if scrap is charged to the furnace hot. Preheating of scrap also eliminates the possibility of charging wet scrap which eliminates the possibility of furnace explosions. Scrap preheating can reduce electrical consumption and increase EAF productivity.

Some suppliers have noted that there is a maximum preheat temperature beyond which further efforts to heat the scrap lead to diminished returns. This temperature lies in the range of 540–650°C. It is estimated that by preheating the scrap to a temperature of 425–540°C, a total of 63–72 kWh/ton of electrical energy can be saved. Early scrap preheaters used independent heat sources. The scrap was usually heated in the scrap bucket. Energy savings reported from this type of preheating were as high as 40 kWh/ton with associated reductions in electrode and refractory consumption due to reduced tap-to-tap times.[4,18]

As fourth hole offgas systems were developed, attempts were made to use the EAF offgas for scrap preheating. A side benefit that was reported was that the amount of baghouse dust decreased because the dust was sticking to the scrap during preheating. Scrap preheating with furnace offgas is difficult to control due to the variation in offgas temperature throughout the melting cycle. In addition a temperature gradient forms within the scrap. Temperatures must be controlled to prevent damage to the scrap bucket and in order to prevent burning or sticking of fine scrap within the bucket. Scrap temperatures can reach 315–450°C (600–850°F) though this will only occur at the hot end where the offgas first enters the preheater. Savings are typically only in the neighborhood of 18–23 kWh/ton.[4] In addition, as operations become more efficient and tap-to-tap times are decreased, scrap preheating operations become more and more difficult to maintain. Scrap handling operations can actually lead to reduced productivity and increased maintenance costs. At Badische Stalwerke the energy savings due to scrap preheating were decreased by 50% when tap-to-tap time was reduced by one third.

Some of the benefits attributed to scrap preheating are increased productivity by 10–20%, reduced electrical consumption, removal of moisture from the scrap, and reduced electrode and refractory consumption per unit production. Some drawbacks to scrap preheating are that volatiles are removed from the scrap, creating odors and necessitating a post-combustion chamber downstream. In addition spray quenching following post-combustion is required to prevent recombination of dioxins and furans. Depending on the preheat temperature, buckets may have to be refractory lined.

10.9.2 Preheating With Offgas

Preheating with offgas from the EAF requires that the offgas be rerouted to preheat chambers which contain loaded scrap buckets. The hot gases are passed through the buckets thus preheating the scrap. For tap-to-tap times less than 70 minutes the logistics of scrap preheating lead to minimal energy savings that will not justify the capital expense of a preheating system. Typical savings are in the range of 15–20 kWh/ton. Some examples of preheating systems are given in Fig. 10.77 and Fig. 10.78.

10.9.3 Natural Gas Scrap Preheating

Natural gas scrap preheating originated in the 1960s and usually involves a burner mounted in a refractory lined roof which sits over the top of the scrap bucket. Scrap is typically preheated to 540–650°C. Above 650°C, scrap oxidation becomes a problem and yield loss becomes a factor. Advantages of this form of scrap preheating are that the preheating process is decoupled from EAF operations and as a result the process is unaffected by tap-to-tap time. However, heat is provided by natural gas as opposed to offgas and as a result an additional cost is incurred.

One of the primary concerns with scrap preheating is that oil and other organic materials associated with the scrap tend to evaporate off during preheating. This can lead to discharge of hydro-

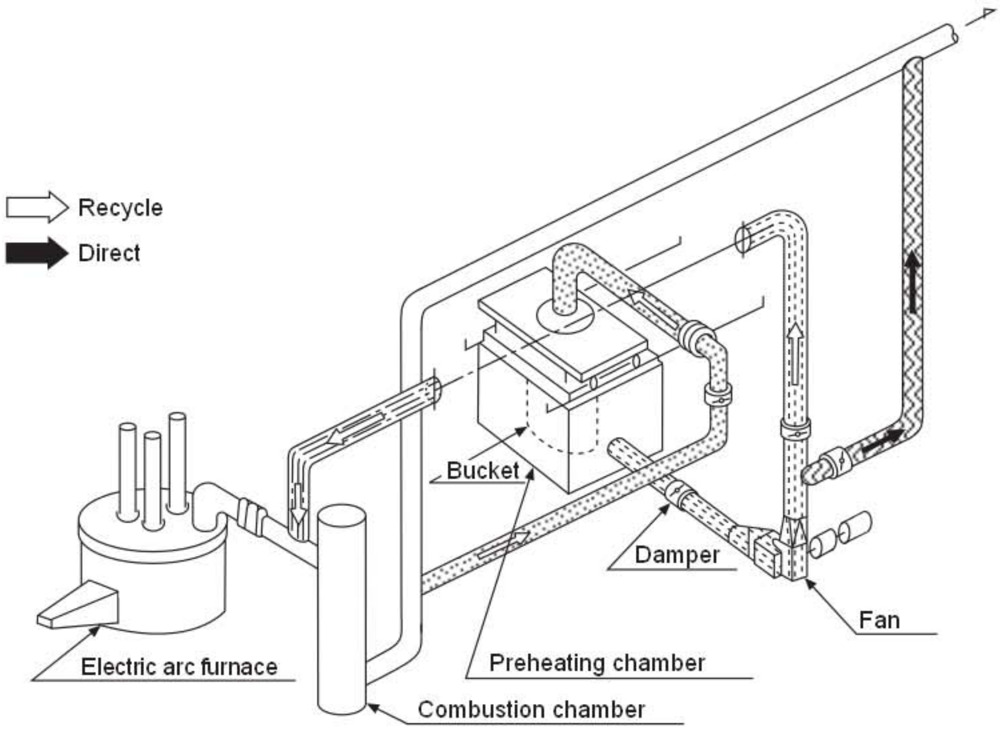


Fig. 10.77 NKK scrap preheater. *(Courtesy of NKK.)*

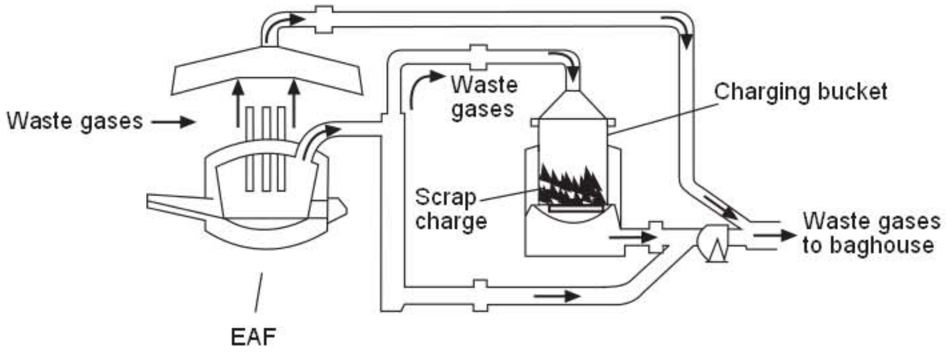


Fig. 10.78 Scrap preheating in the scrap bucket. *(Courtesy of Center for Materials Production.)*

carbons to the atmosphere and foul odors in the shop environment. In some Japanese operations, this has been remedied by installing a post-combustion chamber following scrap preheating operations. In one such operation scrap preheating is conducted in conjunction with a furnace enclosure. It is reported that power savings are 36–40 kWh/ton and electrode consumption is reduced by 0.4–0.6 kg/ton. It is not specified what the additional capital costs and maintenance costs were for this system. In North America, most operations find that the savings offered by conventional scrap preheating do not compensate for the additional handling operations and additional maintenance requirements.

10.9.4 K-ES

The K-ES process is a technology developed jointly by Klockner Technology Group and Tokyo Steel Manufacturing group. Subsequently the rights to the process were acquired by VAI. The process uses pulverized or lumpy coal in the bath as a source of primary energy. Oxygen is injected into the bath to combust the coal to CO gas. The CO gas is post-combusted in the furnace freeboard with additional oxygen to produce CO_2. Thus a large portion of the calorific heat in the process is recovered and is transferred to the bath. In addition the stirring action caused by bottom gas injection results in better bath mixing and as a result accelerated melting of the scrap. The process is shown schematically in Fig. 10.79. Fig. 10.80 shows the carbon and oxygen injectors used in K-ES. Fig. 10.81 presents oxygen consumption versus electrical power consumption.

10.9.4.1 Historical Development

The first K-ES installation was at Tokyo steel in 1986, where a 30 ton EAF was converted to run trials on the process. A productivity increase of 20% and an electrical savings of approximately 110 kWh/ton resulted. Thus the process concept was proven on an industrial scale.[18]

In 1988, Ferriere Nord in Italy decided to install the K-ES process on its 88 ton EAF in Osoppo. At that time Ferriere Nord was producing approximately 550,000 tons per year of steel on a 1975 vintage EAF that was originally designed to produce 220,000 tons per year. This had been accomplished through a combination of long arc/foamy slag practice, high oxygen utilization (32 Nm^3/ton), water-cooled furnace roof and walls and oxy-fuel burners. The first K-ES heat took place early in 1989 following the installation of post-combustion lances, coal injection equipment and bottom injection tuyeres. This process continues to operate at Ferriere Nord.

In December 1989 another K-ES facility was installed on a 82 ton oval tapping furnace at Acciaierie Venete in Padua, Italy. This operation differs in that the A phase electrode is hollow and is used for carbon injection. This installation has seven post-combustion lances that are mounted into the furnace wall, through the water-cooled panels. Five bottom tuyeres are located to form a pitch circle positioned between the furnace wall and the electrode pitch circle. The operation at

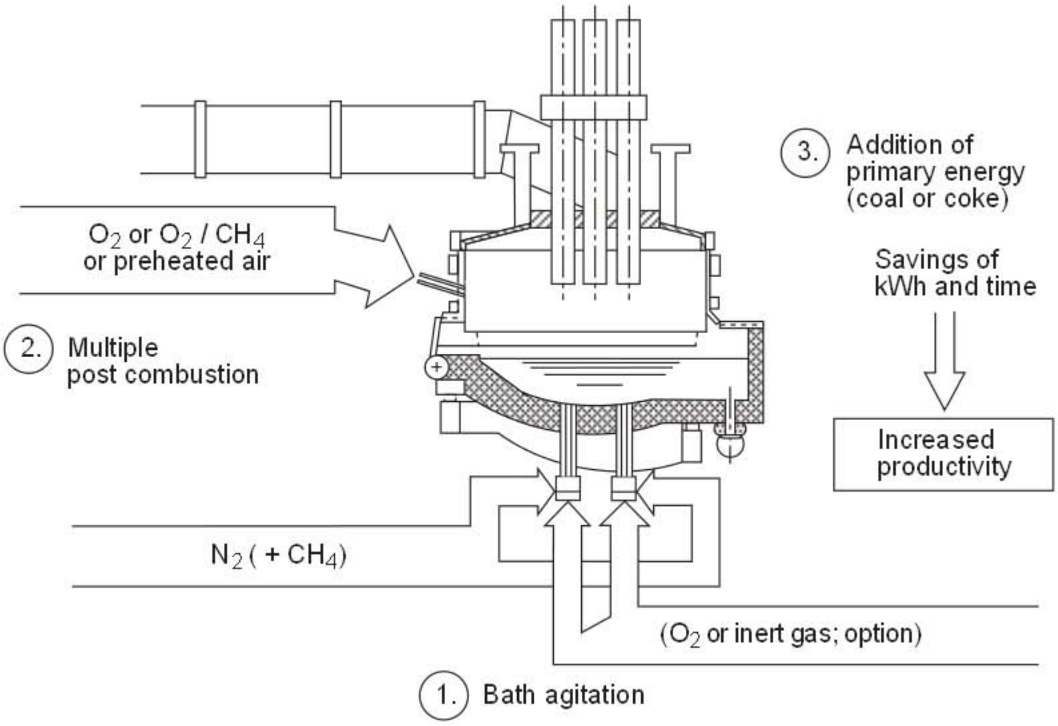


Fig. 10.79 K-ES process. *(Courtesy of Voest Alpine Ind.)*

Venete has achieved a productivity of 2.2 tons/MW. Tap-to-tap times average 54 minutes even though the transformer is only capable of supply a maximum of 45 MW.[41,46]

10.9.4.2 Results

At Tokyo Steel the average power consumption for a set of trials was 255 kWh/ton with an average coal injection rate of 24 kg/ton. This gave an electrical power replacement of 4–5.5 kWh/kg of coal injected. In addition the melting time was reduced from 60 minutes to 48–50 minutes. Ferrosilicon and ferromanganese consumption was also decreased by 1.1 kg/ton. Results are presented in Table 10.16.

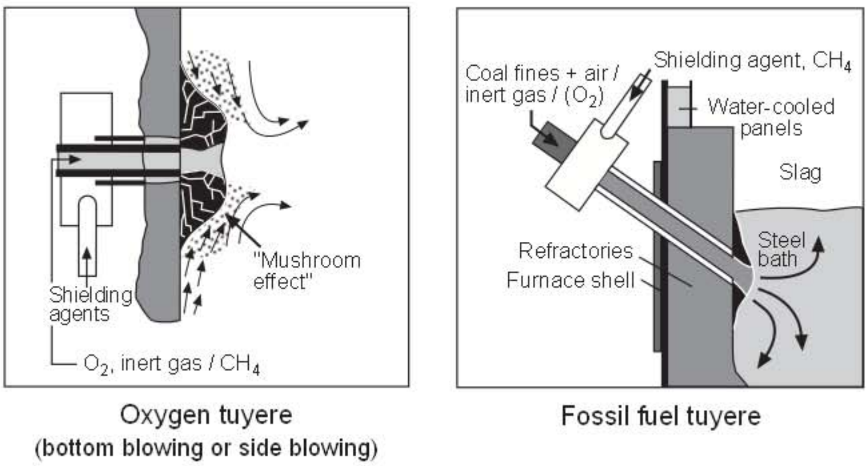


Fig. 10.80 K-ES carbon and oxygen injectors. *(Courtesy of Voest Alpine Ind.)*

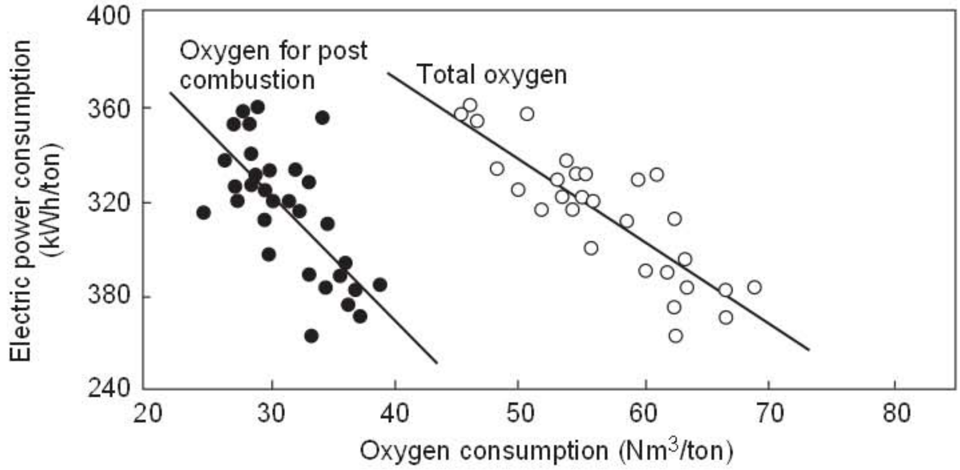


Fig. 10.81 Total oxygen consumption versus electrical power consumption for K-ES operations.

Table 10.16 K-ES Analysis Based on Tokyo Steel Results

Material	Consumption/ton liquid steel	Difference
Power on time	58 minutes	–26 minutes
Tap-to-tap time	79 minutes	–26 minutes
Production	476,000 tons/year	97,000 tons/year
Coal	32 kg/ton	+27 kg/ton
Oxygen	63 Nm^3/ton	+53 Nm^3/ton
Electricity	330 kWh/ton	–150 kWh/ton
Lime		+5.0 kg/ton
Electrodes		–0.3 kg/ton
FeMn		–1.0 kg/ton
Inert Gas	6.0 Nm^3/ton	+6.0 Nm^3/ton
Production Increase		+33%

Following installation and testing of the K-ES system at Tokyo Steel, an 88 ton furnace at Ferriere Nord was converted to K-ES operation. Lumpy coal is added in the charge and additional powdered coal is injected into the bath. Oxygen consumption is 50 Nm^3/ton. The distribution of this oxygen is as follows: 13 Nm^3/ton is injected to the bath via tuyeres, 13 Nm^3/ton is injected to the bath via an oxygen lance and 18 Nm^3/ton is injected into the freeboard through post-combustion lances. Typically carbon consumption is on the order of 20 kg/ton unless the charge is 30% hot pig metal, in which case the carbon consumption is 13 kg/ton. Electrical energy consumption decreased by 60 kWh/ton with the K-ES operation. An energy balance showed that the electrical savings were highly dependent on efficient post-combustion of the process gases in the freeboard.

Best results at Ferriere Nord were achieved at conditions of 180 kWh/ton, 53 Nm^3 O_2/ton (1850 scf O_2/ton), 30% pig iron and tap-to-tap time of 35 minutes and at conditions of 255 kWh/ton, 40 Nm^3 O_2/ton (1400 scf O_2/ton), 100% scrap and tap-to-tap time of 53 minutes.

Similar results have been obtained at Acciaerie Venete using a Fuchs OBT furnace with K-ES.

10.9.5 Danarc Process

The Danieli Danarc process combines high impedance technology with high chemical energy input to the furnace in order to achieve high productivity and energy efficiency. The first installation of this technology was at Ferriere Nord in Italy. Features for chemical energy input are very similar to K-ES. Bottom tuyeres are used to inject oxygen and carbon. Sidewall lances are also used. Post-combustion oxygen is supplied via burners. Fig. 10.82 shows a top view of the Danarc furnace at Ferriere Nord.

The purpose of tuyeres installed in the furnace bottom is to distribute the oxygen throughout the furnace in order to maximize the decarburization rate. With just oxygen lancing, the area around the lance becomes depleted of carbon and some of the oxygen reacts with iron. CO is generated mostly in one part of the furnace around the injection point. With multiple tuyeres, CO generation is spread out within the furnace and the potential for heat recovery via post-combustion is greater. Injection at several points also gives good bath mixing, an added benefit. Multiple sidewall carbon injectors allow for good control of slag foaming across the whole surface of the bath.

Natural gas, nitrogen and carbon dioxide are used as shroud gas for the oxygen tuyeres. Wear rates are approximately 0.5 mm per heat.[47,49]

Traditionally, a series reactor has been installed on the primary side of the furnace transformer to allow for high impedance operation. This allows the furnace to operate at long arc (i.e. high voltage) and low electrode current which gives high arc stability, improved power input to the bath and reduced electrode consumption.

Alternatively, the saturable reactor provides a method to reduce both current and reactive power fluctuation. This reduces the electrodynamic stresses on the furnace transformer secondary circuit and reduces the flicker level in the supply network. The main purpose of the saturable reactor is to control the reactance so as to minimize current fluctuations. The saturable reactor acts as a variable reactance controlled by the excitation current. The current control performance becomes similar to a DC furnace with thyristor controlled current. Several advantages resulting from the use of the saturable reactor are that current fluctuations and electrodynamic stresses are limited, reactive power

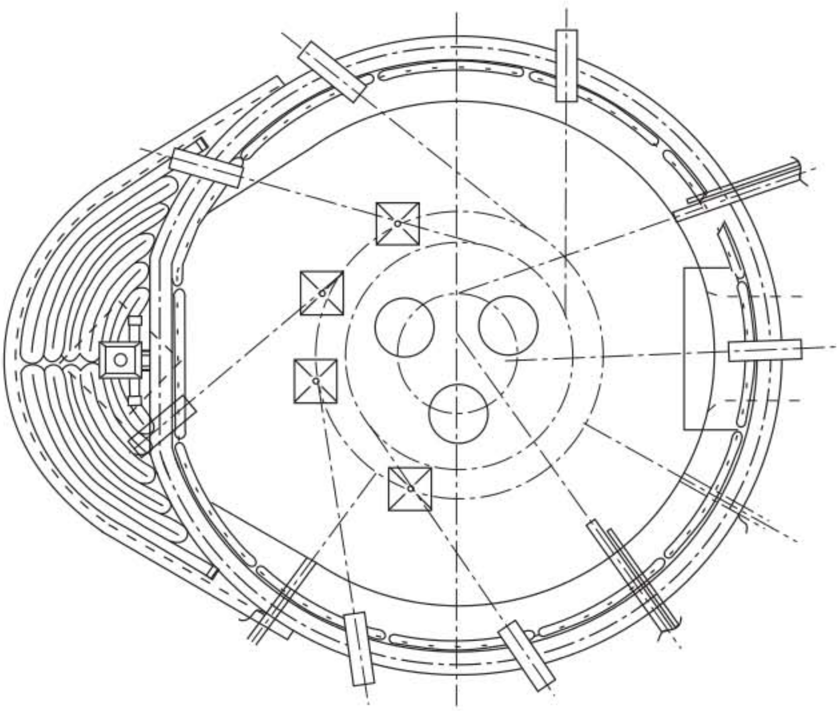


Fig. 10.82 Top view of a Danarc furnace. *(Courtesy of Danieli.)*

fluctuations are reduced resulting in less flicker, energy transfer to the melt is improved, reduced electrode consumption, and tap changing is not necessarily required.

The following results are reported for Ferriere Nord for a charge makeup of 82% scrap and 18% cold pig iron. Tap-to-tap time of 50 minutes was achieved, with an aggregate power-on time of 40 minutes. Power consumption was 270 kWh/ton liquid. Electrode consumption was 1.6 kg/ton liquid. Total oxygen consumption was 50 Nm^3/ton liquid while total natural gas consumption was 10 Nm^3/ton liquid. Total carbon introduced was 10 kg/ton liquid. The resultant tap temperature was 1640°C. For 100% scrap operation, the power consumption increases by 23 kWh/ton liquid.[47,49]

Trials have also been run using hot metal, with 70% scrap and 30% hot pig iron as part of the charge. The results were a tap-to-tap time of 45 minutes and a power-on time of 38 minutes. Power consumption was 160 kWh/ton liquid. Electrode consumption was 1.0 kg/ton liquid. Total oxygen consumption was 40 Nm^3/ton liquid and total natural gas consumption was 2.2 Nm^3/ton liquid. Air supplied to burners was 9 Nm^3/ton liquid and total carbon injected was 14.9 kg/ton liquid. Resultant tap temperature was 1680°C. [47,49]

10.9.6 Fuchs Shaft Furnace

The need to reduce the amount of power input into EAF operations lead to a prototype study of a shaft furnace at Danish Steel Works Ltd (DDS).[18,112] The concept was to load scrap into a shaft where it would be preheated by offgases exiting the EAF. The scrap sat in a column at one end of the furnace and was constantly fed into the furnace as the scrap at the bottom of the column melted away.

In January 1990, a production shaft preheater was retrofitted to one of DDS's two 125 ton EAFs. DDS stopped using the shaft after less than two years of operation. This was partly due to the problems of maintaining a suitable scrap flow in the shaft. In addition, Danish tariff regulations on electricity made a three shift per day operation uneconomical. As a result DDS could not move to a single furnace operation which would make full use of the preheating shaft.

10.9.6.1 Single Shaft Furnace

The Fuchs shaft furnace concept was installed at Co-Steel Sheerness[113,114] in England in 1992. This installation is the outcome of work done in Denmark at DET Danske Stalvalsevaerk on first and second versions of the process. The process has a shaft with a conventional oval EAF bottom with three phase AC arcs. Fig. 10.83 illustrates the shaft. The relatively short shaft is scrap bucket fed. Unlike the EOF, there are no movable fingers and the scrap descends continuously into the bath where the arcs and oxygen produce the unrefined metal. The Fuchs shaft preheater at Sheerness consists of a reverse taper (larger at bottom) shaft which sits on the furnace roof, offset from the furnace centerline.

The furnace shell is mounted on a frame and is tilted by means of four hydraulic cylinders mounted in each corner of the frame, enabling the furnace shell to be lowered by one foot to the first bucket charging position or to be tilted into the tapping or deslagging positions. The furnace runs on four tracks, two in the center and two on the outside. The furnace water-cooled roof, the shaft and the electrode gantry are fixed and do not tilt.

Energy input to the furnace is from an 80 MVA transformer, 6 MW oxy-fuel burners and a water-cooled oxygen lance.

A typical operating cycle is as follows. The heat cycle is started by charging the first basket of scrap. The furnace is lowered one foot onto a bumper and the furnace car is moved west to the charging position where the first bucket is held on a bale arm support structure. The bucket is opened remotely via claw-type actuators. The first charge is approximately 44 tons of scrap.

Once the furnace has been charged it is moved back under the roof and is raised back to the upper position. The second bucket is then charged to the shaft. The second charge is approximately 35 tons. The complete charge cycle for the first two buckets of scrap is typically less than two minutes. Meltdown commences and after approximately four minutes the electrodes are raised and the

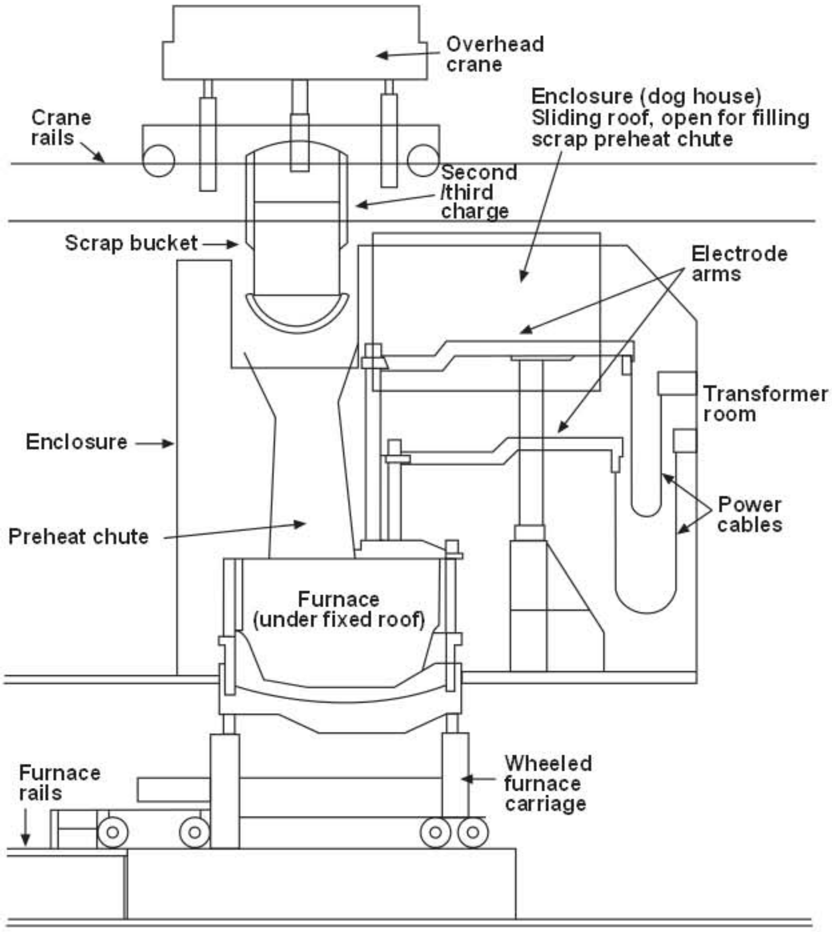


Fig. 10.83 Co-Steel Sheerness shaft furnace.

third and final scrap bucket is charged to the shaft. Meltdown then proceeds uninterrupted and the furnace taps 99 tons at 1640°C with an average power-on time of 34 minutes.

The Fuchs shaft has claimed benefits ranging from reduced EAF dust (due to dust sticking to the scrap in the shaft) to reduced exhaust fan requirements due to lower gas volumes. As all of the scrap is melting in, the zinc quickly volatizes, resulting in furnace dust that is high in zinc content. For a 120 ton furnace with an 80 MVA transformer, Fuchs predicts a tap-to-tap time of 56 minutes with an electrical power consumption of approximately 310 kWh/ton. The projections anticipate that the equivalent of 72 kWh/ton of energy will be recovered from the offgas.

The results reported by Co-Steel Sheerness PLC have indicated the following:[115]

1. Liquid steel yield, due to high FeO reversion from the slag, resulted in an average of 93.5%.
2. The volume of flue dust produced by the shaft furnace was 20% lower than for the conventional furnace (14 kg/ton billet compared to 18 kg/ton billet).
3. The flue dust chemistry was changed with shaft furnace operation. The zinc oxide content rose from 22% to 30%. In addition the lime content was decreased from 13% to 5%.
4. Fan power requirements for the fume extraction fans were decreased from 19.3 kWh/ton billet to 10 kWh/ton billet due to lower gas volumes.

Table 10.17 presents results for various trials conducted at Co-Steel Sheerness. As can be seen, the amount of burner power was increased considerably in the most recent set of trials. The burner efficiency that was achieved was much higher than that in a conventional furnace as the flame contacted cold scrap for a longer duration. In addition the CO content in the offgas was monitored in order to adjust the oxygen flow to the burners to promote CO post-combustion at the base of the shaft.[116]

Table 10.17 Results of Shaft Furnace Operating Trials at Co-Steel Sheerness

	Old Furnace	Standard Shaft Charge	Trial Shaft Charge
Electrical energy (kWh/ton billet)	467	327	250
Burner oxygen (Nm^3/ton billet)	3.0	20.0	3.2
Lance oxygen (Nm^3/ton billet)	10.9	10.0	5.0
Burner natural gas (Nm^3/ton billet)	1.5	10.0	16.0

Fuchs has other single shaft furnaces, of similar design, operating in Turkey, China, Malaysia and at North Star Steel in Arizona. The latter, Fig. 10.84, is powered with a DC transformer and is equipped with an ABB DC bottom.

Fig. 10.84 Shaft furnace in operation at North Star Steel, Kingman, Arizona.

10.9.6.2 The Double Shaft Furnace (DSF)

In order to increase the production capacity of a furnace with one transformer to over 1,000,000 tons per year, the double shaft furnace was developed. There are currently two in operation in Europe, SAM in France and ARBED in Luxembourg, Fig. 10.85. Both are 95 ton AC high impedance furnaces which, when fully utilized, deliver a liquid steel capacity of over 1,000,000 tons per year. The double shaft furnace utilizes one transformer and one set of electrodes for both furnace shells.

North Star BHP Steel Ltd. started up the first double shaft furnace in the United States. This furnace, with a tap weight of 180 tons, has a rated capacity of 1,700,000 tons per year. The AC transformer is rated at 140 MVA and will operate with a secondary voltage of 1200–1300 V. All (100%) of the scrap charged can be preheated. A savings in electric energy consumption of between 100–120 kWh/ton is projected in comparison with a single shell EAF based on 100% scrap charge. North Star BHP Steel plans to utilize DRI/HBI and/or pig iron as part of the charge. This type furnace would also be well suited for the use of iron carbide.

The trend toward the use of more fossil energy by electric arc furnace steelmakers will result in more latent energy in the offgas, primarily in the form of CO. Measurements indicate a value of energy in the offgas of 170–190 kWh/ton in a standard single shell EAF operation. In addition, instead of using energy to circulate water in the duct system to cool the hot offgases, this energy could instead be utilized to heat and melt the scrap, as is done in shaft furnaces.

The double shaft furnace at SAM, Table 10.18, currently operates with a typical scrap charge that is 25% heavy scrap, 30% shredded, 5% municipal recycled, 15% galvanized sheet, 10% turnings

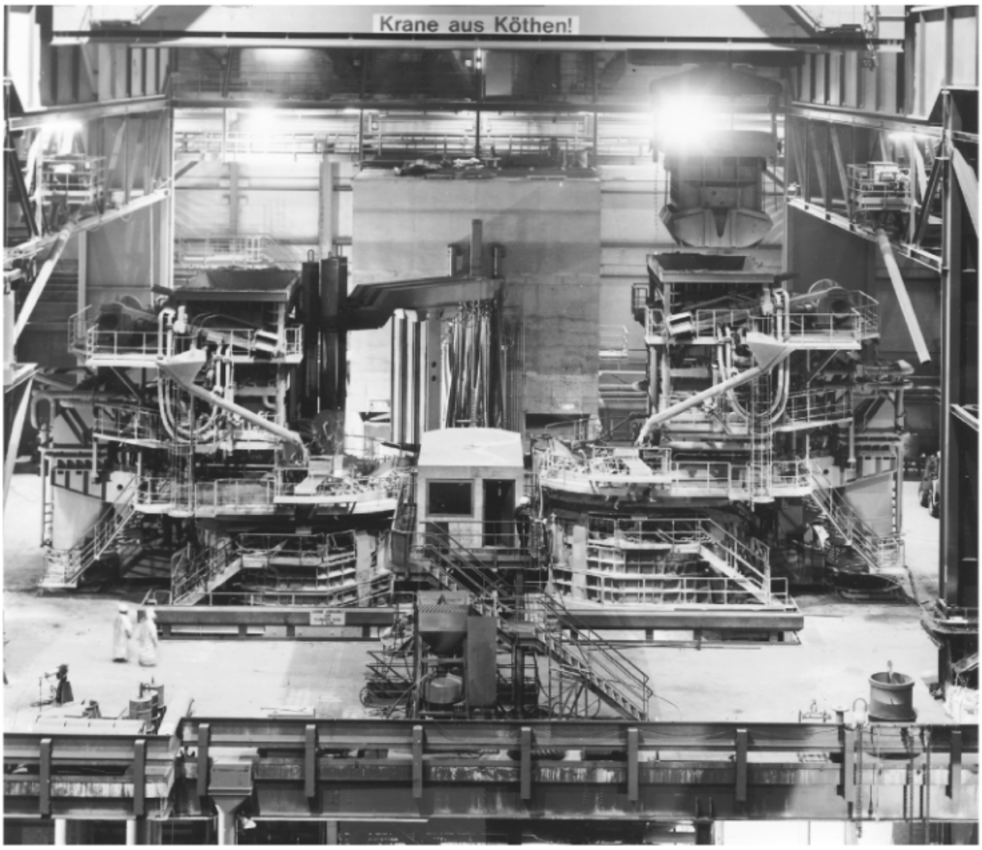


Fig. 10.85 Double shaft furnace in operation at ARBED. *(Courtesy of Fuchs.)*

and 15% light scrap. Three charge buckets are used. A round bucket holding 50 % of the charge is used to charge the furnace. Two rectangular buckets each containing 25 % of the charge are used to charge the shaft. While one shell (A) is tapping, the electrodes are moved to the other shell (B) to begin meltdown. Bore-in proceeds at a voltage of 750V. Meltdown then proceeds at 900 V. Six burners located at the bottom of the shaft aid meltdown. Vessel A, once finished tapping, is charged with 75% of the total charge. The burners in this vessel are placed on high fire to help preheat the scrap. Vessel B completes meltdown and its offgases are directed to the vessel A to aid in scrap preheating. The remaining 25% of the scrap is now charged to the shaft. Once vessel B is ready to tap, the electrodes are moved back over to vessel A to start another meltdown cycle.[117]

Table 10.18 Furnace Data of the AC Double Shaft Furnace at SAM

Furnace Capacity	115 tons
Furnace Diameter	6.3 m
Tapping Weight	99 tons
Liquid Heel	11 tons
Transformer Capacity	95 MVA
Active Power	60 MW
Electrode Diameter	600 mm
Oxy-Gas Burners	7 × 3 MW (ea)
Oxy-Carbon Lances	2 × 35 Nm^3/min
Bottom Stirring	5 elements — nitrogen

Consumption figures reported by SAM and ARBED are presented in Table 10.19 and Table 10.20, respectively.[117]

Table 10.19 Reported Consumption Figures at SAM

	Best Day	Best 4 Days
Charge Mix	Scrap 100%	Scrap 100%
Electrical Energy	320 kWh/ton	338 kWh/ ton
Electrode Consumption	1.3 kg/ ton	1.45 kg/ ton
Oxygen Lance	10.3 Nm^3/ ton	11.8 Nm^3/ ton
Oxygen Burners	11.8 m^3/ ton	12.7 Nm^3/ ton
Gas	6.5 Nm^3/ ton	6.2 Nm^3/ ton
Carbon Charge	7.5 kg/ ton	7.6 kg/ ton
Carbon Foamy Slag	6.7 kg/ ton	5.4 kg/ ton
Lime	35.2 kg/ ton	39.5 kg/ ton
Power On Time	40 min.	45.5 min.Liquid
Yield	91.5%	

Table 10.20 Reported Consumption Figures at ARES Schifflange

Electrical Energy	298 kWh/ ton
Electrode Consumption	1.23 kg/ton
Oxygen	22.0 Nm^3/ton
Gas	5.9 Nm^3/ton
Power On Time	39.2 min.

10.9.6.3 The Finger Shaft Furnace

Various studies have shown that the greatest proportion of the energy leaving the furnace in the offgas occurs during flat bath operations. This is due to the absence of scrap to absorb energy from the offgas as it exits the furnace. A large potential for energy recovery exists if the offgas can be used to preheat scrap. For operations with continuous feed of high carbon iron units (e.g. DRI or iron carbide), the amount of energy escaping in the offgas will be higher yet due to the higher levels of carbon monoxide in the offgas. Finger shaft furnace installations with tap weights ranging from 77–165 tons are located at Hylsa, Cockerill Sambre, Swiss Steel AG and Birmingham Steel.

To use the energy of the offgas efficiently and to reduce the electrical power requirements needed to obtain short tap-to-tap times, Fuchs developed the finger shaft furnace at Hylsa in Monterrey, Mexico. This furnace, tapping 150 tons, maintains a 44 ton liquid heel and is powered with a 156 MVA DC transformer utilizing the Nippon Steel water-cooled DC bottom electrode (3 billet bottom electrode). Reported consumption figures are presented in Table 10.21. The furnace charge is 45% scrap and 55% DRI charged during the 50 minute power-on time of the furnace. Half of the scrap is charged onto a water-cooled finger system in the shaft during the refining period of the previous heat. The other half of the scrap is charged onto the fingers after the first charge has been dropped into the furnace and power is applied.

Table 10.21 Reported Consumption Figures at Hylsa. *From Ref 117.*

Charge Mix	50% Scrap, 50% DRI
Electrical Energy	394 kWh/ton
Electrode Consumption	0.9 kg/ton
Oxygen Lance	20 Nm^3/ton
Oxygen Burners	8.1 Nm^3/ton
Natural Gas	3.6 Nm^3/ton
Carbon Charge	9 kg/ton
Carbon Foamy Slag	4.5 kg/ton
Power-On Time	48 min.
Tap Temperature	1620°C

The furnace roof and shaft sit on a trolley which moves on rails mounted on the slag door and tapping sides of the furnace. To initiate a heat, the electrode is raised and rotated away from the furnace. The shaft is moved into position over the furnace center and the fingers are retracted so that the preheated scrap is charged onto the hot heel. The roof is then moved back and the electrode is returned to its operating position. A second scrap charge is added to the shaft and DRI is fed continuously throughout meltdown. Once the first scrap charge is melted, the second charge is added to the center of the furnace. Initial bore-in utilizes a voltage of 400V and is increased to 550V once the electrode has penetrated to about one metre. During meltdown a voltage of 635V is used. During meltdown of the second scrap charge and refining, the first charge for the next heat is preheated in the shaft. The DRI, containing 2.3% carbon, is cold and continuously charged. Oxy-fuel burners are used during scrap meltdown. Future plans call for running the burners with excess oxygen for post-combustion of CO.

A finger shaft furnace is now operating at Swiss Steel AG (Von Roll). This furnace was converted in only five weeks. It is powered with a high impedance AC transformer. They have seen a 55% improvement in productivity and are tapping 79 tons of steel every 37 minutes. Reported consumption figures are presented in Table 10.22.

Table 10.22 Reported Consumption Figures at Swiss Steel AG. *From Ref 117.*

Charge Mix	100% Scrap
Electrical Energy	260 kWh/ton
Electrode Consumption	1.3 kg/ton
Oxygen Lance	14.4 Nm3/ton
Oxygen Burners	11.8 Nm^3/ton
Natural Gas	5.1 Nm3/ton
Carbon Charge	5.5 kg/ton
Carbon Foamy Slag	2.7 kg/ton
Power On Time	28 min.
Tap Temperature	1620°C

This operating data indicates that 30 minute tap-to-tap times are likely to be achieved in the future.

Paul Wurth S.A. has installed a 154 ton DC finger shaft furnace at Cockerill Sambre in Belgium under license from Fuchs. The unique feature of this finger shaft furnace is that 20–50% of the charge can be liquid iron with a carbon content of 4%. The operation utilizes the Paul Wurth proprietary hot metal charging system.[118]

This operation utilizes four water-cooled billets as the bottom electrode. The hot metal charging system is a technology developed by Paul Wurth based on trials conducted at various ISCOR facilities in South Africa. Hot metal is charged through a side launder in the furnace as shown in Fig. 10.86. This system allows for continuous charging of 44–60 tons of hot metal to the furnace over a period of 15–20 minutes. The furnace is located within an enclosure to minimize fume emissions to the shop. Fig. 10.87 shows the hot metal ladle charging system operating sequence

Reported consumption figures at Cockerill Sambre are reported in Table 10.23.

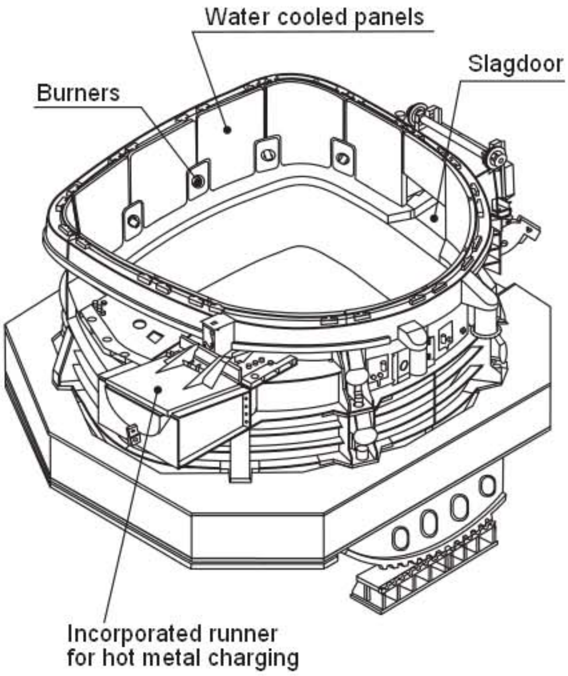


Fig. 10.86 Shaft furnace bottom with runner for hot metal charging. *(Courtesy of Paul Wurth, S.A.)*

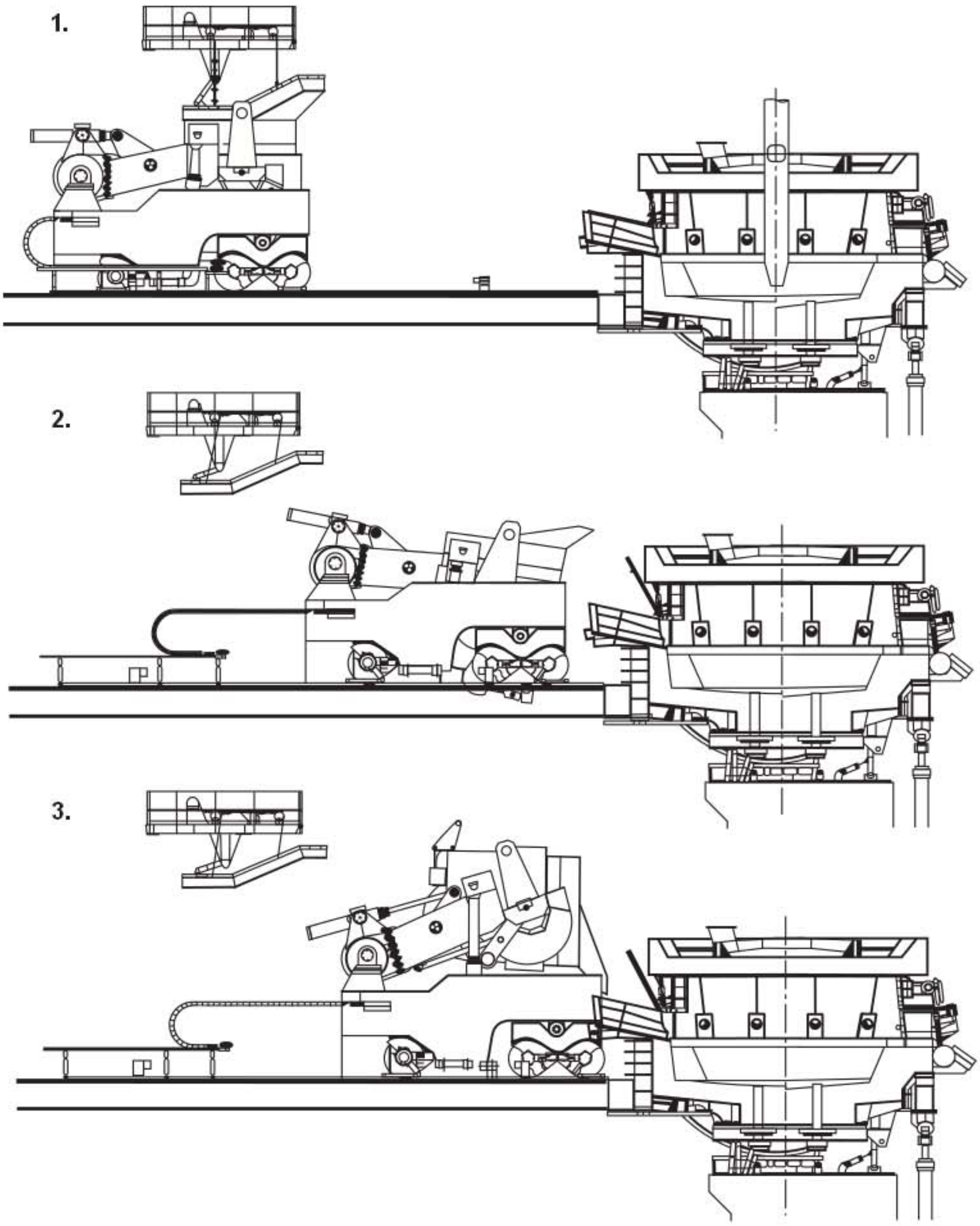


Fig. 10.87 Operating sequence to charge hot metal. 1. Ladle with cover in place. 2. Cover removed and ladle to furnace. 3. Charging hot metal. *(Courtesy of Paul Wurth, S.A.)*

A replacement equation for the equivalent value of hot metal in the furnace has been developed by Paul Wurth based on various plant trials and is presented below.[118]

$$1 \text{ ton hot metal} + 22.7 \text{ kg lime} = 0.92 \text{ ton scrap} + 45 \text{ kg coal} + 300 \text{ kWh electricity}$$

10.9.7 Consteel Process

The Consteel process was developed by Intersteel Technology Inc. which is located in Charlotte, North Carolina. This is another process that is based on recovering heat from the offgas. In this

Table 10.23 Reported Consumption Figures at Cockerill Sambre. *From Ref 118.*

	34% Hot Metal Charge	100% Scrap Charge
Electrical Energy	187 kWh/ton	290–310 kWh/ton
Power On To Tap	<40 min.	
Oxygen		23–27 Nm^3/ton
Natural Gas		5.5–7.3 Nm^3/ton
Carbon Charge		9–13.5 kg/ton
Power On Time		50–55 min.
Tap Temperature		1620°C

case, the scrap enters a long preheater tunnel and is preheated by the offgas as it travels to the furnace. The scrap is moved through the tunnel on a conveyor and is fed continuously to the EAF. The offgas flows counter-current to the scrap. The EAF maintains a liquid heel following tapping. Fig. 10.88 illustrates the key system components.

10.9.7.1 Historical Development

The Nucor Steel Corporation was the first company to commit to an industrial application of the Consteel technology. In 1985 Consteel was retrofitted into an existing Nucor plant located in Darlington, South Carolina. Due to space restrictions the installation was designed with a 90° bend in the scrap conveyor and a relatively short preheater section. This was reported to result in scrap feeding and preheating problems. The unit was operated for eighteen months and was then shut down.

Several key concepts were confirmed by this prototype which was put into operation in 1987. A consistent heel of hot metal acted as a thermal flywheel, increasing efficiencies of scrap melting. It was demonstrated that keeping the bath temperature within a proper range ensured a constant equilibrium between metal and slag and a continuous carbon boil, which resulted in a bath which was homogeneous in temperature and composition. The foaming of the slag could be continuously and precisely controlled and was very important for successful operation.

The first greenfield demonstration of the Consteel process was at AmeriSteel in Charlotte, North Carolina. This layout included a new meltshop that was built parallel to an existing one. In this case the continuous scrap feed system and the scrap preheater were all in line. The scrap was taken from

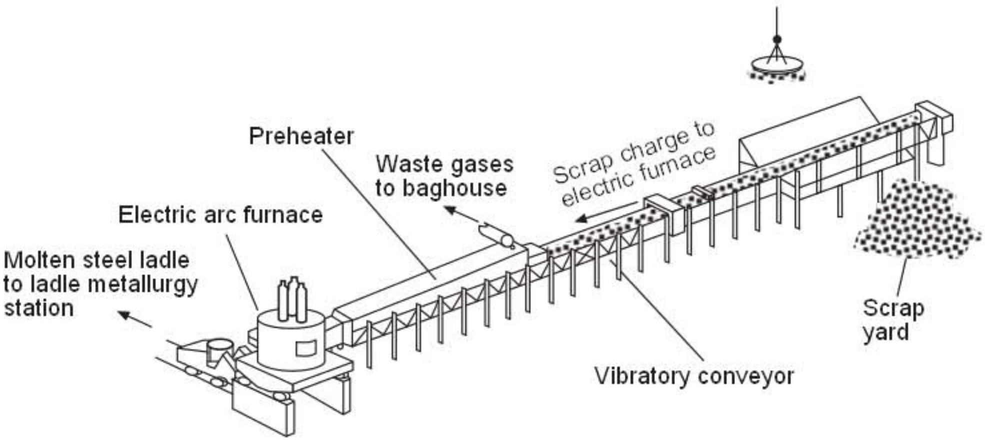


Fig. 10.88 Key components of the Consteel process.

railcars to a loading station through a preheater to a connecting car and then into the furnace. The preheater design included an afterburner for control of carbon monoxide emissions though this feature was never put into operation. The scrap preheater was designed to heat scrap up to 700°C. Particular emphasis was placed on providing a tight seal on the scrap preheater. This was accomplished by providing a water seal in the preheater. The key characteristics of this facility have been described.[119]

The Contifeeding system consists of three conveyors in cascade, each 1.5 metres (5 ft.)$\times$ 0.3 metres (1 ft.) deep $\times$ 60 metres (200 ft.) long. A refractory lined tunnel covers the conveyor and a water seal prevents outside air from leaking between the cover and the conveyor pans. Shredder scrap, #1 scrap, turnings and light structural scrap are charged to the furnace on the conveyor. A leveller bar at the scrap charge end maintains a maximum scrap height of 0.45 metres (18 in.) on the conveyor.

The scrap preheater is 24 metres (80 ft.) in length with 60 natural gas burners of 7.0 Nm^3/min. (250 scfm) capacity mounted in the preheater roof. Preheat temperatures up to 700°C are achieved using the furnace offgas and the burners. The burners were later removed as sufficient preheating was achieved using only furnace offgas.

The furnace proper is a 75 ton EBT EAF designed to tap 40 tons and retain a 30–35 ton heel. The scrap feed rate is approximately 680 kg/min. (1500 lbs/min.). For cold startup the furnace is top charged and this charge is melted in to provide the liquid heel; a slag door burner is used to accelerate meltdown for these heats. Electrical energy is supplied by a 30 MVA transformer through 20 inch electrodes, with resultant tap-to-tap times of approximately 45 minutes. Other facilities include carbon and oxygen bath injection for producing a foamy slag and a pneumatic lime injection system.

In addition to the operation at AmeriSteel, several other plants have installed Consteel at their facilities including Kyoei Steel (Nagoya), Nucor Steel Darlington and New Jersey Steel.

10.9.7.2 Key Operating Considerations

The key to obtaining good operating results with Consteel is based on controlling several operating parameters simultaneously. These are bath temperature, scrap feed rate and scrap composition, oxygen injection rate, bath carbon levels, and slag composition.

It can be seen that any type of imbalance in any of these operating parameters will have a ripple effect through the whole process. Generally, the hot heel size is 1.4 tons times the power input in MW. Some operations use process models to track the process inputs. Slag composition is very important because without good slag foaming it will be difficult to bury the arc. This will result in large heat losses and potentially furnace damage. Continuous evolution of CO from the bath is crucial to maintaining scrap preheat temperatures. Approximately 70–75% of the CO generated in the furnace is available as fuel in the preheater. A reducing atmosphere is maintained in the first 30% of the preheater in order to control scrap oxidation. Complete combustion of CO and VOCs evolving from the scrap is achieved prior to the fluegas exiting the preheater. The FeO content of the slag is maintained around 15% which is suitable for good slag foaming.

A variety of scrap types can be fed to the Consteel operation. The primary stipulation is that the size is less than the conveyor width in order to minimize scrap bridging in the feed system. Bundles can be used but will not benefit from preheating. Best results are obtained when using loose, shredded scrap which is light and has a large surface area for heat transfer from the hot offgases. High carbon scrap such as pig iron can be used but must be distributed within the charge in order to maintain a fairly uniform carbon level in the bath. HBI can be charged to the preheater along with scrap. DRI should only be charged into the section of the preheater with a reducing atmosphere. The key to good operation in all cases is to maintain homogeneity of the scrap feed so that bath composition remains within preset limits. Scrap density must also be optimized so that the desired residence time in the preheater shaft can be attained thus ensuring sufficient preheating of the scrap. Charge permeability also affects the amount of preheat achieved.

The following summary, Table 10.24, from Intersteel lists the latest data on existing and future installations.

Table 10.24 Consteel Operating Results. *From Ref 120*

Facility	AmeriSteel Charlotte	Kyoei Nagoya	Nucor Darlington	New Jersey Steel	ORI Martin Italy	N.S.M. Thailand
Start-up Date	Dec. 1989	Oct. 1992	Sept. 1993	May 1994	On Hold	End 1997
Productivity	54 ton/hr	140 ton/hr	110 ton/hr	95 ton/hr	87 ton/hr	229 ton/hr
EAF Type	AC	DC	DC	AC	AC	AC
New/Retrofit	New	New	New	Retrofit	Retrofit	New
Transformer Power	24 MW	47 MW	42 MW	40 MW	31 MW	95 MW
Power	370 kWh/ton	300 kWh/ton	325 kWh/ton	390 kWh/ton		
Oxygen	22 Nm^3/ton	39 Nm^3/ton	33 Nm^3/ton	23 Nm^3/ton		
Electrodes	1.75 kg/ton	1.14 kg/ton	1.0 kg/ton	1.85 kg/ton		
Yield	93.3	94	93	90		

Note that some of the above figures represent unit consumption records.

10.9.8 Twin Shell Electric Arc Furnace

10.9.8.1 Historical Development

One of the new technologies that seems to be generating much interest is the twin shell furnace. Essentially this technology is similar to conventional scrap preheating with the exception that scrap preheating actually takes place in a furnace shell as opposed to a scrap bucket.

The original work done on in-furnace scrap preheating was in Sweden, where SKF operated an installation which had a single power supply and two furnace shells. In the early 1980s Nippon Steel developed a twin shell process for the production of stainless steel, Fig. 10.89. At the current time, several furnace manufacturers are producing twin shell furnaces. The key goals of a twin shell operation are equal to those of other developing technologies, but in addition the cycle times are similar to those for BOF operations due to the minimization of power-off times.

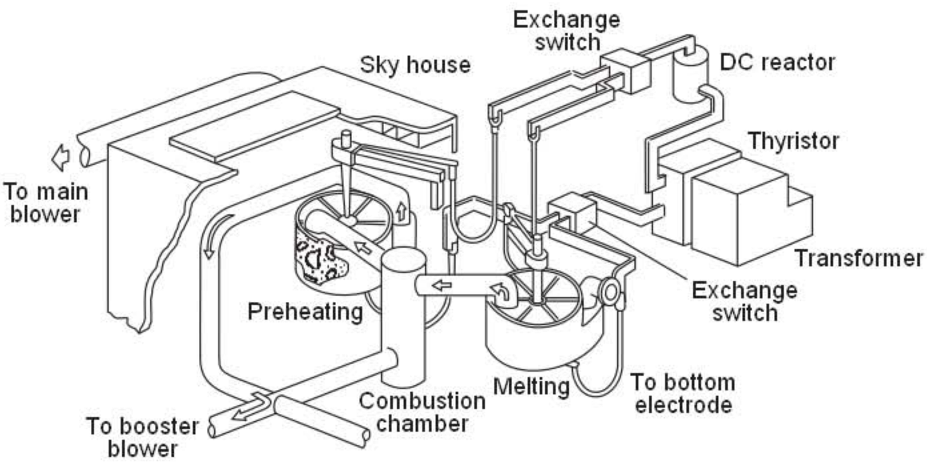


Fig. 10.89 Schematic of the DC twin shell EAF at Nippon Steel.

10.9.8.2 Process Description

This type of operation consists of two furnace shells and one set of electrode arms that are used alternately on one shell and then the other. The scrap is charged to the shell that is not melting and the scrap is preheated by the offgas from the melting furnace. Supplemental burner heat can also be used. The result is that the scrap is preheated in the furnace prior to melting. The more the scrap is preheated, the greater the energy savings.

Some early designs proposed a gantry that moved along a rail from one shell to the other. Most designs now use a furnace roof and electrode arms that can swing between two positions for the shells. Generally a twin shell installation will consist of two identical vessels with a lower shell, an upper shell and a roof and one set of electrode arms and lifting supports with one conventional power supply.

It is interesting to note that several of the recent orders for twin shell installations have opted for DC operation. This obviously has some inherent benefits as there is only one electrode arm to rotate between the two operating positions. Another variation introduced is that some operations have an electrode mast and arm for each furnace and have swithgear allowing the sharing of a common transformer.

There have been several operating modes suggested for the twin shell operation. The Nippon Steel operating cycle consists generally of two phases; scrap preheating and melting. The furnace is only charged one time and thus the heat size is smaller than the actual capacity of the furnace. In the case of the Nippon Steel installation at POSCO, the operation is a two charge operation where 60% of the total charge is preheating during the melting phase on the second furnace. Thus the second charge (40% of the total) is not preheated in the furnace.

The operating cycle proposed by Clecim is similar to that at POSCO. The Mannesmann Demag operating cycle is quite different in that the operation uses two charges but the power is alternated between furnaces between charges. Thus one furnace melts its first charge while the other preheats its first charge. Once the first charge is melted, the second charge is dropped and the power is diverted to the first preheated charge. Thus the second charge is preheated as well as the first during the preheating cycle for the second charge; oxygen lancing or oxy-fuel burners can also be used to accelerate preheating and melting. This operating method maximizes the recovery of heat from the offgas but requires very precise process control to keep from freezing the molten heel in the furnace during the preheat phase.

10.9.8.3 Operating Results

10.9.8.3.1 Nippon Steel In this operation, there is only one charge per heat. While one charge is being melted, a second charge is being heated in the second shell. Combustion gas from a burner on the second shell is mixed with offgas from the first shell to give a constant temperature of 1650°F for preheating in the second vessel. The remaining offgas from both shells is used to preheat scrap in a scrap bucket. The net electrical power input requirement is reported at 260 kWh/ton which is 29% lower than for the two conventional furnaces that were replaced by the twin shell operation.

10.9.8.3.2 Davy-Clecim The Clecim installation at Unimetal Gandrange tapped its first heat in July 1994. This twin shell operation is DC with the Clecim water-cooled billet bottom electrode. The shells have a nominal capacity of 165 tons with a nominal diameter of 7.3 metres. Each shell has a working volume of approximately 200 m^3 which enables the furnace to operate with a single charge each heat. The rated installed transformer capacity is 150 MVA. The maximum secondary voltage is 850V. Maximum power input is 110 MW. Each shell has four bottom billet electrodes. The graphite electrode swings between the two furnaces as required. Each shell has a manipulator with consumable oxygen and carbon injection lances. Initially, integrated steelmaking operations continued to operate which led to large furnace delays due to the logistics of movements within the

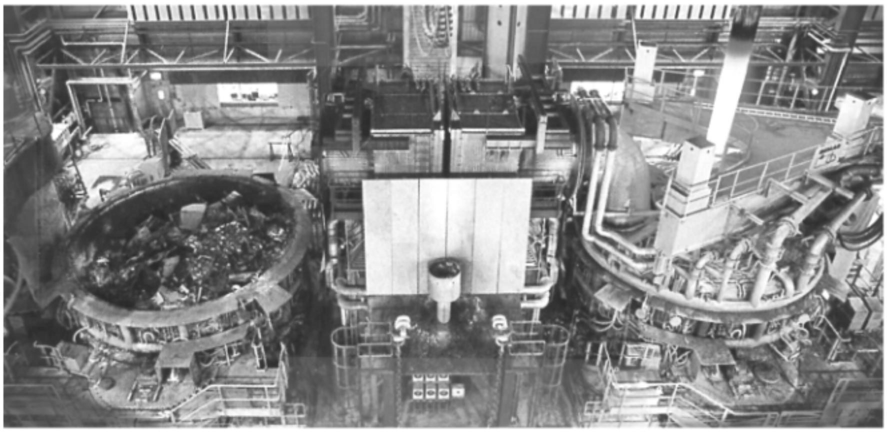


Fig. 10.90 Twin shell furnace operation at ProfilArbed.

shop. This operation experienced problems with arc flare due to the configuration of the anode bus tubes. The configuration was altered and this problem was eliminated.

The ProfilArbed installation started up at the end of 1994 and is shown in Fig. 10.90.

Several recent North American twin shell installations include Steel Dynamics, Gallatin Steel, Tuscaloosa Steel and Nucor-Berkeley. Fig. 10.91 shows the operation at Tuscaloosa Steel.

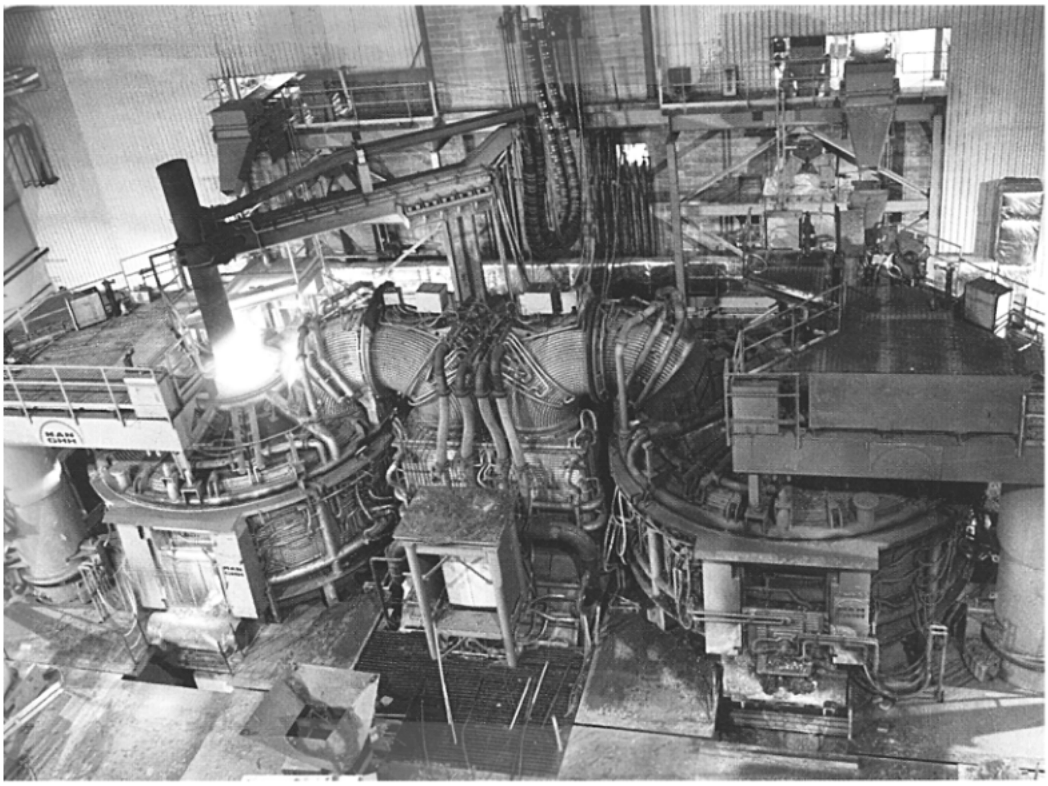


Fig. 10.91 Twin shell furnace operation at Tuscaloosa Steel.

10.9.9 Processes Under Development

Based on the success of several of the scrap preheat technologies (Consteel, Fuchs Shaft Furnace, EOF), several new shaft type technologies are currently in the development stage. These are outlined in this section.

10.9.9.1 Ishikawajima-Harima Heavy Industries (IHI)

IHI is currently developing a shaft type preheat furnace based on twin electrode DC technology. The first commercial installation has started up at the Utsunomiya plant of Tokyo Steel. The DC furnace itself is oval in shape with two graphite electrodes and two bottom electrodes consisting of conductive hearth brick (as per ABB's DC furnace design). There are two DC power supplies which are individually controlled. The power feeding bus is arranged so that the two arcs will deflect towards the center of the furnace, thus the energy of the arcs will be concentrated at the center of the furnace and the thermal load to the furnace walls will be low compared to a conventional furnace. As a result refractory walls are used instead of water-cooled panels thus reducing heat loss. Scrap is charged to the furnace between the electrodes. The furnace will maintain a large hot heel (110 ton heel, 140 ton tap weight) so that uniform operating conditions can be maintained (similar to the Consteel concept). Steel is tapped out periodically via a bottom taphole in the furnace. A diagram of the furnace is given in Fig. 10.92.

The scrap charging system consists of two main components, the preheat chamber and the charging equipment. The scrap is fed into the upper part of the chamber from a receiving hopper. The exhaust gas from the furnace flows up through the chamber, preheating the scrap. In the pilot plant, scrap preheat temperatures as high as 800°C were achieved. Gas exit temperatures from the chamber were as low as 200°C. At the base of the preheat chamber are two pushers. These operate in two stages, allowing scrap to feed into the furnace at a constant rate. Offgas leaves the top of the preheat chamber and flows to a bag filter. Some gas can be recycled to the furnace to regulate the inlet gas temperature to the preheater.

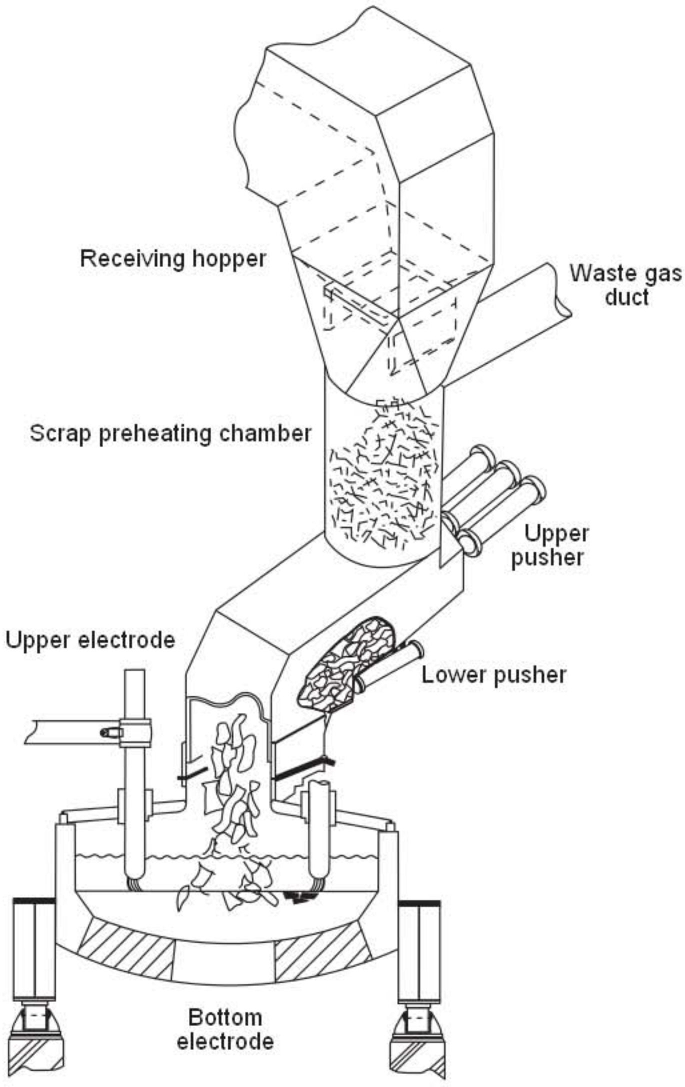


Fig. 10.92 IHI shaft furnace.

10.9.9.1.1 Operation Scrap is fed continuously to the furnace until the desired bath weight is achieved. This is followed by a short refining or heating period leading to tapping of the heat. Power input is expected to be almost uniform throughout the heat. Most furnace operations will be fully automated. Charging of scrap into the preheater will be fully automated based on the scrap height in the chamber. Carbon and oxygen injection will be controlled based on the depth of foamy slag.

The results presented in Table 10.25 have been reported for the IHI shaft furnace.[121]

Table 10.25 Reported Results at Tokyo Steel.

	Utonomiya Plant	Takamatsu Plant
Tap-to-tap time	60 min.	45 min.
Power-on time	55 min.	40 min.
Power consumption	236 kWh/ton liquid	236 kWh/ton liquid
Electrode consumption	1.0 kg/ton liquid	1.0 kg/ton liquid
Oxygen consumption	28 Nm^3/ton liquid	25 Nm^3/ton liquid
Carbon consumption	27 kg/ton liquid	25 kg/ton liquid

10.9.9.2 VAI Comelt

VAI has been working on a new furnace design they call Comelt. This furnace features a shaft and four individually moveable graphite electrodes which protrude into the furnace at an angle. A bottom anode is installed in the hearth center. The power supply is DC. The lower part of the shaft contains openings for lime and coke additions via a bin system. The upper part of the shaft contains a lateral door for scrap charging and an opening through which the furnace offgases flow. Scrap is charged to the shaft via a conveyor belt.

The furnace consists of a tiltable furnace vessel and a fixed shaft which are connected with a movable shaft ring. The furnace employs eccentric bottom tapping. The whole vessel structure rests on a tiltable support frame. The upper part of the vessel, the shaft and shaft ring are all lined with water-cooled panels. The shaft is enclosed with a steel structure which sits on a carriage. The shaft can be moved off the furnace shell via this carriage. The shaft ring provides a tight connection between the furnace vessel and the shaft. Key furnace components are shown in Fig. 10.93.

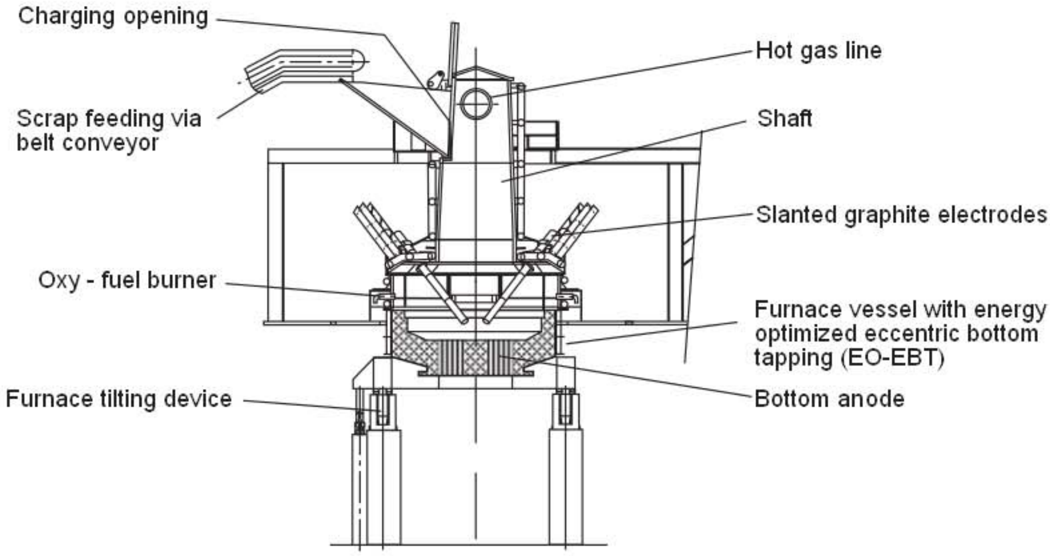


Fig. 10.93 Key components of the VAI Comelt furnace.

10.9.9.2.1 Process Description Following tapping, the furnace is charged through the shaft with up to 80% of the total charge weight, lime and carbon. After charging, the charging door in the top of the shaft is closed and the electrodes are moved into working position. Oxy-fuel door burners are used to clear an area for insertion of oxygen lances which immediately begin to inject oxygen into the heel. Additional lances higher up in the furnace provide post-combustion oxygen. As the scrap melts in and the shaft begins to empty, additional scrap is fed to the shaft. The electrode arrangement allows the flow of gases to penetrate down into the scrap and then rise up through the scrap column thus maximizing heat recovery.

VAI has projected that the capital cost for Comelt will be greater than that for an equivalent DC operation for small to medium heat sizes. This is due to the large investment required for the electrical systems for the electrodes. For large heat sizes however, the projections indicate that Comelt should be less expensive than conventional DC technology.[122]

Though this process is still in the pilot plant stage, projections indicate that this furnace will have lower power (304 kWh/ton liquid) and electrode (0.8 kg/ton) consumption than either conventional AC or DC furnaces.[122]

10.9.9.3 Mannesmann Demag Huettentechnik Conarc

The MDH Conarc is based on an operation which combines converter and EAF technologies, namely a <u>con</u>verter <u>arc</u> furnace. This technology is based on the growing use of hot metal in the EAF and is aimed at optimizing energy recovery and maximizing productivity in such an operation. The use of hot metal in the EAF is limited by the maximum oxygen blowing rates which are dependent in turn on furnace size. The basic concept of Conarc is to carry out decarburization in one vessel and electric melting in another vessel. The Conarc system consists of two furnace shells, one slewable electrode structure serving both shells, one electric power supply for both shells, and one slewable top oxygen lance serving both shells

Thus one shell operates in the converter mode using the top lance while the other shell operates in the arc furnace mode. The first order for a Conarc was placed by Nippon Dendro Ispat, Dolvi, India. This operation will be based on the use of hot metal, scrap, DRI and pig iron. Projections for this operation are an energy consumption below 181 kWh/ton when operating primarily with hot metal and DRI. Once complete, this facility will contain two independent twin shell furnaces each with a tap weight of 164 tons. Fig. 10.94 shows the key system components for a Conarc facility.

A Conarc has also been ordered for the Saldahna Steel facility which is being built in Saldahna Bay, South Africa. This operation will be based on 45% Corex hot metal and 55% DRI feed to the furnaces. Hot metal will be charged to one shell and the top lance will be used for decarburization. Simultaneously, DRI will be added to recover the heat generated. Once the aim carbon level is achieved, the electrodes will be moved to this shell and additional DRI will be fed to make up the balance of the heat.

Large quantities of heat are generated during the oxygen blowing cycle. As a result it is important that charge materials be added during this period so that some of this energy can be recovered. This will also help to protect the furnace shell from overheating.

10.9.9.4 Mannesmann Demag Huettentechnik Contiarc

The Contiarc EAF is a technology that has been proposed by MDH. It is based on a stationary ring shaft furnace with a centrally located DC arc heating system. Scrap is fed continuously by conveyor to the ring shaft. There the scrap is picked up by a series of magnets and is distributed evenly throughout the ring shaft area. The magnet train runs on a circular track below the furnace roof. Fig. 10.95 shows the main components of the Contiarc.

The scrap settles in the ring shaft and is preheated as furnace offgas rises up through the furnace. The descending scrap column is monitored using a level measuring system. If the charge sinks unevenly, additional scrap is fed to the lower points to restore an even scrap height profile. Scrap

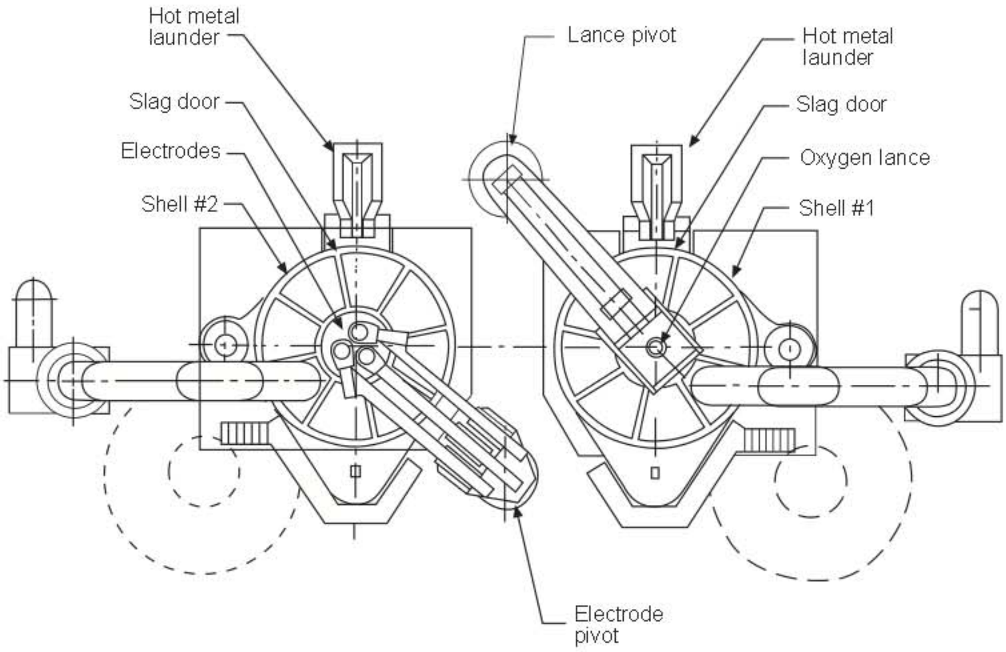
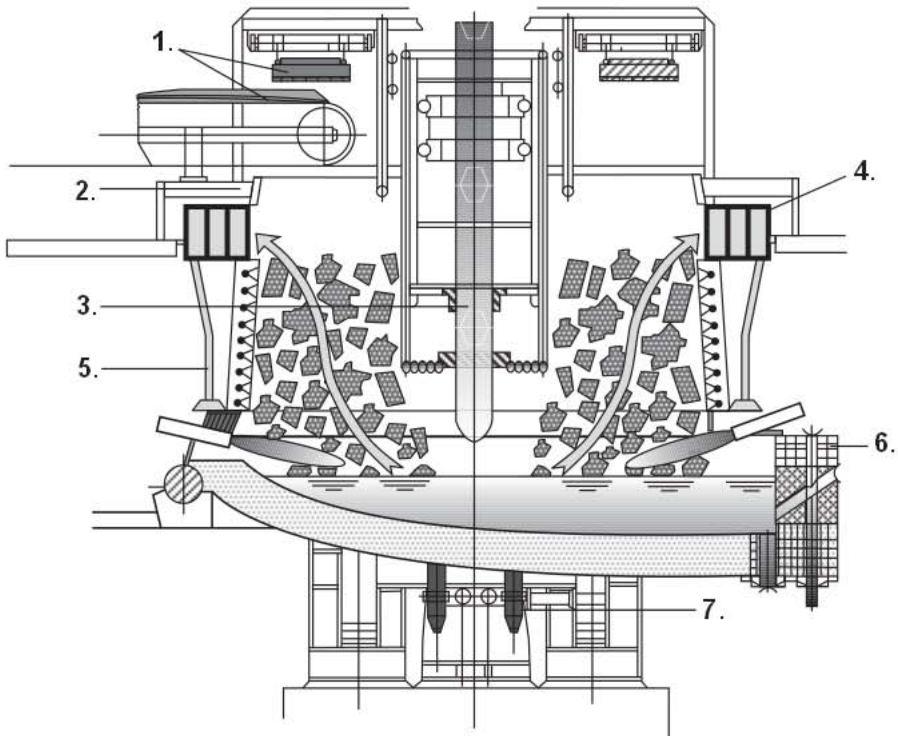


Fig. 10.94 Schematic of the Conarc process components.



1. Scrap feeding and distribution system
2. Inner and outer ring type shell
3. Graphite electrode with guiding and lifting device
4. Offgas pipe
5. Fume hood
6. Syphon tapping system
7. Bottom electrode

Fig. 10.95 Elements of the Contiarc process.

is always present to protect the furnace walls so the arc can run continuously at maximum power without fear of damage to the furnace sidewalls.

The graphite electrode is located in a protective inner vessel to protect it from falling scrap. The inner vessel has a wear guard and a cooling water system in the lower hot zone. An electrically insulated ceramic electrode bushing is provided at the point where the electrode penetrates out into the furnace. Electrode regulation is performed hydraulically.

Chemical energy input, such as burners, is highly efficient because gas residence times in the Contiarc are greater than for conventional furnaces. Burners are located near the taphole to superheat the steel prior to tapping. Burners are also located near the slag door to facilitate slagging off. Slag free tapping is performed intermittently using a syphon tapping system.

Energy efficiency is extremely high using Contiarc as heat losses to the shell are minimized due to the continuous cover provided by the scrap. The elimination of top charging also reduces heat losses. The furnace is essentially airtight so that all offgas rises up through the scrap and is collected at the ring header at the top of the shaft. Dust losses are reduced 40% over conventional EAF operations as the scrap acts as a filter, trapping the dust as the offgas rises through the scrap. MDH projections indicate that the total energy input requirement for Contiarc will be only 62% of that required by an equivalent conventional EAF operation.[123]

10.9.9.5 Future Melting Furnace Design

Without doubt, current trends in EAF design indicate that high levels of both electrical and chemical energy are likely to be employed in future furnace designs. As so many have pointed out, we are headed towards the oxy-electric furnace. The degree to which one form of energy is used over another will be dependent on the cost and availability of the various energy forms in a particular location. Raw materials and their cost will also affect the choice of energy source. The use of alternative iron sources containing high levels of carbon will necessitate the use of high oxygen blowing rates which equate to high levels of chemical energy use. A word of caution is in order though for such operations, as some thought must enter into how to maximize energy recovery from the offgases generated. In addition, high levels of materials such as cold pig iron can sometimes lead to extended tap-to-tap times which will reduce furnace productivity. If hot metal is used, some method must be provided to recover energy from the offgases. Materials which are also high in silicon will provide additional energy to the bath but at the cost of additional flux requirements and greater slag quantities generated.

A closed furnace design appears to be imperative for complete control of reactions in the furnace freeboard. Minimization of the offgas volume exiting the furnace will help to minimize offgas system requirements. If air infiltration is eliminated, offgas temperatures will be quite high which will give more efficient heat transfer in scrap preheating operations. If high levels of hydrogen and CO are present in the cooled gas, it could be used as a low grade fuel. Alternatively, if the CO is post-combusted in several preheater stages, maximum energy recovery to the scrap could be achieved.

The main disadvantage to operation of a closed furnace is that conventional oxygen lancing and coal injection operations would be difficult to carry out. These can however be carried out in a closed furnace if a series of submerged and side injectors are installed in the furnace shell and sidewalls similar to those used in conventional BOFs. The required injection rates would likely require that the furnace bath depth be increased in order to minimize blow through. A deeper bath with intense mixing will be beneficial for slag bath mixing which should improve steel quality and maximize recovery of iron units. Generation of CO in the bath will help to flush nitrogen and hydrogen out of the bath.

Some form of scrap preheating should be utilized to recover heat from the offgas. The best method for scrap preheating is still under debate but staged preheating coupled with post-combustion in each stage could help to minimize the need for a secondary post-combustion system downstream in the offgas system.

If a slanted electrode configuration similar to that used in Comelt is employed, it would be conceivable to retract the electrodes if necessary during part of the tap-to-tap cycle. This would allow preheated scrap to be charged periodically if a finger type preheat shaft were used. Alternatively the scrap could be continuously charged into a shaft by conveyor or a preheat tunnel conveyor similar to that used in Consteel, could be installed. A DC arc could be directed at the point where scrap feed falls into the furnace similar to the IHI furnace concept. This would help to maximize heat transfer from the arc to the scrap. If a scrap buildup on the walls of the furnace can be maintained, heat losses to the furnace shell can also be minimized.

10.9.9.6 Conclusions

There are many new processes for steelmaking which are now being commercialized. In almost all cases the goal is to minimize the electrical energy input and to maximize the energy efficiency in the process. Thus several technologies have attempted to maximize the use of chemical energy into the process (EOF, K-ES, LSF, etc.). These processes are highly dependent on achieving a pseudo-equilibrium where oxygen has completely reacted with fuel components (carbon, CO, natural gas, etc.) to give the maximum achievable energy input to the process. Other processes have attempted to maximize the use of the energy that is input to the furnace by recovering energy in the offgases (Fuchs shaft furnace, Consteel, EOF, IHI Shaft). These processes are highly dependent on good heat transfer from the offgas to the scrap. This requires that the scrap and the offgas contact each other in an optimal way.

All of these processes have been able to demonstrate some benefits. The key is to develop a process that will show process and environmental benefits without having a high degree of complexity and without affecting productivity. There is no perfect solution that will meet the needs of all steelmaking operations. Rather, steelmakers must prioritize their objectives and then match these to the attributes of various furnace designs. It is important to maintain focus on the following criteria:

1. To provide process flexibility.
2. To increase productivity while improving energy efficiency.
3. To improve the quality of the finished product.
4. To meet environmental requirements at a minimum cost.

With these factors in mind, the following conclusions are drawn :

1. The correct furnace selection will be one that meets the specific requirements of the individual facility. Factors entering into the decision will likely include availability of raw materials, availability and cost of energy sources, desired product mix, level of post furnace treatment/refining available, capital cost and availability of a trained workforce.

2. Various forms of energy input should be balanced in order to give the operation the maximum amount of flexibility. This will help to minimize energy costs in the long run, i.e. the capability of running with high electrical input and low oxygen or the converse.

3. Energy input into the furnace needs to be well distributed in order to minimize total energy requirements. Good mixing of the bath will help to achieve this goal.

4. Oxygen injection should be distributed evenly throughout the tap-to-tap cycle in order to minimize fluctuations in offgas temperature and composition. Thus post-combustion operations can be optimized and the size of the offgas system can be minimized. In addition, fume generation will be minimized and slag/bath approach to equilibrium will be greater.

5. Injection of solids into the bath and into the slag layer should be distributed across the bath surface in order to maximize the efficiency of slag foaming operations. This will also enable the slag and bath to move closer to equilibrium. This in turn will help to minimize flux requirements and will improve the quality of the steel.

6. Submerged injection of both gases and solids should be maximized so that the beneficial effects of bath stirring can be realized. Slag fluxes could be injected along with oxygen. The high temperatures achieved in the fireball coupled with the high oxygen potential will help the lime flux into the slag faster, and as a result, dephosphorization and desulfurization operations can proceed more optimally.

7. If injection of solids and gases is increased, it will likely be beneficial to increase the bath depth. This will also be beneficial for steel quality. For operations using high levels of solids injection (those feeding high levels of iron carbide), a deeper bath will help to reduce blow through (solids exiting the bath) and could also allow higher injection rates.

8. The melting vessel should be closed up as much as possible in order to minimize the amount of air infiltration. This will minimize the volume of offgas exiting the furnace leading to smaller fume system requirements. In the case of post-combustion operations coupled with scrap preheating, the gas volume will be minimized while maximizing offgas temperature for efficient heat transfer to the scrap. If secondary post-combustion is required it can be achieved most effectively at minimum cost by minimizing the volume of offgas to be treated.

9. If high carbon alternative iron sources are used as feed in scrap melting operations, some form of post-combustion is imperative in order to recover energy from the high levels of CO contained in the offgas. For these operations, energy recovery will likely be maximized by coupling post-combustion with some form of scrap preheating.

10. Scrap preheating provides the most likely option for heat recovery from the offgas. For processes using a high degree of chemical energy in the furnace, this becomes even more important, as more energy is contained in the offgas for these operations. In order to maximize recovery of chemical energy contained in the offgas, it will be necessary to perform post-combustion. Achieving high post-combustion efficiencies throughout the heat will be difficult. Staged post-combustion in scrap preheat operations could optimize heat recovery further.

11. Alternatively, operations which generate high levels of hydrogen and carbon monoxide may find it cost effective to try to recover the calorific energy contained in the offgas in the same manner as some BOF operations which cool and clean the gas prior to using it as a low grade fuel. For this type of operation, it will be necessary to operate with a closed furnace. To minimize fuel gas storage requirements, generation of CO in the furnace should be balanced throughout the cycle. This can be achieved by using side shell or bottom oxygen injectors and by injecting oxygen at constant levels throughout the heat. If this form of operation is coupled with scrap preheating, any VOCs resulting from the scrap will form part of the offgas stream. A greater scrap column height can be utilized because secondary post-combustion downstream in the offgas system is not necessary. The high scrap column will also help strip the fume and dust from the offgas stream thus minimizing cleaning requirements. The resulting fume system requirement will be considerably smaller than that for a conventional furnace with similar melting capacity.

12. In the future, it will likely be necessary to recycle furnace dust and mill scale back into the steelmaking vessel. Preliminary trials indicate that this material is a good slag foaming agent. Key drawbacks are handling the dust prior to recycle and its concentration buildup of zinc and lead over time.

13. Twin shell furnace configurations will continue to be used in order to reduce cycle times below 45 minutes (comparable to BOF operations) and to optimize power-on time. In the case of operations using a high percentage of hot metal, operations similar to Conarc will be used, thus employing oxygen blowing into a hot heel with

scrap addition. Thought must be given to maximizing recovery of energy from the hot offgases generated during periods of high oxygen injection rates.

14. The use of hot metal in the EAF will increase as more high tonnage EAF operations attempt to achieve cycle times of 35–40 minutes. Hot metal should be added to the heat gradually so that large fluctuations in the bath chemistry do not occur which could result in explosions. In addition, high decarburization efficiency can be achieved if the bath carbon level is kept at approximately 0.3% until final refining takes place (blowing down the desired tap carbon level at the end of the heat). Decarburizing from high carbon levels has been shown to generate high iron losses to fuming. At very low bath carbon levels, large amounts of FeO are generated and report to the slag.

15. DC furnaces will continue to be of interest for high efficiency melting operations. The additional space available on the furnace roof due to a single electrode operation will allow for much greater automation of operating functions such as bath sampling, temperature sampling, solids/gas injection, etc. The concept of using side slanted electrodes as in the case of Comelt affords several advantages to furnace operation and could allow for electrodes to be retracted in some phases of the of the tap-to-tap period. This would allow scrap to be charged from a preheating shaft without fear of electrode damage. The use of DC furnaces might also allow for smaller furnace diameters coupled with greater bath depth.

16. Operations which desire maximum flexibility at minimum cost will result in more hybrid furnace designs. These designs will take into account flexibility in feed materials and will continue to aim for high energy efficiency coupled with high productivity. For example operations with high solids injection, iron carbide or DRI fines, may choose designs which would increase the flat bath period in order to spread out the solids injection cycle. Alternatively, a deeper bath may be used so that higher injection rates can be used without risk of blow through.

17. Operating practices will continue to evolve and will not only seek to optimize energy efficiency in the EAF but will seek to discover the overall optimum for the whole steelmaking facility. Universally, the most important factor is to optimize operating costs for the entire facility and not necessarily one operation in the overall process chain.

Along with added process flexibility comes greater process complexity. This in turn will require greater process understanding so that the process may be better controlled. Much more thought consequently must enter into the selection of electric furnace designs and it can be expected that many new designs will result in the years ahead. As long as there is electric furnace steelmaking, the optimal design will always be strived for.

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