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Research and Development

Control of Copper Smelter Fugitive Emissions

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May 1980

CONTROL OF COPPER SMELTER FUGITIVE EMISSIONS

by

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DISCLAMIER

FOREWORD

When energy and material resources are extracted, processed, converted, and used, the related pollutional impacts on our environment and even on our health often require that new and increasingly more efficient pollution control methods be used. The Industrial Environmental Research Laboratory - Cincinnati (IERL-Ci) assists in developing and demonstrating new and improved methodologies that will meet these needs both efficiently and economically.

This report evaluates potential solutions for the collection of fugitive emissions from copper smelters. A brief estimate of emission rates has been provided based on available sampling reports. The results of this investigation will enable EPA to identify potential control technology applications not currently used by the domestic industry and to conduct engineering testing and evaluation on these technologies. Questions or comments regarding this report should be addressed to the Metals and Inorganic Chemicals Branch of the Industrial Environmental Research Laboratory in Cincinnati.

David G. Stephan
Director
Industrial Environmental Research Laboratory
Cincinnati

ABSTRACT

This report presents the results of a study of fugitive emission controls for the copper industry. The study was conducted by PEDCo Environmental, Cincinnati, Ohio, under contract to the United States Environmental Protection Agency.

During the study period, many of the domestic primary copper smelters were visited to investigate current controls being used and potential applications which might be proposed. The purpose of the study was to document improved controls as well as identify avenues for new research where controls are currently unavailable.

The report presents existing and proposed emission control devices for many of the furnaces commonly used in the primary copper industry, as well as for hot metal transfer devices such as launders and ladles or other transport devices. Several of the devices that are reported are currently under study by the Nonferrous Metals and Minerals Branch to determine their control effectiveness on both the pilot and full-scale application.

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FIGURES (continued)

FIGURES (continued)

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ACKNOWLEDGMENT

This report was prepared under the direction of Mr. Timothy W. Devitt. Mr. L. Yerino was the Project Manager. Project Officer for the U.S. Environmental Protection Agency was Mr. A.B. Craig, Jr., of the Industrial Environmental Research Laboratory, Cincinnati.

The helpful suggestions from plant officials of the copper smelting facilities and Mr. Henry Dolezal of the U.S. Bureau of Mines are appreciated.

The report was written by Mr. L. Yerino, Mr. T.K. Corwin, and Mr. R. Price. The cost and energy calculations were computed by Mr. L. Yerino and Mr. M. Giordano.

SECTION 1

INTRODUCTION

This report deals with fugitive emissions from copper smelting and with emission control measures. The PEDCo study involved evaluation of the controls now used in the copper smelting industry and development of suggestions for alternative control devices and practices.

A brief overview of copper smelting processes (Section 2) is followed by a more detailed analysis of the conventional processes, identifying the portions of the operating cycle that produce fugitive emissions (Section 3). Emphasis is placed on the Peirce-Smith converter, which is one of the major emission sources in copper smelting.

Section 4 describes some of the alternative process systems now in limited use in the United States and in other countries. These newer types of furnaces and converters, although usually designed primarily to facilitate production, also reduce the generation of fugitive emissions and therefore may be regarded as possible means of emission control. Section 5 summarizes the fugitive emissions from both conventional and alternative copper smelting processes; the values are based the small amount of usable published data.

The balance of the report concerns emission control measures. The devices and practices in current use are considered, along with proposed modifications that might enhance control efficiency (Section 6). Finally, some alternative control systems are presented (Section 7); these include potential adaptations of equipment now used in other industries and also changes in the smelting process, ranging from introduction of relatively simple process devices to major process changes. Potential problems associated with operation of new equipment, especially in retrofit installations, are considered, along with the potential benefits in reduction of fugitive emissions. Tables indicating order-of-magnitude costs of current and alternative control measures are given in Appendix A.

SECTION 2

OVERVIEW OF COPPER SMELTING PROCESSES

The copper produced by the domestic primary copper industry is recovered mainly from sulfide ores containing a variety of minerals. Small amounts of copper are also recovered from oxide ores, low-grade waste, and imported ores. Because most of the domestic ore is mined in the southwestern states, most of the plants are located in that area. Figure 1 shows the locations of the 16 primary copper smelters in the United States. The smelting processes recover copper while removing most of the impurities from the copper ore. Refining removes the remaining impurities.

Copper is recovered from copper ores primarily by pyrometallurgical processing; some hydrometallurgical processing is done also. Pyrometallurgical processes convert ore concentrate into an impure copper called blister copper. The process steps may consist of roasting or drying, smelting, converting, and fire refining. The anode copper product, which may contain as much as 99.8 percent copper, is sent to an electrolytic refinery for final purification.

The ore concentrate, containing about 25 percent copper, is generally conveyed to the smelter by rail or truck and stored. From that point, different process steps in various combinations are used at different smelters. The following brief process description deals with smelting in a reverberatory furnace, and the Peirce-Smith converter, the most common process in domestic use today.

From storage the ore concentrate is conveyed to a dryer or roaster. After processing, the dried or partially roasted (calcined) ore concentrate is usually transferred in a larry car to a reverberatory furnace, into which it is charged through pipes and/or hoppers located at the top or along the side walls. The concentrate is melted by reverberatory heating. With the addition of a silica fluxing agent, the melt forms a copper-bearing matte layer and a waste slag layer. The matte is tapped near the bottom of the furnace, and the lighter slag is tapped at a higher elevation. The slag is collected in a slag pot or ladle, carted to a disposal area, and dumped.

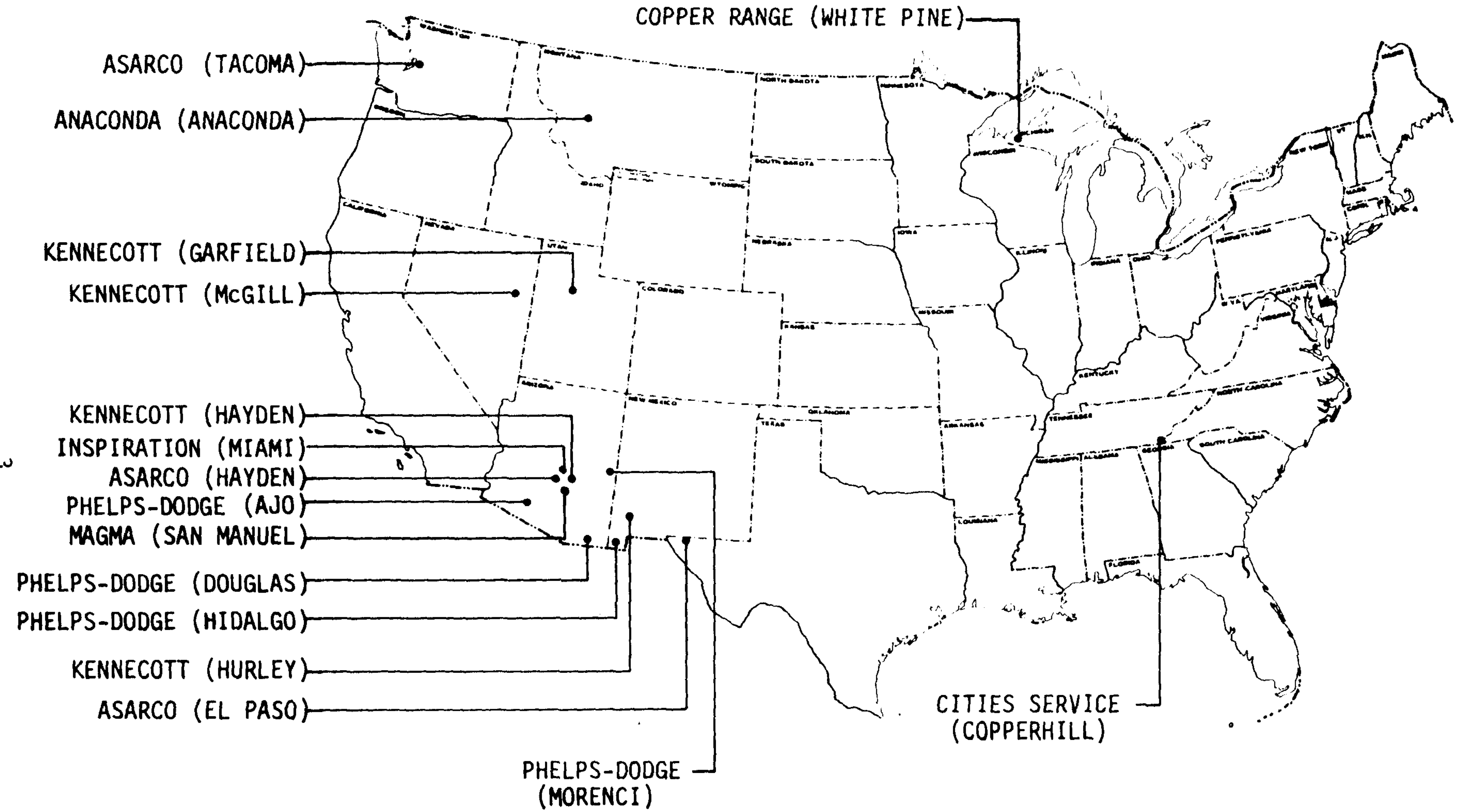


Figure 1. Locations of primary copper smelters in the United States.

The matte is collected in a ladle and is transferred by crane to a Peirce-Smith converter, which is a horizontal cylindrical furnace. This furnace converts the matte into blister copper and slag by reaction with air blown in from below the top of the bath line. The reaction is exothermic. The slag, which contains recoverable copper, is poured from the mouth of the converter into a ladle, then returned to the reverberatory furnace. After the final blowing, the blister copper contains about 98 to 99 percent copper. It is poured into a ladle, then transferred to an anode furnace, usually gas-fired, which completes the smelter refinement of copper. The anode copper, containing over 99.5 percent copper, is poured from the furnace into molds or a continuous casting wheel.

The anode copper molds range in weight from 209 to 454 kg. They are cooled by quenching, then stored or loaded onto rail cars for delivery to a refinery. Figure 2 is a general flowsheet representing the copper industry in the United States. The figure shows the traditional pyrometallurgical steps just described, as well as alternative processes currently in limited use.

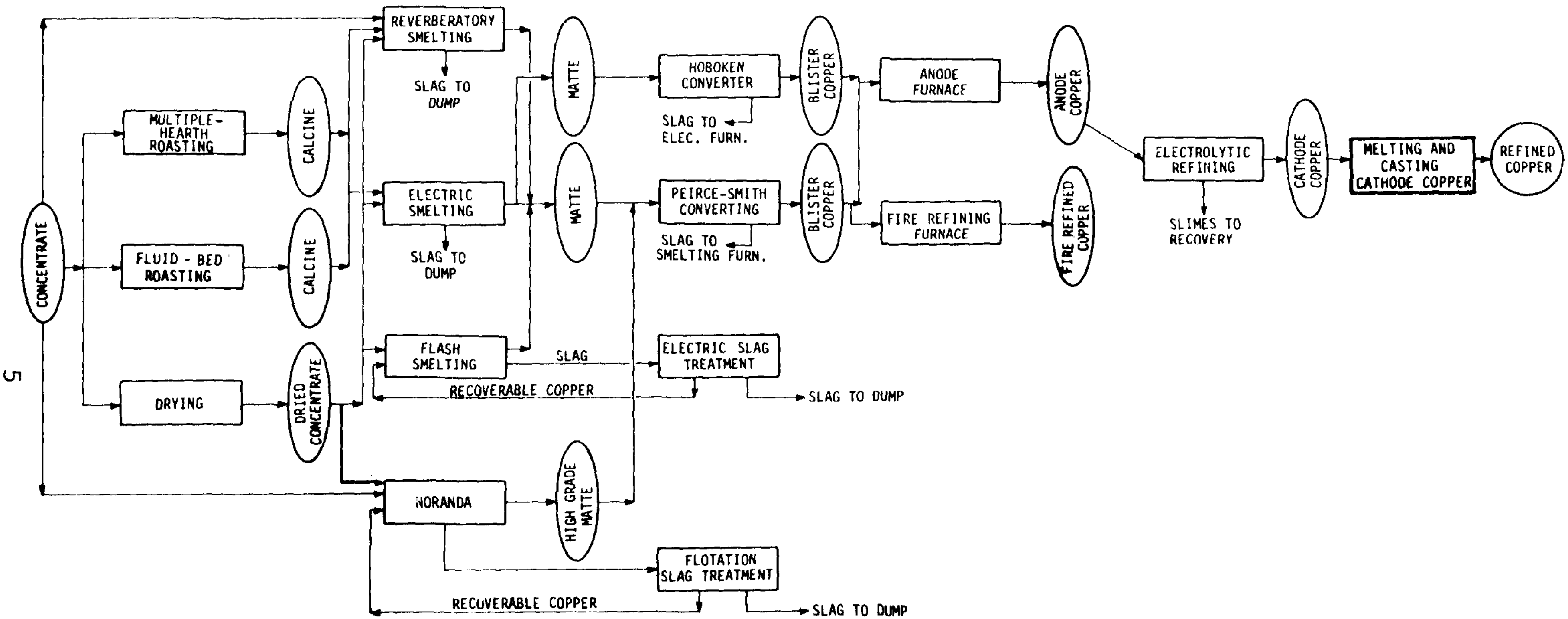


Figure 2. General flow sheet of the copper industry in the United States.

SECTION 3

BASIC COPPER SMELTING PROCESSES AND RELATED EMISSIONS

DRYERS AND ROASTERS

Some smelters use rotary dryers or roasters to precondition the concentrate before smelting. At least seven smelters either dry their concentrates onsite or process feed that has been dried at a concentrator plant. The main functions of dryers and roasters are to remove moisture and some impurities from the concentrate and to preheat the feed to the smelting furnace(s).

Figure 3 shows a countercurrent, direct-heat rotary dryer.[1] "Green" concentrate is charged to the dryer through the feed chute. By means of lifting flights, rotation of the dryer, and declination of the unit from the feed end to the discharge end, the concentrate is dried and moved to the point of discharge. The degree of drying achieved depends on the residence time of the concentrate in the dryer and the temperature throughout the dryer. Fugitive emissions occur at the discharge and charging ends during upsets or when concentrate is charged improperly.

Roasters used at domestic smelters are of two varieties: multiple-hearth and fluid-bed. Among the domestic smelters, four plants have multiple-hearth and four have fluid-bed units. In addition to removing moisture from charged concentrates and preheating them for charging to a smelting furnace, a roaster also serves the important function of partially removing some sulfur from the concentrate to give a working balance of copper, sulfur, and iron in the calcine product. Basically, sulfur is removed by converting it to SO_2 gas. This is done by maintaining control of temperature and air in the roaster so that the sulfur will ignite and burn (oxidize). The roasting process also oxidizes iron in the concentrate to ferric oxide, which can react with silica and be removed as slag in the smelting furnace. Roasting also helps to volatilize impurities such as arsenic and antimony, and thus facilitates their removal downstream.

The conventional multiple-hearth roaster, illustrated in Figure 4, is cylindrical and vertical and has from seven to twelve hearths. The casing is steel, lined with refractory brick. Concentrate is charged onto the top hearth, on which rabble arms, driven by a central shaft, move the feed. The rabble blades or "plows" push the concentrate toward the center

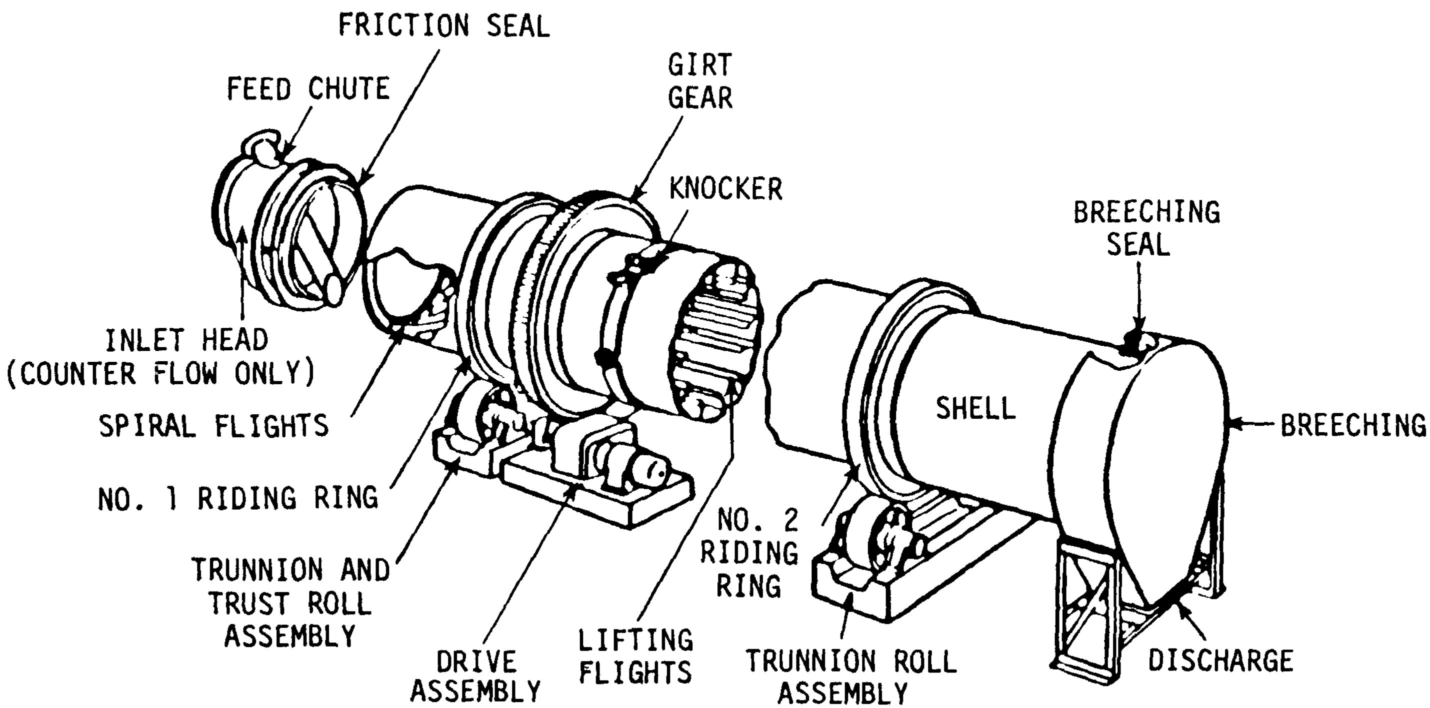


Figure 3. Countercurrent direct-heat rotary dryer
(combustion chamber not shown). (Adopted from Ref. 1.)

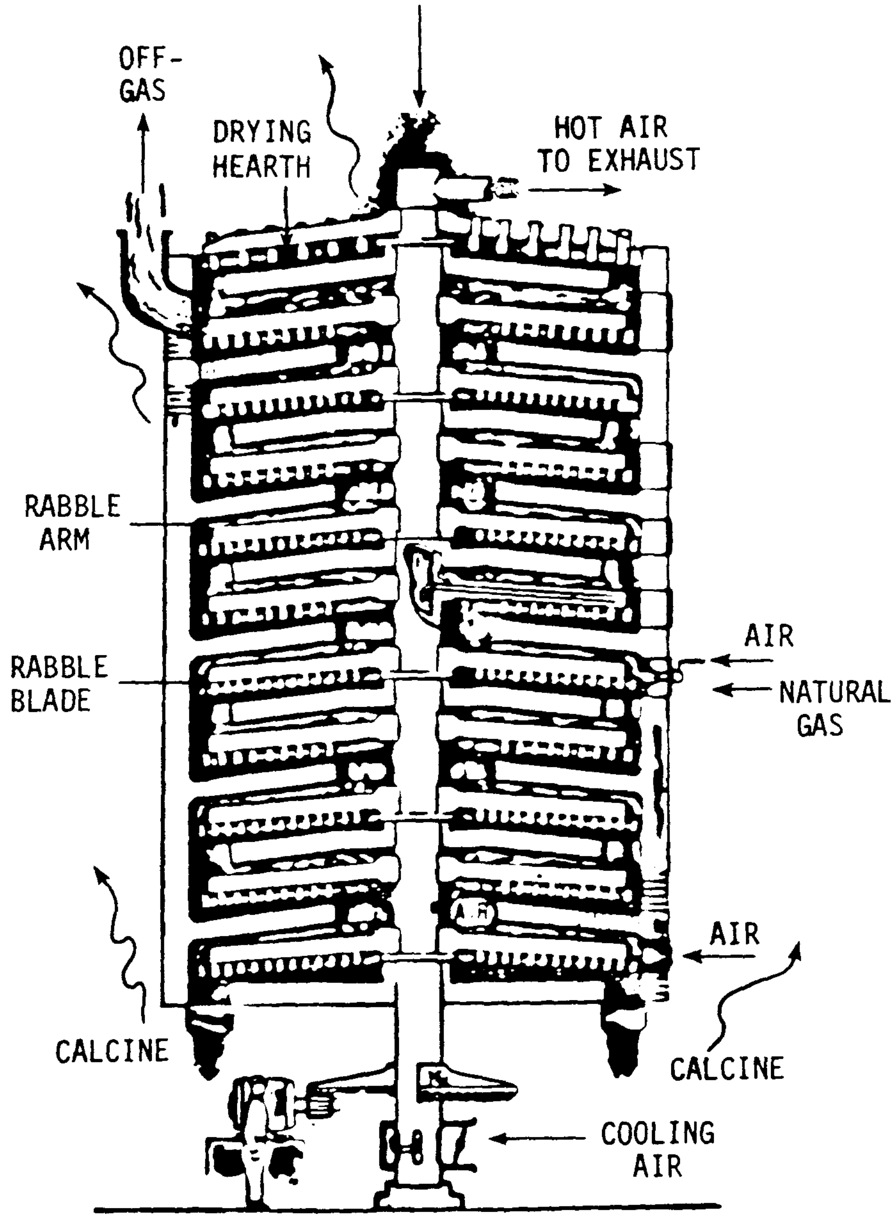


Figure 4. Multiple-hearth roasting furnace.

of the hearth, from which it cascades through an opening to the hearth below. Plows on the next hearth push the concentrate to the periphery, from which it falls to the hearth below, and so on. This gradual movement of concentrate back and forth and down through the hearths exposes the surfaces of the concentrate and permits the partial roasting of calcine to take place.

Off-gases from a multiple-hearth roaster are approximately 150° to 200°C. Particulates in the gas stream are usually collected by an electrostatic precipitator; the outlet gas containing some SO_2 and volatilized compounds is ducted to a stack.

The fluid-bed roaster, illustrated in Figure 5, is cylindrical and performs the same function as the multiple-hearth unit, but works on an entirely different principle. Rather than being roasted on hearths, the concentrate particles are suspended by an air stream moving upward. Each particle of the suspended "bed" is in constant agitated motion and is in intimate contact with the air stream. Because this fluidization roasting exposes much more of the overall surface area of the concentrate, the reactions are almost instantaneous. Also, oxygen in the air within the roaster completely reacts with sulfur and iron; thus the outlet gas contains a higher concentration of SO_2 than that generated by the multiple-hearth roaster. This concentrated SO_2 stream is sent to an acid plant for SO_2 removal.

Fugitive emissions from the roaster occur by leakage through the shell or open ports and during the filling of the transfer car.

REVERBERATORY FURNACE

The workhorse of the U.S. copper industry is the reverberatory furnace (reverb), which was first introduced in 1879 and is still used in a modified form at 11 of the 16 domestic smelters (Figure 6). The reverb is an arch-roofed or suspended-roof horizontal chamber, approximately 35 m long and 10 m wide. Heat is supplied by fossil-fuel-fired burners located at one end of the furnace. The reverb receives the charge from the roaster or dryer and, with heat supplied from the burners, reduces the charge to matte and slag. The reverb is extremely flexible with respect to concentrate composition and is capable of accepting as much as 1800 Mg of material per day.

Operation

Although methods vary considerably, reverbs are generally charged either through the furnace top or along the top portion of the side walls. Belt slingers (high speed conveyors), hoppers, and Wagstaff guns (inclined chutes) are used to distribute

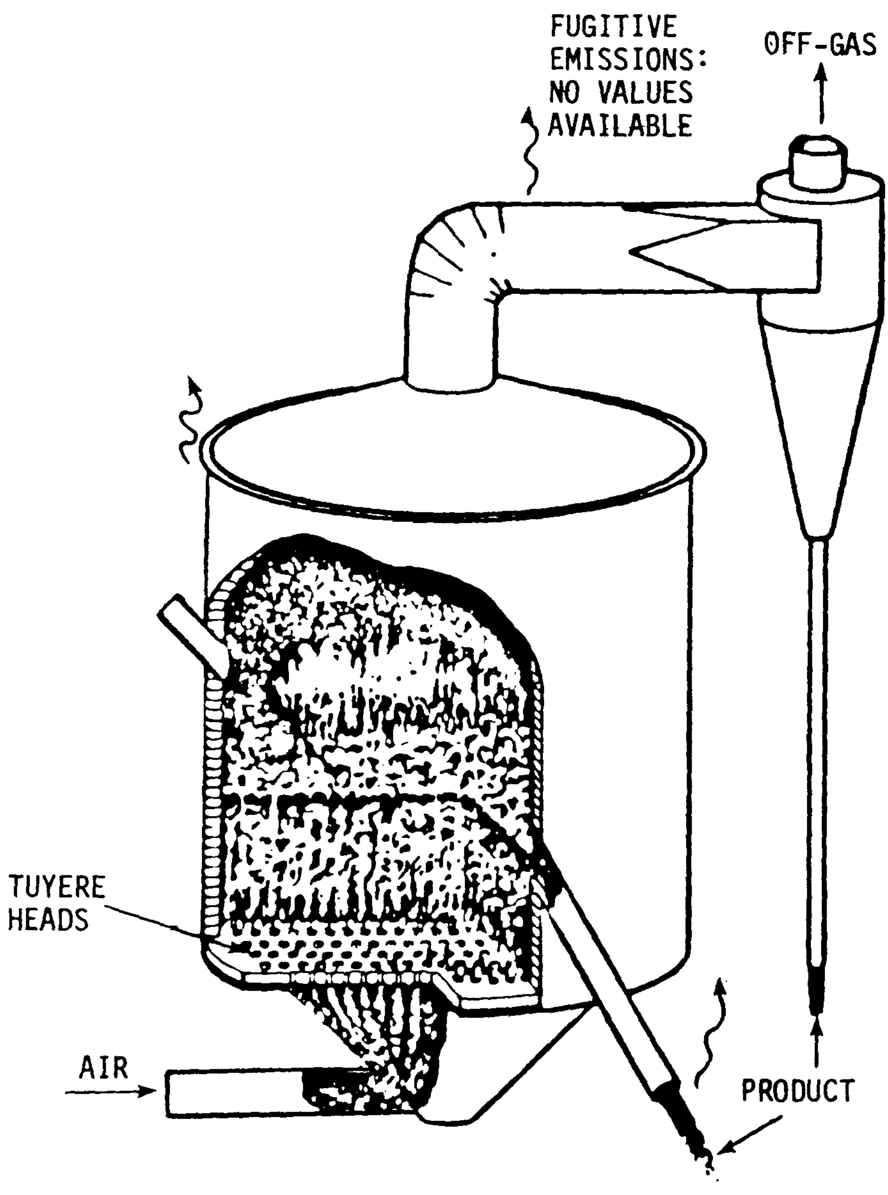


Figure 5. Fluid-bed roaster.

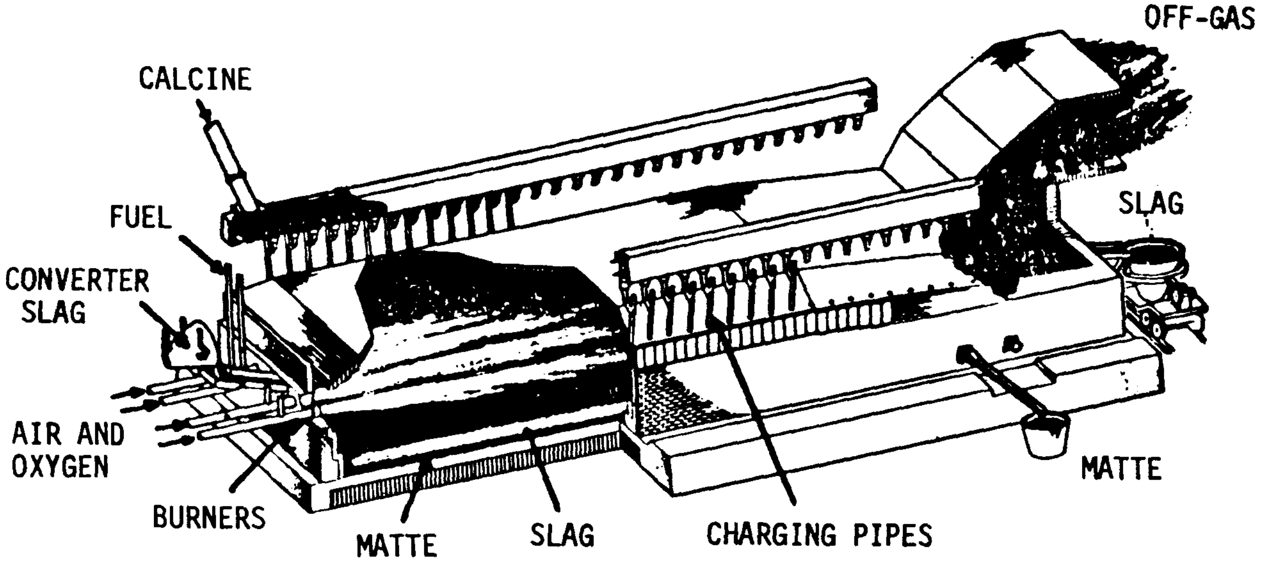


Figure 6. Reverberatory smelting furnace.

the charge over the molten bath. Drag chains and screw conveyors have also been used for charging. One plant processes "wet, unroasted" concentrate charged to the furnace by means of charge cans.

In operations at this plant, the concentrate is conveyed from a filter plant, and lime rock is added. An electric overhead travelling (EOT) crane places the charge can above one of the slingers (high-speed belt conveyors), which are adjustable in height and angle. One of the retractable charging port doors on the side of the furnace is raised, and feed from the can is discharged onto the furnace bath area. Slag is drained periodically from one end of the furnace and conveyed by launders to slag pots. The slag can be cooled, solidified, granulated, or dumped molten. Matte is withdrawn periodically through tap holes in the lower furnace wall. The matte flows down launders and into ladles, which are conveyed by overhead cranes to the converter. Outlet gases from the reverb are generally passed through waste heat boilers to recover as much of the heat of the combustion gas as possible. Usually, the gases are cleaned of most of the particulates by means of electrostatic precipitators and vented to the atmosphere.

Emissions

The fugitive emissions from or in the vicinity of a reverberatory furnace occur at openings in the furnace brickwork (caused by inadequate repair and maintenance or length of time in service); during charging of calcine or green concentrate; during addition of converter slag to the furnace; at the slag and matte launders during tapping operations; and by leakage at the uptake and the waste heat boiler.

PEIRCE-SMITH CONVERTER

The Peirce-Smith converter is a horizontal, refractory-lined, cylindrical furnace, generally about 4 m in diameter and 9 m long. An opening in the horizontal side serves as a mouth for charging feed materials, discharging the products of combustion, and pouring slag and blister copper (see Figure 7). The converter can rotate through an arc of about 150 degrees from the vertical for operational purposes. First developed in 1909, the Peirce-Smith converter is now used at 15 of the 16 domestic copper smelters, with as many as 9 units installed at one plant. Two or three converters are generally associated with each smelting furnace. The Peirce-Smith converter is a relatively efficient furnace, whose high rates of air flow permit both the charging of bulky materials and large copper throughputs, typically about 9 Mg of blister copper per blowing hour.

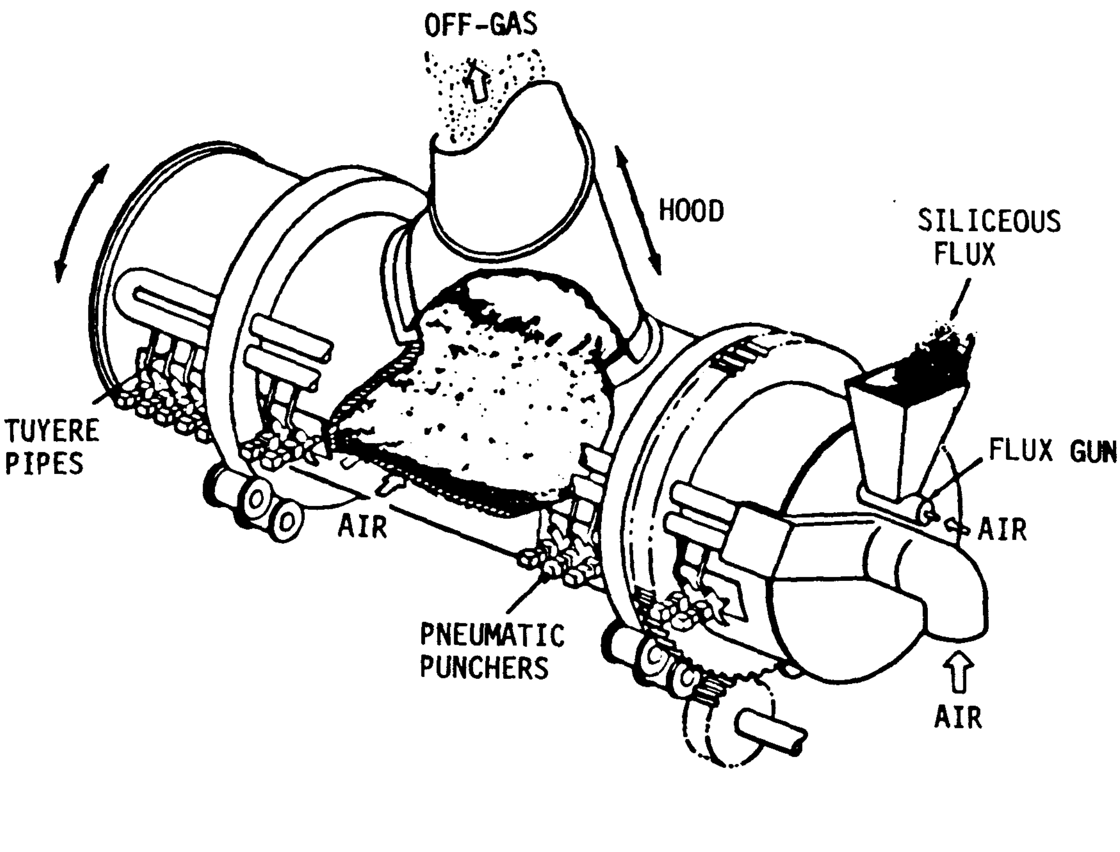


Figure 7. Peirce-Smith converter.

Operation: General

In the smelting process, the converter receives molten matte consisting of copper, iron, sulfur, and small amounts of other elements. The matte contains 40 to 45 percent copper from the reverberatory furnace; flux is added from bins or hoppers located adjacent to the converter. Air for combustion is forced through the tuyeres, a series of holes in the side of the converter 6 to 8 inches below the normal bath surface. Oxygen in the air reacts with the iron sulfides to form an iron silicate slag; this is removed, and the remaining copper sulfide is oxidized to blister copper. The reaction is exothermic. The blowing operations remove iron, as a slag iron silicate, and sulfur, as sulfur dioxide. The resultant product is blister copper, which is poured into ladles and transported to the anode furnace.

Operating Phases of a Peirce-Smith Converter

A Peirce-Smith converter ready to come on line is first preheated, usually by gas or oil-fired burners, until the converter can accept a hot charge. For charging, the converter is in a "rolled out" position ready to accept matte. A ladle of matte is transferred by the EOT crane from a smelting furnace to the converter. The hot metal is poured from a ladle into the converter (Figure 8). Three to four ladles of matte are charged to the converter in this manner and blowing begins. A total of 10 to 12 ladles of copper matte may be charged during the slag blows. A ladle of cold dope (cold material such as copper, scrap, or high-copper slag) also may be charged during one more of the slag blows. A fluxing agent, generally silica, usually is added. Before the converter is tilted back to its upright position, an air blow is initiated; as the converter is tilted, this air stream prevents the hot metal from clogging the tuyeres (typically 40 to 50). Blowing rates range from 42,500 to 68,000 m^3air/h.

As the operation proceeds, the lower-density slag floats on top of the molten layer. As the slag builds up, the converter is rolled out and slag is removed into a ladle (Figure 9). The slag, containing 6 to 8 percent copper, can be returned by overhead crane to the smelting furnace. Additional matte and/or cold dope are charged to the converter as required, and blowing begins again. Slag is again removed, and the operations are repeated until enough copper sulfide has accumulated in the converter. The slag blows are then complete, and the finish or copper blow begins.

During the slag and copper blowing periods, sulfur in the copper/sulfur/iron matte reacts with oxygen in the blowing air to form sulfur dioxide (SO_2), most of which is discharged into a primary hooding system. The concentration of the SO_2 gas during

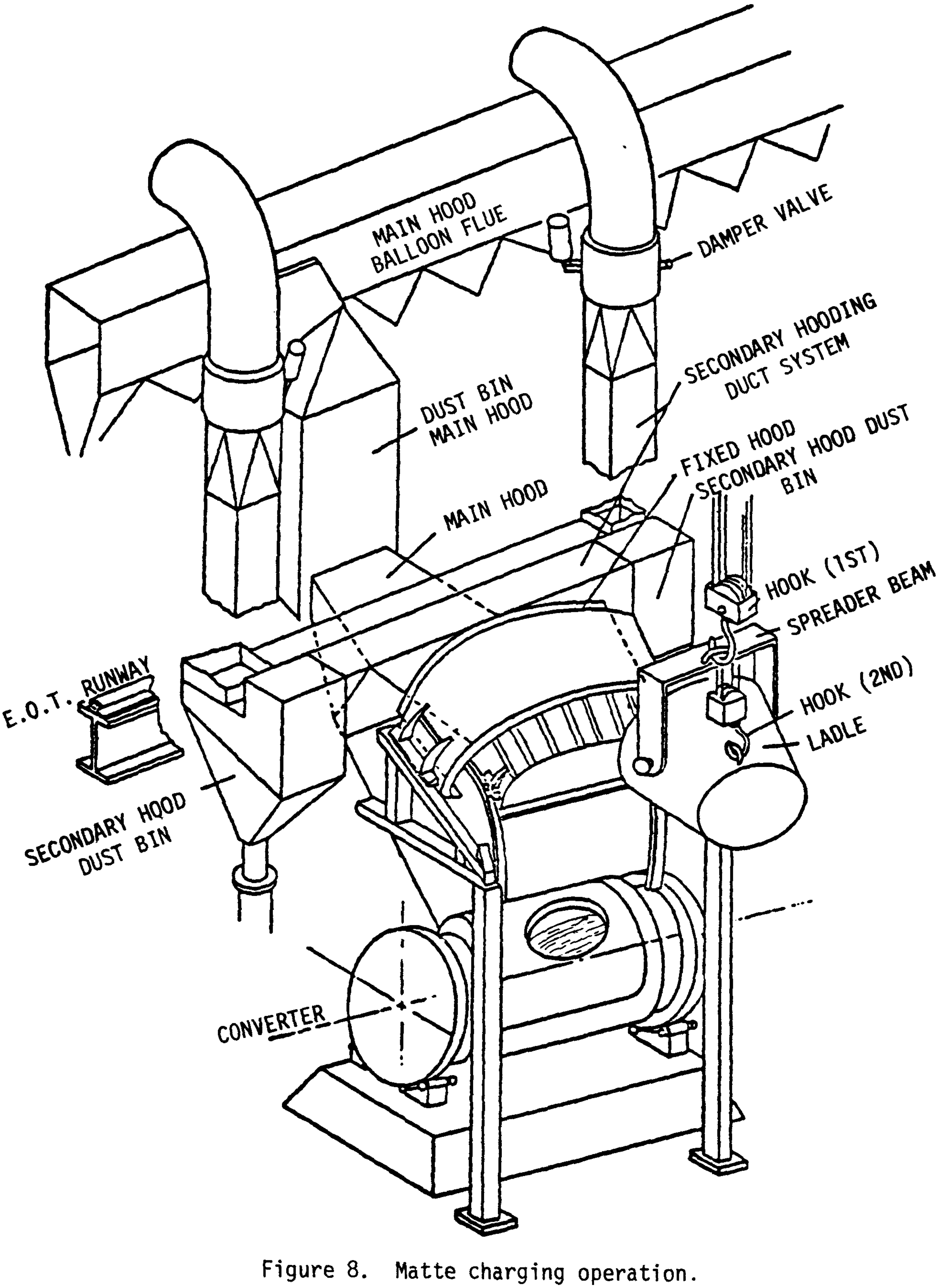


Figure 8. Matte charging operation.

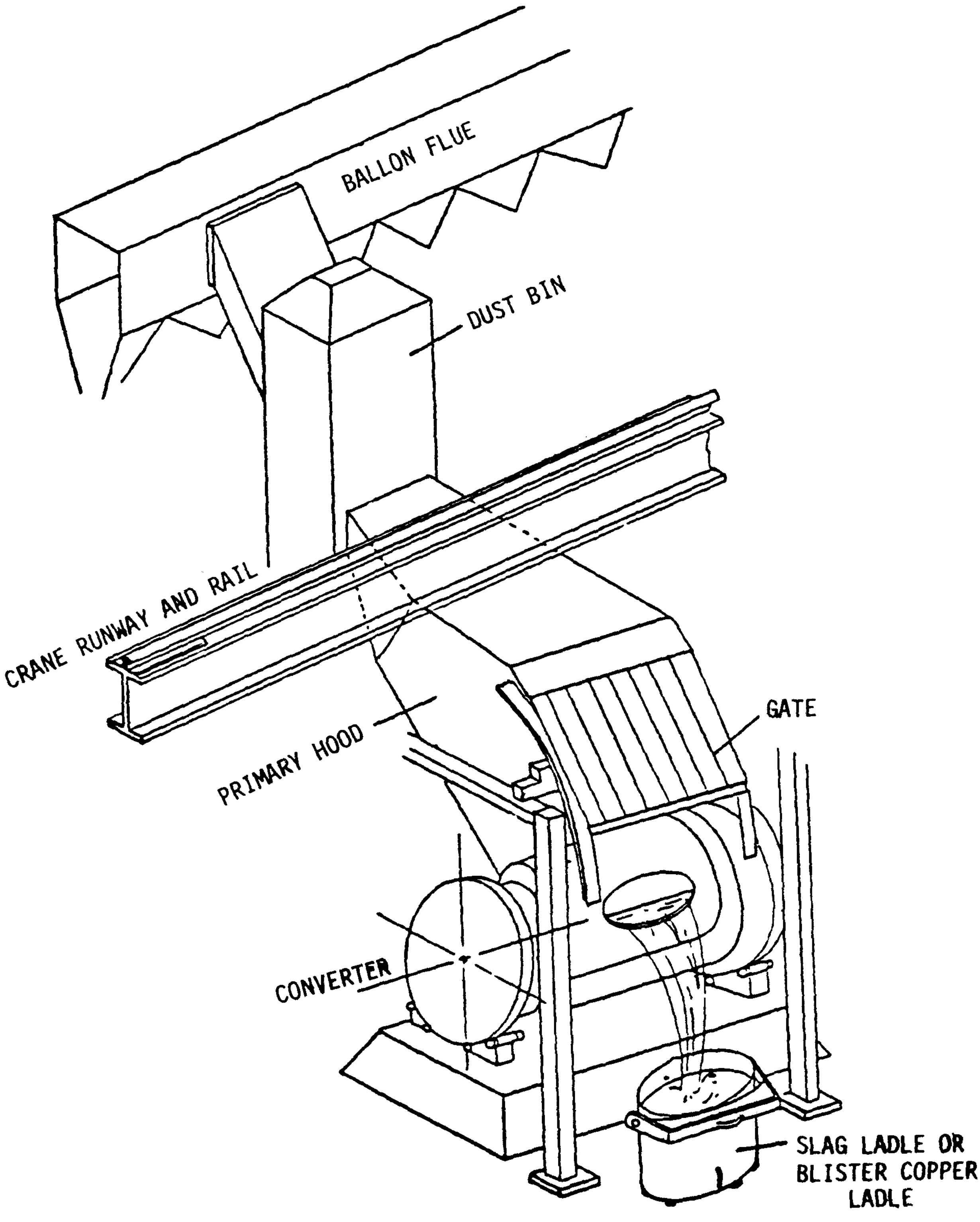


Figure 9. Position of the converter during slagging or blister copper pouring.

the slag and copper blows, even with the infiltration of dilution air, is usually great enough that the gas is sent to an acid plant for production of sulfuric acid. With the elimination of impurities, mainly the sulfur as SO_2 and iron in the slag, the matte is converted to the blister copper product that consists of about 98 to 99 percent copper.

Pouring of the blister copper, like the slag, is done through the mouth of the converter (Figure 9). The filled ladles of blister copper are transferred by the overhead crane and dumped into the anode furnaces or refining furnaces. Occasionally some slag may be formed in an anode furnace; this slag is poured off into a ladle, and then recharged into a converter.

Emissions

All of the Peirce-Smith converters at primary copper smelters in this country are equipped with primary hoods to direct the gas flow from the converter when it is in the upright, i.e., blowing or "in-stack" position. This gas stream, containing particulates and SO_2, is passed through particulate control equipment and at most smelters is routed to an acid plant for SO_2 removal.

Fugitive emissions from a Peirce-Smith converter consist of those that escape the primary hooding system and those that are emitted directly from the mouth of the converter when it is positioned in the "out-of-stack" mode, i.e., when it is receiving a cold or hot charge, or when slag or blister copper is poured from the mouth of the converter.

The primary hooding system (Figure 9) at most smelters consists of a fixed hood with a sliding gate located above and slightly away from the converter. The primary hooding system is connected by ducting usually to an ESP. The sliding gate is lowered close to the converter mouth to help guide the emissions into the fixed hood and reduce the intake of dilution air during the blowing period. Minimizing the dilution air serves to maximize the SO_2 concentration in the gas stream to an acid plant.

Regardless of the merits of any primary hooding system, some of which have been modified to increase collection efficiency, none are 100 percent efficient. The hooding system with the sliding gate does not form a perfectly tight seal with the converter body (even with the closest fits, since there must be some clearance between the two to allow the converter to rotate). At some plants, because the gates were retrofitted rather than designed and installed as part of the original smelter, the gates may not completely cover the converter mouth nor seal tightly to the primary hood. Emissions therefore escape through the open areas between the gate, the primary hood, and the converter mouth. The emission rate is dependent on the size of the openings and the discharge gas flow rate through the mouth of the converter to the hooding system. Leaks can also occur between the edges of the gate and the converter primary hood. Additionally, emissions from even a well designed and installed hood system will increase in time unless preventive maintenance is practiced to forestall the effects of normal wear and tear.

During the frequent transport and charging of materials, bumping of a gate and guide may cause them to become misaligned or damaged, thus allowing additional fugitive emissions. Emissions vary among smelters because of restrictions on overhead space for installation of the primary hood. At smelters that were originally designed with low crane rail runways, the clearance is limited. In these plants the optimal slope and configuration of the primary hoods have been compromised by lack of space, and flow of the gases from the mouth of the converter is impeded. Although the installed system may be the best possible, it does permit fugitive emissions.

The second category of fugitive emissions from a Peirce-Smith converter consists of those emitted directly from the mouth of the converter in the rolled-out position. As the converter is rolled-out, the gate is usually moved up and away from the converter mouth for clearance. The blast air is left on until the bath is below the tuyere level. Even if the blast air flow is reduced to some minimum rate, fugitive emissions are still rather heavy. Also, fugitive emissions are heavy when matte is charged to the converter. Unless a converter is equipped with a secondary hooding system, emissions that occur during charging, pouring of slag, or pouring of blister copper are uncontrolled. When the blast air is turned off, the hot bath still generates some fugitive emissions, which continue to leave the converter at a lower rate. Fugitive emissions usually become heavy also when cold material such as copper scrap is charged to the converter. Before a Peirce-Smith converter is rolled back, the blast air is turned on again. Emissions from within the converter are blown out of the mouth and are not reasonably controlled until the gate is down and the converter is in the in-stack position.

Characterization of Emissions

To date, the fugitive emissions from Peirce-Smith converters have not been collected for the purpose of chemical characterization. Since data are lacking, the following is an estimate of the composition by weight of the constituents: SO_2 - 90%, Cu - 4%, Fe - 4%, and S - 2%. The fugitive emissions also contain some trace metals such as arsenic and lead.

ANODE FURNACE

The final process at a copper smelter is purification of the blister copper in an anode furnace. The anode furnace is usually cylindrical, very similar in shape and size to a Peirce-Smith converter, and generally lined with magnesite refractory. Like the Peirce-Smith converter, the anode furnace is tilted or rolled out to receive its charge, which is poured from a blister ladle carried from the converter by an EOT crane. In the upright position, the furnace is blown with a gas high in hydrogen content. The reaction that takes place removes oxygen from the bath. This deoxidation of the cuprous oxide (Cu_2O) reduces it to nearly pure copper. Most, but not all, of the oxygen is removed from the molten bath. A small oxygen residue in the bath is necessary to cast a shape free of blisters or shrinkage holes.

The anode furnace also serves as a holding furnace, from which the anode copper product is poured, usually into molds on a continuous casting wheel. The formed anodes are shipped to an electrolytic refinery.

Any fugitive emissions from an anode furnace come directly from its mouth when not hooded. These emissions are minimal. Most or all of the sulfur, iron, and other impurities have been removed in preceding operations. The characteristic "greenish flame" shooting a few feet from the anode mouth during deoxidation probably indicates the presence of some copper in the off-gas stream.

SECTION 4

ALTERNATIVE COPPER SMELTING PROCESSES AND RELATED EMISSIONS

Several alternatives to the reverberatory furnace and the Peirce-Smith converter are in use in this country and abroad. Some of these furnaces combine several of the conventional process steps. In addition, these alternative processes usually generate lesser quantities of fugitive emissions, and for this reason each may be considered also as a means of additional air pollution control.

The alternative processes, however, do not eliminate fugitive emissions. As in the foregoing description of conventional smelting processes, this section briefly describes the operation of these units and the points at which fugitive emissions occur. The discussion includes the Noranda furnace, the electric furnace, flash smelters (Outokumpu and Inco), and the Hoboken converter.

NORANDA FURNACE

In the Noranda continuous furnace (Figure 10), the roasting, smelting, and partial converting reactions are combined in a vessel similar to a lengthened Peirce-Smith converter. One U.S. plant started operating Norandas within the past year. The reactor is a horizontal, cylindrical furnace about 21 m long. It is fired from both end walls, and oxygen-enriched air is blown into the matte layer through side-mounted tuyeres. The furnace can be rotated on its horizontal axis to bring the tuyeres out of the bath and stop the smelting process. The compact design facilitates process control, and the domestic Noranda smelter is highly instrumented. The Noranda was originally developed as a one-step process that would eliminate the converter, thus significantly reducing capital costs and eliminating the need for a converter aisle. In U.S. commercial applications to date, however, the Noranda is used with a converter to allow greater production, better control of trace elements, and longer life of the reactor lining.

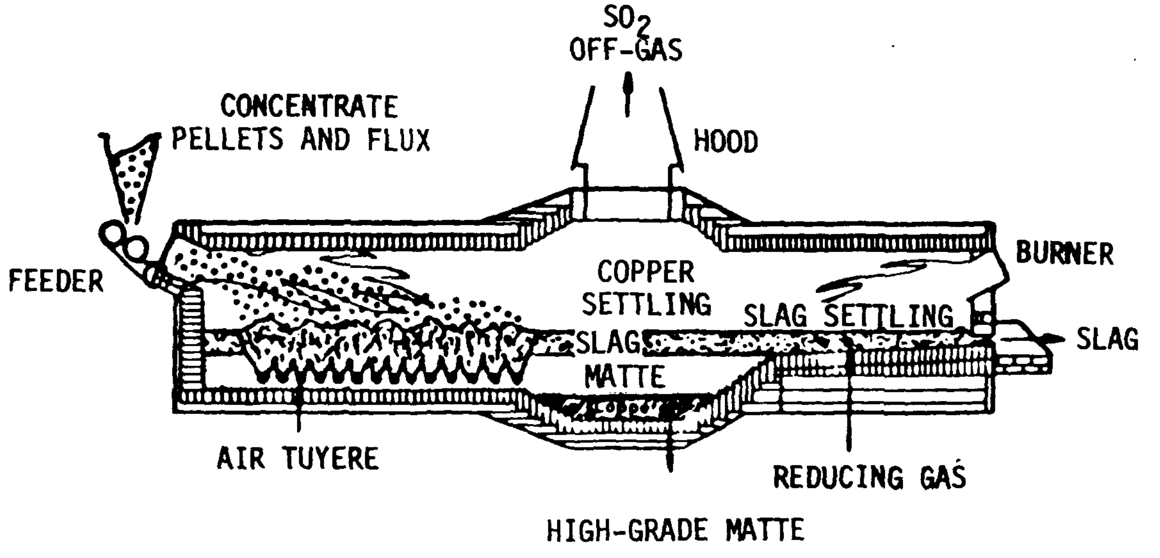


Figure 10. Noranda continuous smelter.

Operation

Concentrate and fluxes are fed to the Noranda by a slinger at one end wall that spreads pelletized feed over the molten bath. High-grade matte, which typically contains about 70 percent copper, is periodically tapped from the side of the furnace and transported by ladles to standard Peirce-Smith converters, where it is batch-treated to remove additional sulfur and iron prior to fire refining. Slag containing 6 to 8 percent copper is periodically tapped from the end of the vessel opposite the slinger. The slag is upgraded by milling to produce a concentrate, which is returned to the reactor, and a tailing, which is discarded. The off-gases leave the Noranda furnace through its mouth, where they are captured by water-cooled hoods and ducted to a waste heat boiler. The gases are passed through cyclones and electrostatic precipitators to remove particulate matter, and then used as feed to a sulfuric acid plant. With 30 percent oxygen enrichment, the off-gases to the acid plant contain 5 to 6 percent SO_2. The SO_2 concentration is approximately 4 percent without oxygen enrichment.

Emissions

If uncontrolled, fugitive emissions evolve from the following areas around a Noranda smelting furnace: between primary uptake hood and furnace mouth; from the mouth when in the rolled out position; around matte and slag holes during tapping; and at the port for feeding concentrate and fluxes.

ELECTRIC FURNACE

The electric copper smelting furnace (Figure 11) has been used traditionally in Scandinavian areas where hydroelectric power is cheap and fossil fuels are expensive. The first such furnace in the United States started operation in 1972; two more smelters have since adopted this technology.

The electric furnace is rectangular in cross-section with a firebrick sprung-arch roof. The largest furnaces are about 35 m long and 10 m wide. Carbon electrodes are placed in the molten slag, and the heat required for smelting is generated by electrical resistance of the slag to the submerged arc between electrode pairs. Electrical ratings range as high as 51,000 kVA. The chemical and physical changes that occur in the molten bath are somewhat similar to those that occur in a reverberatory furnace. The reverberatory furnace with a waste heat boiler is more efficient than the electric smelting furnace.[1]

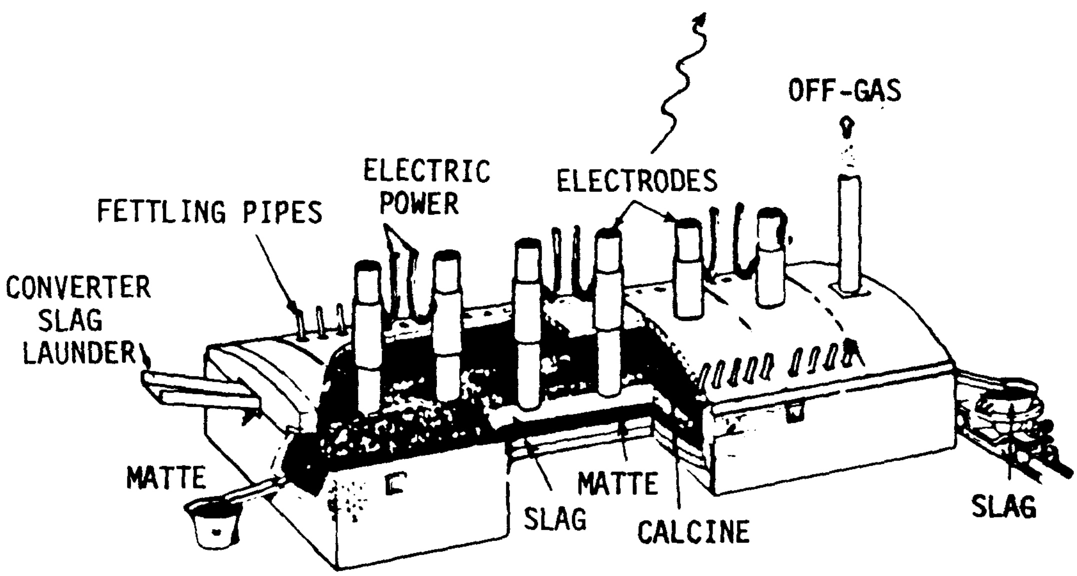


Figure 11. Electric smelting furnace.

Operation

The charge of concentrate and fluxes is delivered to the roof of the furnace by drag conveyors and then fed to the molten bath through multiple-feed spouts near the electrodes and between the sidewalls. As the charge materials melt, they settle into the bath and form additional matte and slag. Separate launders or chutes on the furnace end wall are used to charge converter slag and reverts. Matte is tapped into ladles from tap holes placed in the hearth area near one end wall. Slag is skimmed from tap holes in the opposite end wall and delivered by launders into slag pots, which are usually hauled to a dump by trucks.

Although originally designed as an alternative to the use of expensive fuels in Scandinavian countries, the electric furnace also facilitates air pollution control. Because it does not require large amounts of combustion air, the volume of outlet gases is about an order of magnitude less than those from a reverberatory furnace. Sulfur dioxide concentrations of 2 to 4 percent can be expected, and particulate emissions should be lower than from a reverberatory because of the lower gas volume and more uniform gas flow. The electric furnace off-gas at all three domestic smelters is combined with other high-SO_2 gas streams and fed to contact sulfuric acid plants.

Emissions

Fugitive emissions around electric arc furnaces are lower than those from most reverberatory furnaces. For example, the electric arc furnace at the Anaconda smelter in Montana has tight brickwork, which prevents the leakage of fugitive emissions from the sides of the smelter; with poor maintenance, however, the brickwork could be a source of emissions. Where a hooding system is used ineffectively over the slag ladle during slagging, emissions will occur. Other sources of fugitive emissions are matte tapping, the converter slag return launder, around the electrodes, and the calcine handling system.

FLASH SMELTING SYSTEMS

A recent development in copper metallurgy is the continuous flash furnace, which is more efficient in terms of energy consumption and also produces a more easily controlled stream of flue gas than the reverberatory or electric furnaces. Flash furnaces are of two types, the Outokumpu Oy and the Inco, which differ primarily in their use of either preheated air or commercial-grade oxygen to sustain the smelting reaction. The flash furnace is in widespread use throughout the world, although only one is operating in this country, under license from Outokumpu.

The Outokumpu Furnace

The Outokumpu furnace (Figure 12) combines the functions of roasting, smelting, and partial converting in a single furnace with three sections--reaction shaft, settler, and uptake shaft. Dried ore concentrates are injected continuously along with flux and preheated air into the reaction shaft through concentrate burners. Oil may also be injected into the shaft. The finely divided concentrate burns in a "flash" combustion as the particles fall down through the shaft, and the heat released from the combustion of the oil and sulfur sustains the smelting reaction. The process is similar to the combustion of pulverized coal. The molten particles fall into the settler part of the furnace and separate into matte and slag layers. The matte, which contains 45 to 75 percent copper, is tapped from the settler and transferred to converters for further processing. The slag, which contains too much copper to discard, is also processed further in an electric furnace. From the electric furnace the copper containing-material from the slag is sent to the Peirce-Smith converter. The slag from the electric furnace is then dumped. Outlet gases from the Outokumpu furnace are conveyed from the uptake shaft. They contain 10 to 20 percent SO_2 and considerable quantities of entrained particulate matter. The gases are cooled in a waste heat boiler, cleaned of particulates in an electrostatic precipitator, and then sent for sulfuric acid production.

Fugitive emissions from operation of the Outokumpu furnace can occur at the launders and ladles and from leakage through the furnace walls and roof.

The Inco Smelter

The Inco smelter (Figure 13) is similar to a reverberatory furnace except that the off-gases are discharged at the center of the furnace. The dry ore concentrate and fluxing agents are injected into the furnace through both end walls, and oxygen is also injected through both end walls. Fine particles are dispersed into the furnace, and flash combustion of the sulfides creates the heat required for the smelting. The molten matter falls on the molten bath, again forming a matte and a slag layer. Charging is continuous. The slag, of high copper content, is treated for copper recovery. The matte grade is higher than that of matte from the reverberatory furnace. The volume of off-gases decreases as oxygen enrichment is utilized. Fugitive emissions from the Inco smelter can occur at the launders and ladles, and by leakage from the furnace sides, roofs, and off-take.

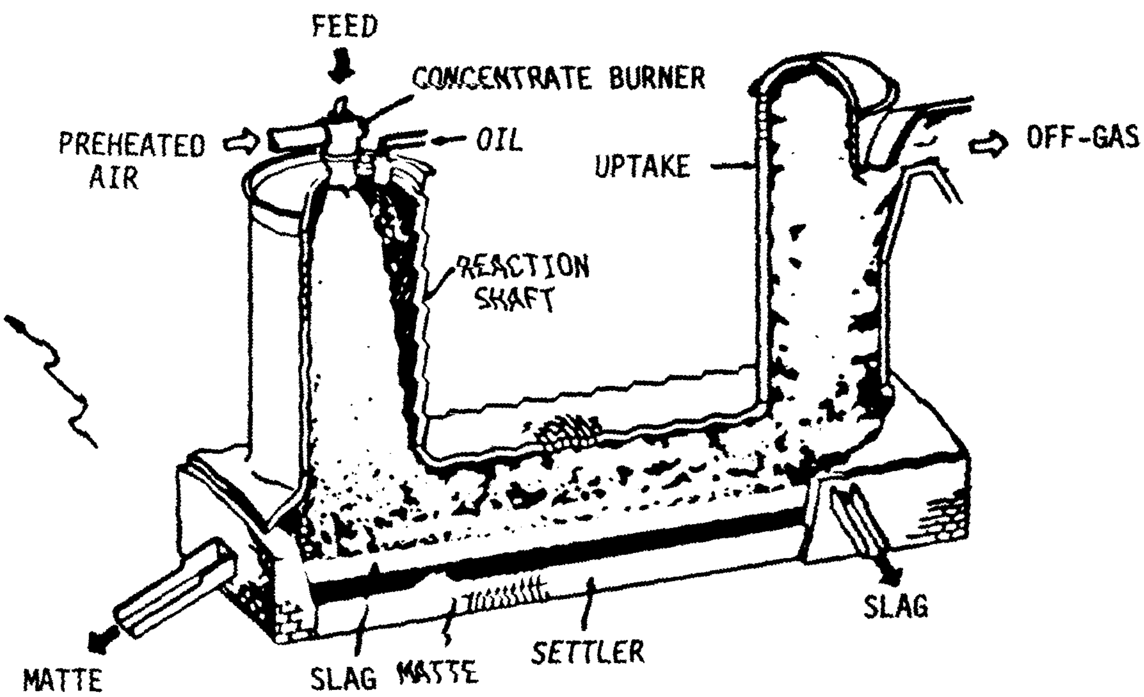


Figure 12. Outokumpu flash smelting furnace.

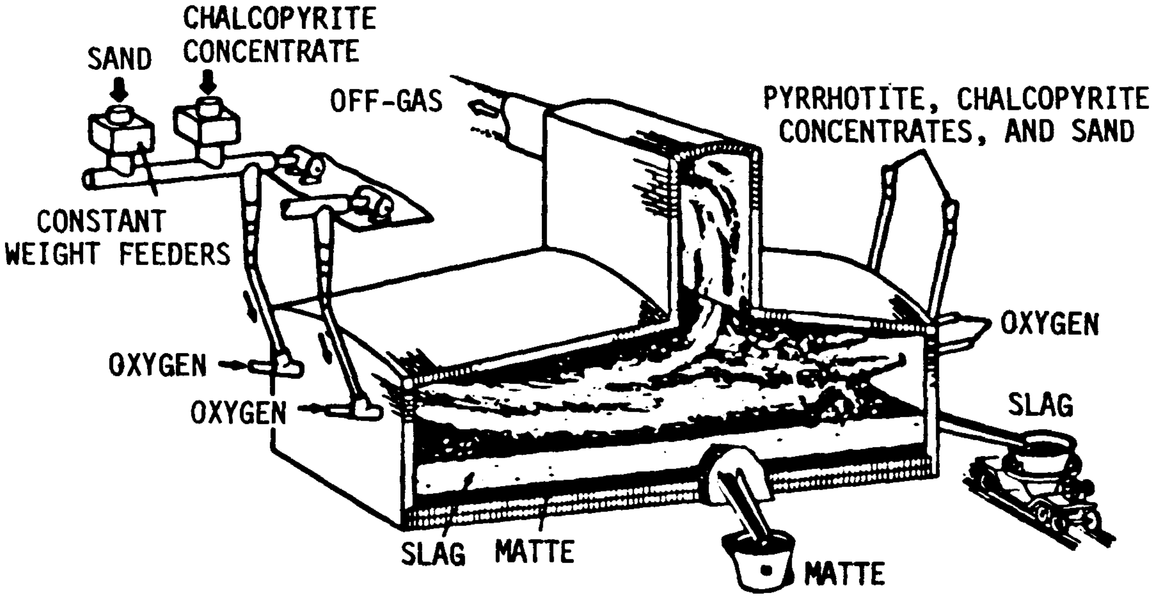


Figure 13. INCO flash smelting furnace.

HOBOKEN CONVERTER

The Hoboken converter, shown in Figure 14, is an alternative to the Peirce-Smith converter. The Hoboken was designed to largely eliminate the problem of excess air infiltration into the flue gas off-take system. First developed in the early 1930's in Belgium, a Hoboken has been operated at a single domestic primary smelter for about 4 years; Hobokens are also installed at a number of copper plants in six foreign countries.

The Hoboken converter is similar to the Peirce-Smith, but is equipped with an integral side flue at one end of the furnace for withdrawal of the off-gas. Shaped like an inverted "U", this flue, or siphon, rotates with the converter, as does the cylindrical duct to which it is connected. A counterweight balances the siphon. The cylindrical duct is connected by an airtight rotating joint to a fixed vertical duct that leads to the gas cleaning system. The Hoboken thus provides a direct link at all times between the converter and the gas off-take, regardless of its operating position.

The Hoboken converters at the U.S. smelter in Inspiration, Arizona, are 4.3 m in diameter by 11.6 m long. Each converter has fifty-two 3.8 cm diameter tuyeres; air blast is approximately 31,500 Nm^3/h.

Operation

Matte from the electric furnace at the Inspiration plant is charged into one of the refractory-lined Hoboken converters. When the converter is in the blowing position, the tuyere line is 20 to 25 cm below the bath level, as with the Peirce-Smith. To operate as designed, the draft at the mouth should be maintained at zero or a slightly negative draft; the design concept is that emissions will not escape the mouth of the converter in this operating mode but will be ducted to the end-mounted flue.

At Inspiration, temperature of the gas stream in the siphon area of the Hoboken converter is 950° to 1100°. Gases containing SO_2 from the converter are combined with the gas stream from the electric furnace and are sent to the acid plant after cooling and particulate removal. So that flow of SO_2-containing gas to the acid plant can be maintained, at least one converter must always be in the blowing phase. A continuous stream of hot gas through the duct system also minimizes condensation.

Charging, blowing, slagging or skimming, and pouring of blister copper pouring are very similar to operations with the Peirce-Smith converter. A Hoboken converter ready to come on-line is heated and then charged with matte. The matte is poured from a ladle, which is held and tilted by the EOT crane

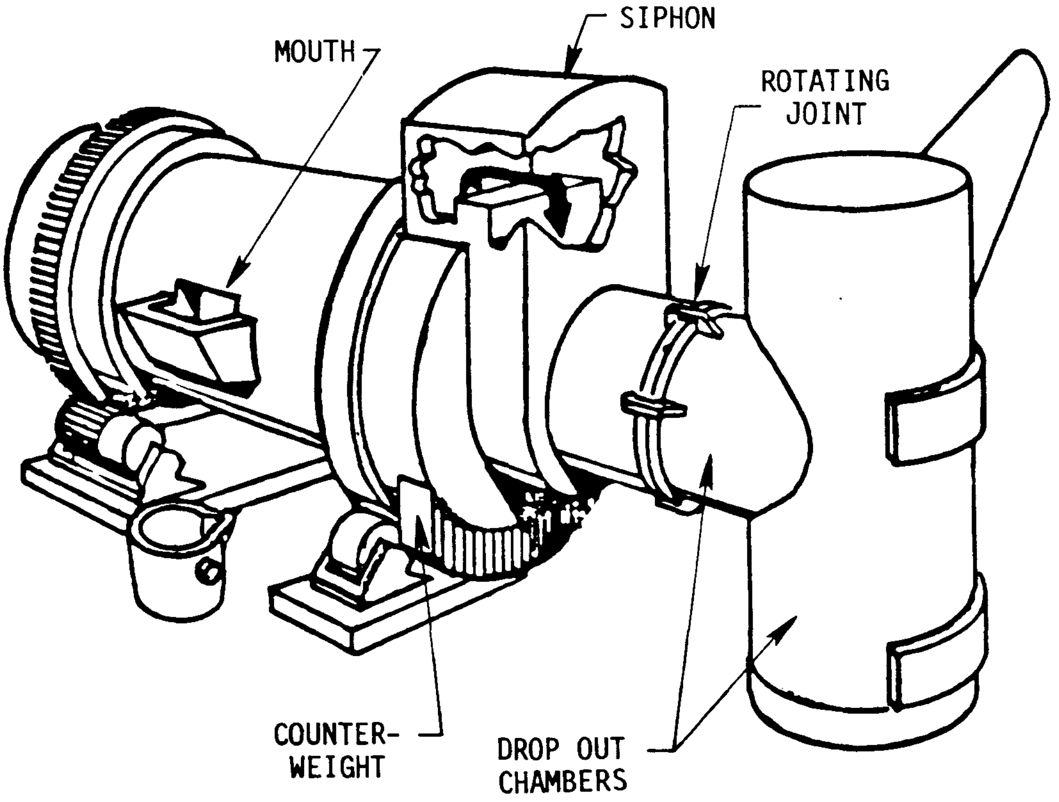


Figure 14. Hoboken converter.

hooks, into the rolled-out converter. The initial charge consists of several ladles of matte. After flux is charged, the air blast is turned on to begin the slag blow. The bath is blown for 45 minutes to 1 hour, then the blow air is turned down, the converter is tilted, and slag is poured off into a ladle positioned on the floor of the converter aisle. After one or two ladles of slag are poured, a similar amount of matte is charged to the converter. When in the upright position, the converter is blown again to further burn off the sulfur combined with the iron, and allow the iron oxide that is formed to react with the flux to produce slag. The off-gas stream containing the sulfur oxidized during blowing is pulled through the siphon to the uptake duct, then sent to an acid plant. At the Inspiration plant, slag is poured off and transferred in ladles via the EOT crane and charged into the electric smelting furnace for recovery of the copper.

As with the Peirce-Smith converter, the slag blow is followed by the blister copper blow. The copper sulfide bath is blown until all or nearly all of the sulfur is burned off, mainly oxidized as SO_2. Upon completion of the copper blow, the product is blister copper. It is poured into ladles that are carried by the EOT crane to the anode furnace.

Emissions

One major difference between the Hoboken and Peirce-Smith converters is that the Hoboken is not equipped with a primary uptake hood over the mouth, but rather with the siphon and side-mounted flue. Any emissions that escape the mouth are fugitive emissions.

A properly designed, operated, and maintained Hoboken converter (such as the one operated by Hoboken Overpelt in Belgium) generates only minimal fugitive emissions, even though the mouth is not hooded, because the converter is operated under zero or slightly negative draft. At slight negative draft, air from outside of the converter is sucked into the mouth and exits through the converter siphon with the gas stream containing the SO_2 generated during blowing. Air brought into the converter in this manner cools the discharge gas stream and dilutes the SO_2 in the off-gas stream, but this dilution can be minimized. Slight negative or zero draft prevents fugitive emissions from the mouth of the converter.

The small amounts of fugitive emissions from a Hoboken converter usually occur during charging of matte or cold material or during slagging or blister copper pouring. During these operations, the converter is rolled out and the blowing air is turned down, but the zero or slightly negative draft is maintained. During matte charging, some emissions are released from the hot metal stream as it flows from ladle to furnace. Emissions during slagging and blister copper pouring occur in a similar manner.

In summary, a Hoboken converter that is well designed, installed, operated, and maintained generates small quantities of fugitive emissions. The addition of a swing-away hood (Figure 15) would minimize emissions during slagging or pouring. During the blowing periods, fugitive emissions are controlled to a much greater degree than with the Peirce-Smith converter.

No characteristics or analyses of fugitive emissions are available.

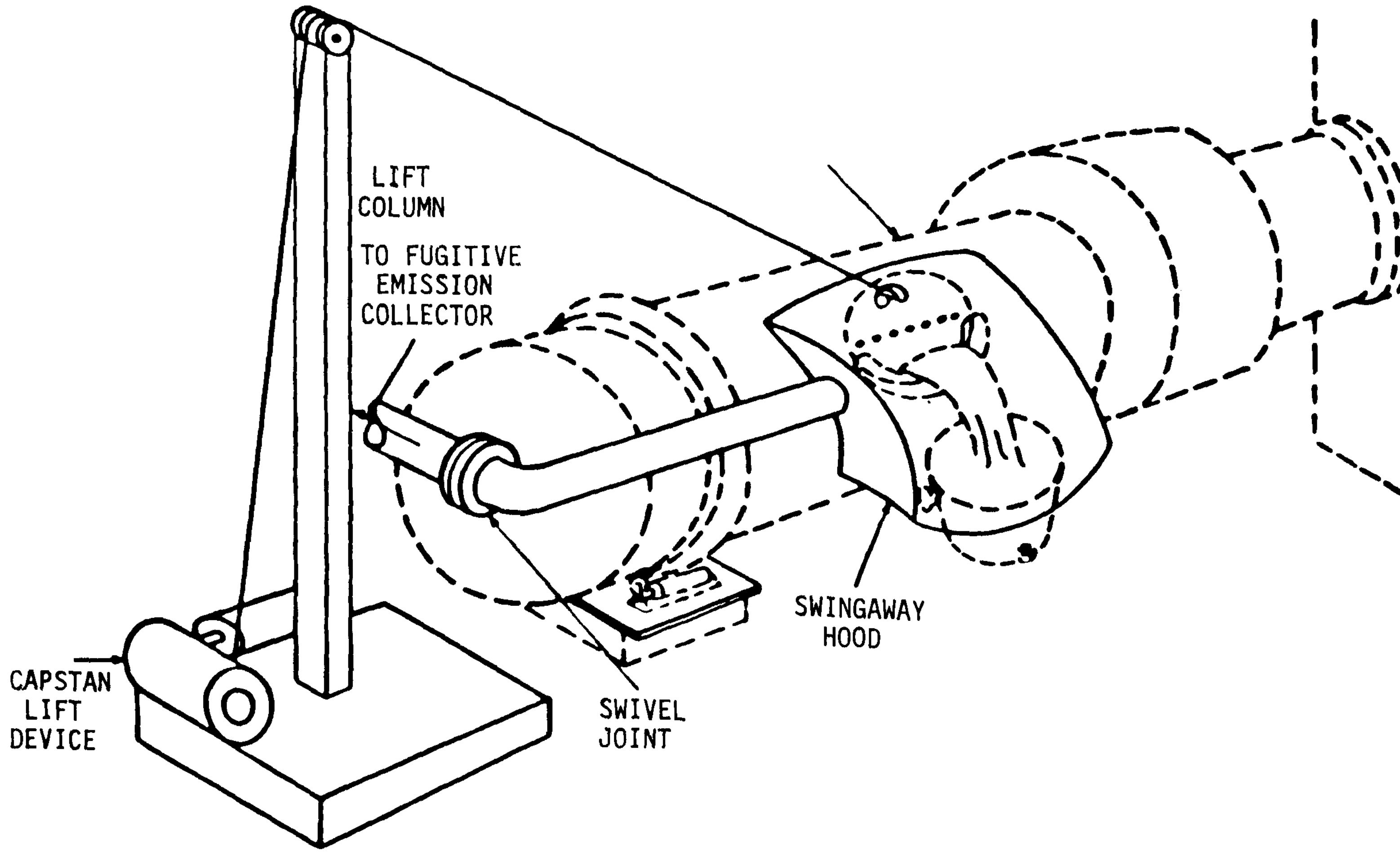


Figure 15. Hoboken converter with swingaway hood.

SECTION 5

SUMMARY OF FUGITIVE EMISSIONS FROM COPPER SMELTERS

In efforts to determine typical quantities of fugitive emissions from copper smelters, the authors consulted the technical literature and undertook an inquiry by telephone with knowledgeable persons in the copper industry and in government. The literature search indicated a minimal amount of valid published data on process rates, emission rates, and emission characteristics. In the course of the inquiry we learned of no actual published measurements of fugitive emissions from reverberatory furnaces, converters, or other process equipment at copper smelters.

We have therefore prepared a set of emission estimates corresponding to several arrangements of process equipment, summarized in Table 1. For each combination of process equipment the emission estimates are based on maintaining the same production rate, 303 tons of blister copper per day. The emission values are based on data from several sources, primarily reports (References 2 and 3) and conference presentations (References 4 and 5). Reference 2 was the principal source of the emission estimates. Data from all of the sources have been modified to accommodate the selected production rate and thus allow comparison among the several process arrangements.

Figures 16 through 27 depict material balances for each of the equipment items listed in Table 1. Each figure is intended to approximate the material balance that would occur when the equipment is operated in the specific combination designated in Table 1. Thus, although the Peirce-Smith converter is a component of each of the five equipment combinations in the table, the materials entering the converter depend upon the equipment that precedes it in the smelting process; e.g., matte entering the converter from a reverberatory furnace (Figure 18) contains greater quantities of sulfur, iron, and other materials than matte from the Outokumpu furnace (Figure 24). The emission estimates of Table 1 reflect these differences and thus provide an indication of the relative magnitude of emissions under various process arrangements.

TABLE 1. ESTIMATED FUGITIVE EMISSIONS FROM COPPER SMELTING IN VARIOUS PROCESS ARRANGEMENTS

		Estimated Fugitive Emissions			
		SO_2[a]	Cu[b]	Fe[b]	Others[b]
1.	Multihearth roaster (Fig. 16)	c	c	c	c
	Reverberatory furnace (Fig. 17)	4.2	0.16	0.10	0.10
	Peirce-Smith converter (Fig. 18)	6.5	0.16	0.21	0.11
2.	Fluid-bed roaster (Fig. 19)	c	c	c	c
	Reverberatory furnace (Fig. 20)	4.2	0.16	0.09	0.12
	Peirce-Smith converter (Fig. 21)	6.5	0.16	0.21	0.13
3.	Noranda furnace (Fig. 22)	2.9	0.11	0.07	0.04
	Peirce-Smith converter (Fig. 23)	1.5	0.03	0.33	0.12
4.	Dryer	c	c	c	c
	Outokumpu furnace (Fig. 24)	2.9	0.15	0.15	0.07
	Peirce-Smith converter (Fig. 25)	2.1	0.03	0.38	0.08
5.	Dryer	c	c	c	c
	Electric furnace (Fig. 26)	3.0	0.15	0.07	0.04
	Peirce-Smith converter (Fig. 27)	6.5	0.19	0.21	0.06

[a] Percentage of sulfur charged to the process equipment, expressed as SO_2.

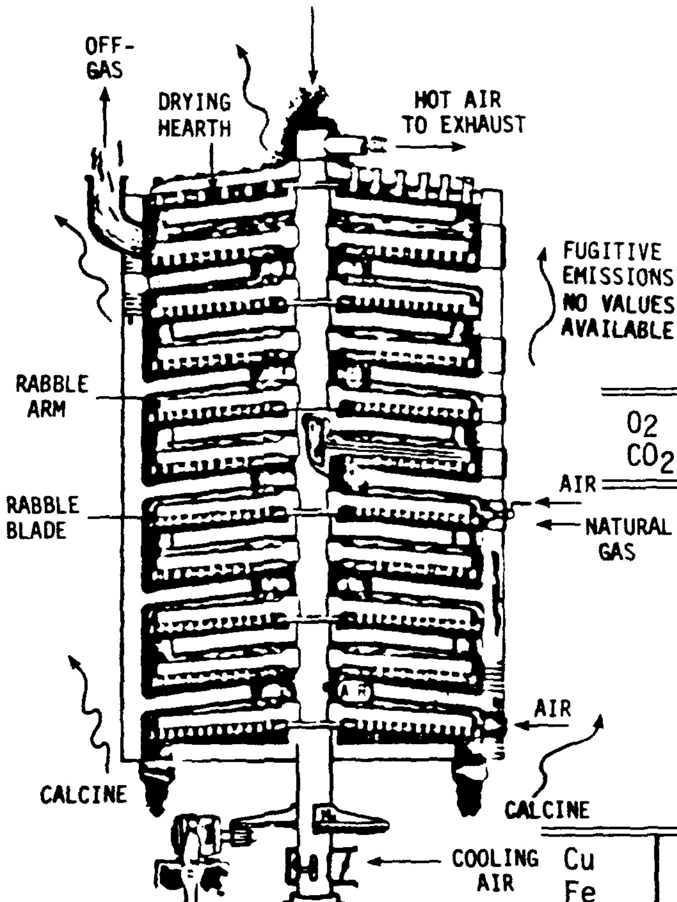
[b] Percentage of the total copper, iron, or other materials charged to the process equipment.

[c] No values available.

SO_2	383.0	(54.8)	7.0% VOL.
CO_2	53.5	(7.6)	

FEED		
Cu	308.6	(44.1)
Fe	308.6	(44.1)
S	382.9	(54.7)
SiO_2	102.9	(14.7)
CaO	5.7	(0.8)
Other	34.2	(4.9)

FLUX		
CaO	49.0	(7.0)
SiO_2	97.1	(13.9)
CO_2	37.3	(5.3)
Other	12.2	(1.7)

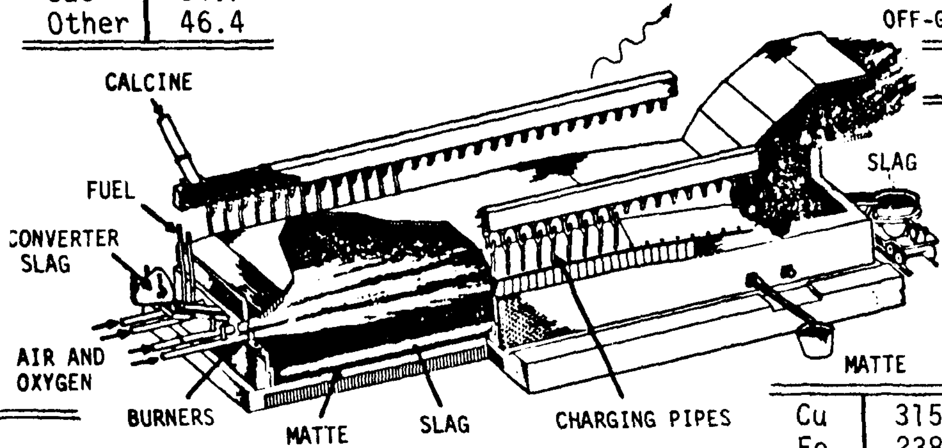


O_2	191.5	(27.4)
CO_2	16.2	(2.3)

Cu	308.6	(44.1)
Fe	308.6	(44.1)
S	191.4	(27.3)
SiO_2	200.0	(28.6)
CaO	54.7	(7.8)
Other	46.4	(6.6)

MATERIAL BALANCE IN TONS/DAY.
NUMBERS IN PARENTHESES INDICATE CAPACITY OF EACH UNIT.
OTHER VALUES ARE BASED ON 303 TONS/DAY OF BLISTER COPPER.

Figure 16. Material balance: multihearth roasting furnace.[6]



Calcine:

Cu	308.6
Fe	308.6
S	141.4
SiO_2	200.0
CaO	54.7
Other	46.4

Converter slag:

S	3.7
Cu	12.0
Fe	239.1
SiO_2	115.2
CaO	5.0
O_2	69.2
Other	6.9

Air and oxygen:

O_2	103.8

FUGITIVES	
SO_2	8.1
Cu	0.5
Fe	0.5
S	0.2

OFF-GAS

SO_2	114.7	1 VOL. %

Slag:

Cu	4.6
Fe	308.7
S	6.8
SiO_2	315.2
CaO	56.0
O_2	63.7
Other	53.1

MATTE

Cu	315.5
Fe	238.5
S	126.6
CaO	3.7
O_2	47.7

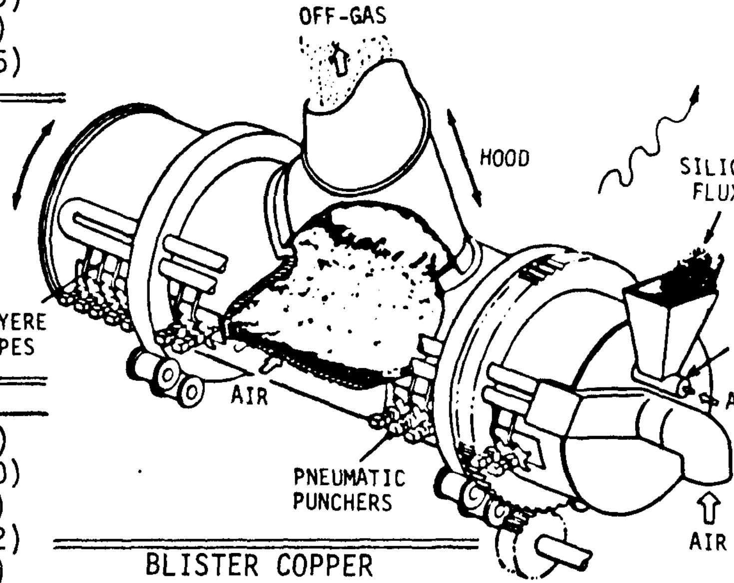
MATERIAL BALANCE IN TONS/DAY
ALL VALUES ARE ON BASIS OF 303 TONS/DAY OF BLISTER COPPER (SLIGHTLY BELOW CAPACITY OF UNIT).

Figure 17. Material balance: reverberatory furnace after multihearth roaster.[6]

MATTE		
Cu	315.5	(129.3)
Fe	238.5	(97.7)
S	126.6	(51.9)
CaO	3.7	(1.5)
O_2	47.7	(19.5)

SO_2	237.0	(97.2)	4% VOL.

FUGITIVES		
SO_2	8.2	(3.4)
Cu	0.5	(0.2)
Fe	0.5	(0.2)
S	0.2	(0.1)



Fe	2.6	(1.1)
SiO_2	115.2	(47.2)
CaO	1.3	(0.5)
O_2	0.7	(0.3)
Other	8.2	(3.4)

SLAG		
Cu	12.0	(4.9)
Fe	239.1	(98.0)
S	3.7	(1.5)
SiO_2	115.2	(47.2)
CaO	5.0	(2.0)
O_2	69.2	(28.4)
Other	6.9	(2.8)

O_2	143.3	(58.9)

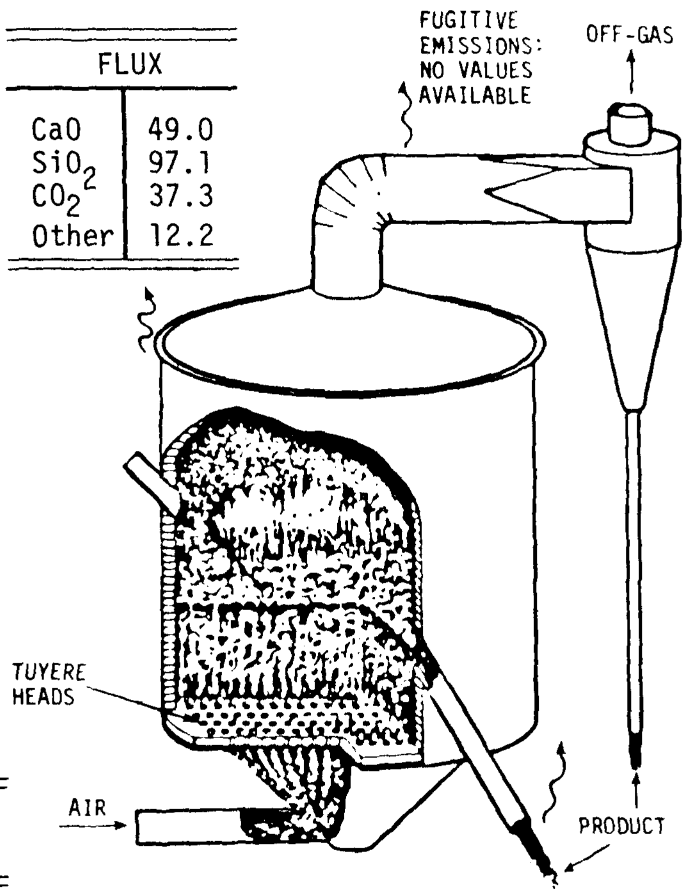
BLISTER COPPER		
Cu	303.0	(124.2)
Fe	1.5	(0.6)
Other	1.5	(0.6)

MATERIAL BALANCE IN TONS/DAY.
VALUES IN PARENTHESES INDICATE CAPACITY OF UNIT; OTHER VALUES ARE ON BASIS OF 303 TONS/DAY BLISTER COPPER.

Figure 18. Material balance: Peirce-Smith converter after reverb and multihearth.[6]

FEED	
Cu	308.6
Fe	308.6
S	382.9
SiO_2	102.9
CaO	5.7
Other	34.2

FLUX	
CaO	49.0
SiO_2	97.1
CO_2	37.3
Other	12.2



SO_2	255.8	13% VOL.
CO_2	39.0	

O_2	127.9
CO_2	1.7

Cu	308.6
Fe	308.6
S	255.0
SiO_2	200.0
CaO	54.7
Other	46.4

MATERIAL BALANCE IN TONS/DAY.
TONNAGES ARE FOR CAPACITY OF UNIT

Figure 19. Material balance: fluid-bed roaster.[6]

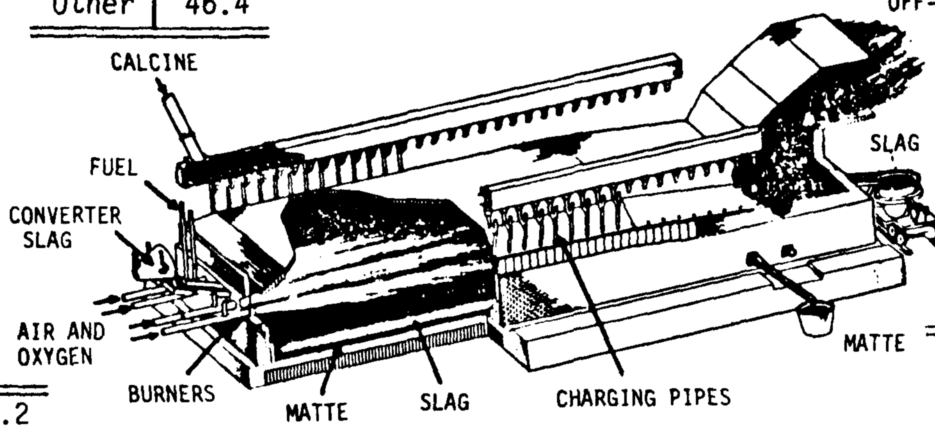
CONVERTER SLAG	
Cu	12.0
Fe	240.0
S	5.0
SiO_2	115.2
CaO	5.0
O_2	91.2
Other	7.8

O_2	139.2

Cu	308.6
Fe	308.6
S	255.0
SiO_2	200.0
CaO	54.7
Other	46.4

FUGITIVES	
SO_2	10.8
Cu	0.5
Fe	0.5
S	0.3

SO_2	153.0	1% VOL.



Cu	308.6
Fe	308.6
S	255.0
SiO_2	200.0
CaO	54.7
Other	46.4

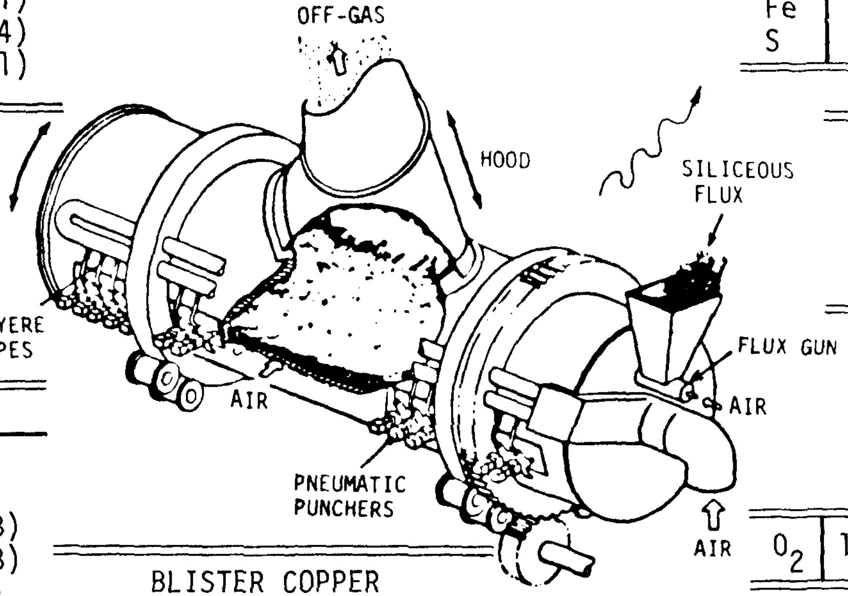
MATERIAL BALANCE IN TONS/DAY.
TONNAGES ARE FOR CAPACITY OF UNIT.

Figure 20. Material balance: reverberatory furnace after fluid-bed roaster.[6]

MATTE		
Cu	315.5	(119.8)
Fe	238.5	(90.6)
S	168.7	(64.1)
CaO	3.7	(1.4)
O_2	63.4	(24.1)

SO_2	316.0	(120.0)	4% VOL.

FUGITIVES		
SO_2	11.0	(4.2)
Cu	0.5	(0.2)
Fe	0.5	(0.2)
S	0.2	(0.1)



Fe	2.6	(1.0)
SiO_2	115.2	(43.8)
CaO	1.3	(0.5)
O_2	0.7	(0.3)
Other	8.2	(3.2)

SLAG		
S	5.0	(1.9)
Cu	12.0	(4.6)
Fe	239.1	(90.8)
SiO_2	115.2	(43.8)
CaO	5.0	(1.9)
O_2	91.2	(34.6)
Other	6.9	(2.6)

BLISTER COPPER		
Cu	303.0	(115.0)
Fe	1.5	(0.6)
Other	1.5	(0.6)

O_2	190.8	(72.3)

MATERIAL BALANCE IN TONS/DAY.
VALUES IN PARENTHESES INDICATE CAPACITY OF UNIT; OTHER VALUES ARE ON BASIS OF 303 TONS/DAY BLISTER COPPER.

Figure 21. Material balance: Peirce-Smith converter after fluid-bed and reverberatory furnace.[6]

FLUX

Fe	11.7	(14.1)
S	2.4	(2.9)
SiO_2	181.6	(219.2)

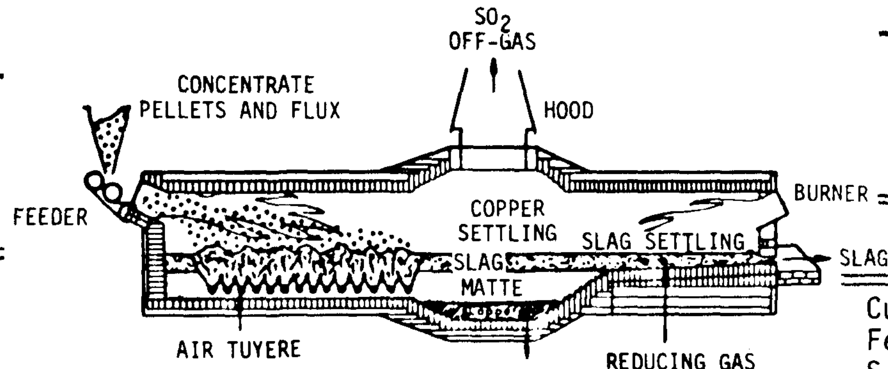
SO_2	650.7	(785.4)	6% VOL.

PELLETIZED CONCENTRATES

Cu	357.7	(431.7)
Fe	395.7	(477.6)
S	421.9	(509.2)
SiO_2	67.6	(81.6)

FUGITIVES

SO_2	12.3	(14.8)
Cu	0.4	(0.5)
Fe	0.3	(0.4)
S	0.3	(0.4)



SLAG

Cu	42.3	(51.0)
Fe	398.7	(481.2)
S	10.3	(12.4)
SiO_2	249.2	(300.8)

O_2	331.5	(400.1)

HIGH-GRADE MATTE

Cu	315.0	(380.2)
Fe	8.4	(10.1)
S	82.2	(99.2)

MATERIAL BALANCE IN TONS/DAY.
VALUES IN PARENTHESES INDICATE CAPACITY OF UNIT; ALL OTHERS ARE ON BASIS OF 303 TONS/DAY BLISTER COPPER.

Figure 22. Material balance: Noranda continuous smelter.[6]

MATTE

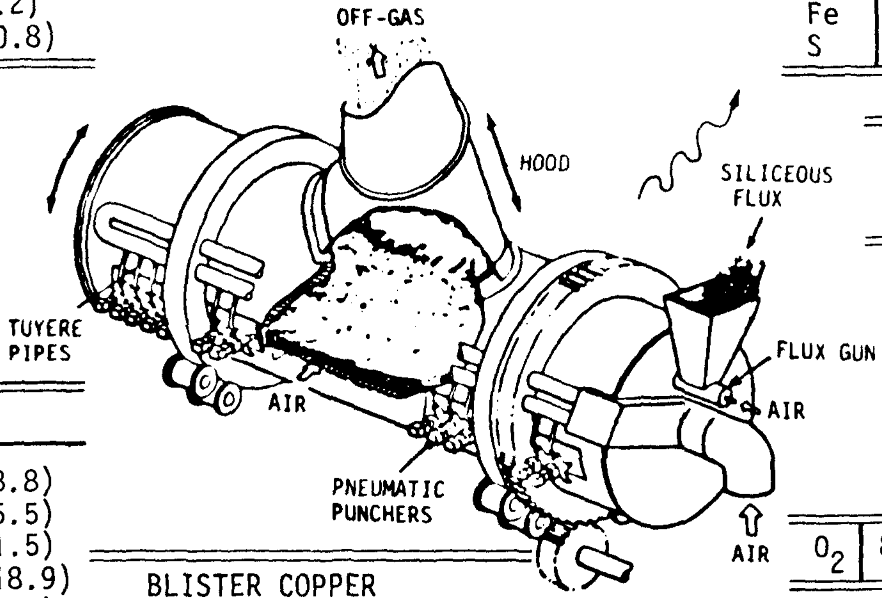
Cu	315.0	(233.0)
Fe	8.4	(6.2)
S	82.2	(60.8)

SO_2	159.0	(117.5)	4% VOL.

FUGITIVES

SO_2	1.2	(0.9)
Cu	0.1	(0.1)
Fe	0.005	(0.004)
S	0.1	(0.1)

SiO_2	25.5	(18.9)
Fe	0.6	(0.4)
Other	2.3	(1.7)



SLAG

Cu	11.9	(8.8)
Fe	7.5	(5.5)
S	2.0	(1.5)
SiO_2	25.5	(18.9)
Other	0.8	(0.6)

O_2	80.1	(59.2)

BLISTER COPPER

Cu	303.0	(224.1)
Fe	1.5	(1.1)
Other	1.5	(1.1)

MATERIAL BALANCE IN TONS/DAY.
VALUES IN PARENTHESES INDICATE CAPACITY OF UNIT; OTHER VALUES ARE ON BASIS OF 303 TONS/DAY BLISTER COPPER.

Figure 23. Material balance: Peirce-Smith converter after Noranda furnace.[6]

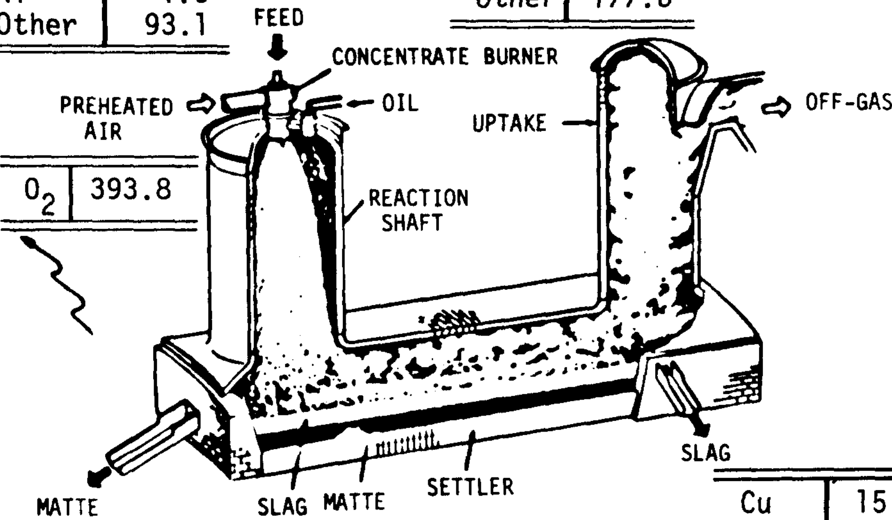
DRIED CONCENTRATES	
S	499.6
Fe	467.6
Cu	330.6
SiO_2	64.1
Ni	1.6
Other	93.1

FLUX	
SiO_2	205.7
Fe	8.1
Ni	2.3
Other	177.6

FUGITIVES	
SO_2	14.5
Cu	0.5
Fe	0.7
S	0.6

O_2	393.8

SO_2	773.1
(10% VOL.)	



HIGH-GRADE MATTE	
Cu	315.0
Ni	3.5
Fe	25.5
S	89.1
Other	8.4

Cu	15.1
Ni	0.4
Fe	449.5
S	16.1
SiO_2	269.8
Other	262.3

MATERIAL BALANCE IN TONS/DAY.
VALUES INDICATE CAPACITY OF UNIT.

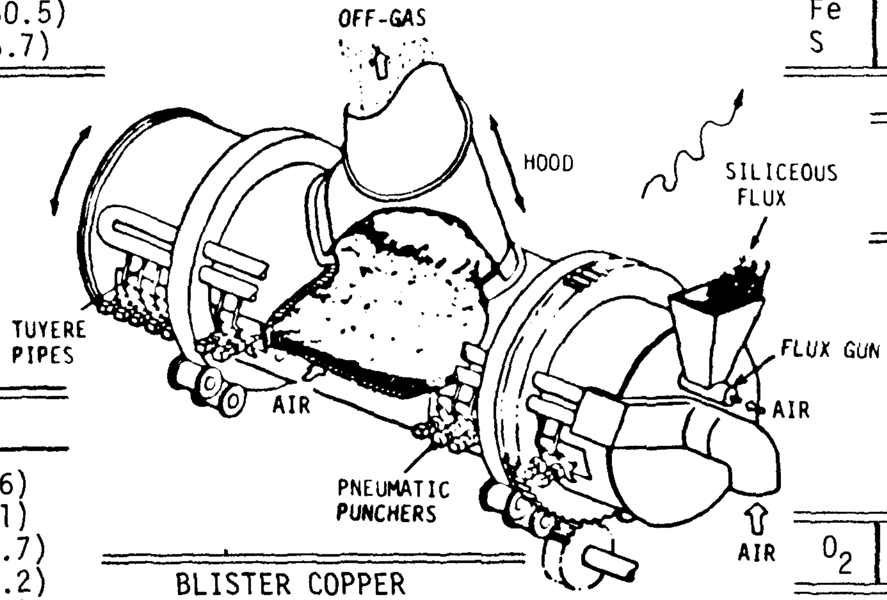
Figure 24. Material balance: Outokumpu flash smelting furnace.[6]

MATTE		
Cu	315.0	(214.0)
Ni	3.5	(2.4)
Fe	25.5	(17.3)
S	89.1	(60.5)
Other	8.4	(5.7)

SO_2	171.5	(116.3)	4% VOL.

FUGITIVES		
SO_2	1.9	(1.3)
Cu	0.1	(0.1)
Fe	0.01	(0.01)
S	0.1	(0.1)

Fe	0.8	(0.5)
SiO_2	35.6	(24.2)
Other	3.2	(2.2)



SLAG		
S	2.3	(1.6)
Cu	11.9	(8.1)
Fe	24.8	(16.7)
SiO_2	35.6	(24.2)
Ni	3.5	(2.4)
Other	10.1	(6.9)

BLISTER COPPER		
Cu	303.0	(205.8)
Fe	1.5	(1.0)
Other	1.5	(1.0)

O_2	86.7	(58.8)

MATERIAL BALANCE IN TONS/DAY.
VALUES IN PARENTHESES INDICATE CAPACITY OF UNIT; OTHER VALUES ARE ON BASIS OF 303 TONS/DAY BLISTER COPPER.

Figure 25. Material balance: Peirce-Smith converter after Outokumpu furnace.[6]

DRIED CONCENTRATES

Cu	308.6	(404.9)
Fe	308.6	(404.9)
S	379.1	(497.4)
SiO_2	102.9	(135.0)
CaO	5.7	(7.5)
Other	38.4	(50.4)

FUGITIVES

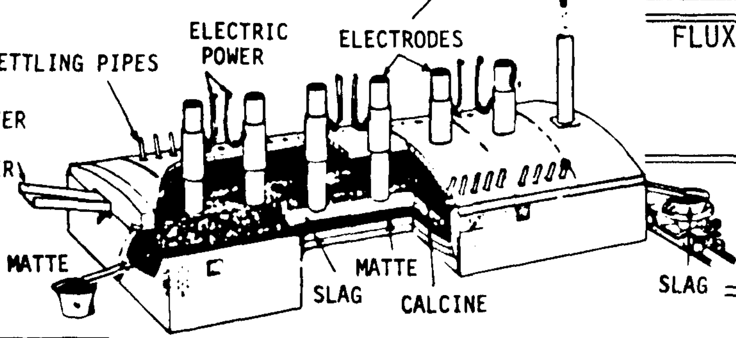
SO_2	11.6	(15.2)
Cu	0.5	(0.7)
Fe	0.5	(0.7)
S	0.3	(0.4)

CONVERTER SLAG

Cu	18.9	(24.8)
Fe	372.9	(489.2)
S	7.7	(10.1)
SiO_2	179.4	(235.4)
CaO	7.8	(10.2)
Other	154.0	(202.0)

SO_2	238.0	(310.2)	5% VOL.

FLUX	SiO_2	32.9	(43.2)
	CaO	48.0	(63.0)
	CO_2	37.3	(48.9)
	Other	9.0	(11.8)



Cu	322.5	(432.1)
Fe	371.2	(487.0)
S	253.5	(332.6)
CaO	5.8	(7.6)
Other	61.0	(80.0)

Air	O_2	123.9	(162.7)

Cu	4.5	(5.9)
Fe	309.8	(406.4)
S	9.1	(11.8)
SiO_2	315.1	(413.6)
CaO	55.7	(73.1)
Other	140.4	(184.2)

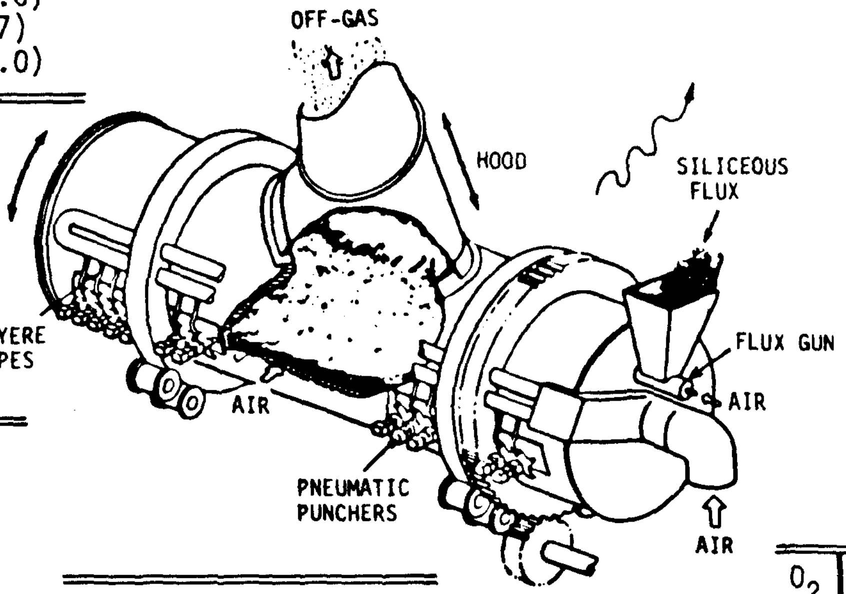
MATERIAL BALANCE IN TONS/DAY.
VALUES IN PARENTHESES INDICATE CAPACITY OF UNIT; OTHER VALUES ARE ON BASIS OF 303 TONS/DAY BLISTER COPPER.

Figure 26. Material balance: electric smelting furnace.[6]

MATTE		
Cu	332.5	(95.4)
Fe	371.2	(109.8)
S	253.5	(75.0)
CaO	5.8	(1.7)
Other	61.0	(18.0)

SO_2	474.5	(140.3)	3.5% VOL.

FUGITIVES		
SO_2	16.5	(4.9)
Cu	0.6	(0.2)
Fe	0.8	(0.2)
S	0.3	(0.1)



Fe	4.0	(1.2)
SiO_2	179.4	(53.1)
CaO	2.0	(0.6)
Other	14.1	(4.2)

SLAG		
Cu	18.9	(5.6)
Fe	372.9	(110.4)
S	7.7	(2.3)
SiO_2	179.4	(53.1)
CaO	7.8	(2.3)
Other	154.0*	(45.6)

*Contains O_2 80.4 (23.8).

BLISTER COPPER		
Cu	303.0	(89.6)
Fe	1.5	(0.4)
Other	1.5	(0.4)

O_2	325.9	(96.4)

MATERIAL BALANCE IN TONS/DAY.
VALUES IN PARENTHESES INDICATE CAPACITY OF UNIT; OTHER VALUES ARE ON BASIS OF 303 TONS/DAY BLISTER COPPER.

Figure 27. Material balance: Peirce-Smith converter after electric furnace.[6]

SECTION 6

CURRENT CONTROLS AND PROPOSED MODIFICATIONS

Control of fugitive emissions at copper smelters currently consists of application of certain fundamental control principles, such as preventive maintenance, use of hooding, and installation of emission collection systems at various stages of the process.

Most plants schedule periodic maintenance of major equipment and perform repairs as needed to correct malfunctions. In addition to these practices, preventive maintenance would involve close attention throughout the daily plant operations to detect potential problems and to remedy defects as they occur. The goal is to prevent catastrophic malfunction or upset, which not only can retard or completely curtail production, but also can cause the release of large quantities of fugitive emissions. Programs of preventive maintenance should be geared specifically to the specific type of plant equipment, with emphasis on prevention or leakage from roasters and furnaces through attention to the condition of refractories, tight fit of doors and covers, and all other areas of the unit that might allow leakage.

The following types of emission collection systems are in current use, either singly or in various combinations:

Fixed secondary hoods

Swingaway and movable secondary hoods

Converter aisle forced-exhaust system with baghouse (i.e., enclosed building)

Air curtains

Roof monitors.

Table 2 briefly describes the design and operation of each of these systems and lists the typical operating and maintenance problems that are encountered. Appendix B discusses air curtains.

Some smelters have introduced innovative control equipment and procedures, and other control strategies are proposed and under investigation. In addition to descriptions of the common

TABLE 2. SUMMARY OF CURRENT FUGITIVE EMISSION CONTROL SYSTEMS

Type	Design and operation	Operational and maintenance problems	Efficiency
Monitor, natural (U.S.)	Simple design; relies on outside air movement for removal of emissions	Haze in building during emissions; outside air movement affects time required to clear the area; crane operator and maintenance personnel working above the converter may be required to wear face aspirators; visible emissions in the monitor area; maintenance in the converter area; EOT and roof trusses for removal of the settled emissions other than gases.	Dependent on outside air currents and inside air motion
Monitor, powered	Simple design; large air movement required at fans; removal rate is constant	Blind pockets or short-circuited flows could cause haze and emission buildup in the roof line area; crane operator may need to wear face aspirator at times; maintenance in converter area, EOT crane, and roof trusses for removal of settled emissions other than gases.	Dependent on number of monitors, fan size, building design above the converter proper; air motions
Fixed hood with secondary emission ducting (U.S.)	Clearance needed for crane hook and cables during collar pulling or matte additions; retrofit difficult for ducting, fans, breeching, and dust bins; operational at all times that converters are on line; good face and capture velocities required.	Operational damage to hood by swinging or uncontrolled EOT crane action during matte addition or collar pulling; maintenance is less in the converter area, EOT crane, and roof trusses	Dependent on distance of the mouth of fixed hood from the emission source; also on capture and face velocity created by the fan at mouth of fixed hood.
Enclosed converter hood swing-away type with fixed hood	Clearance needed for crane hook and cables during collar pulling or matte additions (fixed hood). Clearance needed for floor space relative to fixed hood; rugged drive mechanism needed for swing-away converter hood	Space occupied in aisle by the swing-away converter hood when adding matte, rabbling, or skimming could hamper crane movements; crane must deposit ladles for pouring or skimming and at completion of the operation must await retraction of the hood before engaging the ladle; maintenance of swing-away mechanism and minimal maintenance for removal of particulate buildup that occurs during matte additions and rabbling	Dependent on operational cycle; efficient during pouring, blowing, or slagging; efficiency similar to that of fixed hood during matte addition or rabbling; air motions influence efficiency in all operations
Enclosed building	Requires careful design of all openings (personnel, truck, rail, materials) to minimize air motion; roof monitor must handle all ventilation air for workers; building costs high because of wind load design and need for tightness and close fits	All openings must be maintained constantly against excessive air infiltration; light siding and roofing required; air circulation within building for the workers and process must be carefully controlled; intake and exhaust fans need preventive maintenance; cleanup maintenance for settled particulates in the converter area is similar to that for a monitored system, either natural or powered.	Dependent on building tightness, air motion control, monitor exhaust capabilities

controls in current use at copper smelters, this discussion presents suggested modifications and control measures that could further reduce fugitive emissions. Emphasis is placed on The Peirce-Smith converter, a significant source of uncontrolled or poorly controlled fugitive emissions.

HOODING SYSTEMS

Emissions of sulfur dioxide and particulate from copper smelting are contained to some extent by hooding systems. All of the domestic smelters are equipped with primary hoods, and some also have various types of secondary hoods, which can be very effective over reverberatory matte and slag launders. At some smelters the slag ladles are operated within a partial enclosure with a hood overhead to collect fugitives during slag discharge. Swing-away hoods that are lowered to cover the matte ladles during filling are used at some plants and are effective while the ladle is being filled. When the filled ladle must be removed from under this cover, however, fugitive emissions arise from the matte surface.

Secondary hooding systems are being used also to partially control fugitives from some Peirce-Smith converters. At some smelters a fixed hood (Figure 28) located above the sliding gate is used in conjunction with the primary hood during blowing operations. Although it is somewhat effective during the blowing phases, the hood serves little use in controlling fugitive emissions when the converter is in the "rolled-out" position, i.e., when the gate is in the upmost position and the converter is pouring or receiving charge. Following are details of the various hooding systems.

Fixed Secondary Hood

In the United States the Phelps Dodge Corporation has installed some fixed secondary hood units at their Ajo and Morenci plants. The configuration of a fixed secondary hood depends on the location of the converter relative to the crane runway girders, the configuration of the primary uptake hood, and the requirements for maintenance and operation. The effectiveness of a secondary emission control system is influenced considerably by these factors. Air movement in the converter area and the capture velocity at the face of the secondary hood are also important.

At the Ajo Smelter, the atmosphere of the converter room is relatively clean; the crane operator does not wear a face respirator. Conditions of the converter aisles appear to be less satisfactory at some of the other U.S. smelters not equipped with fixed secondary hooding.

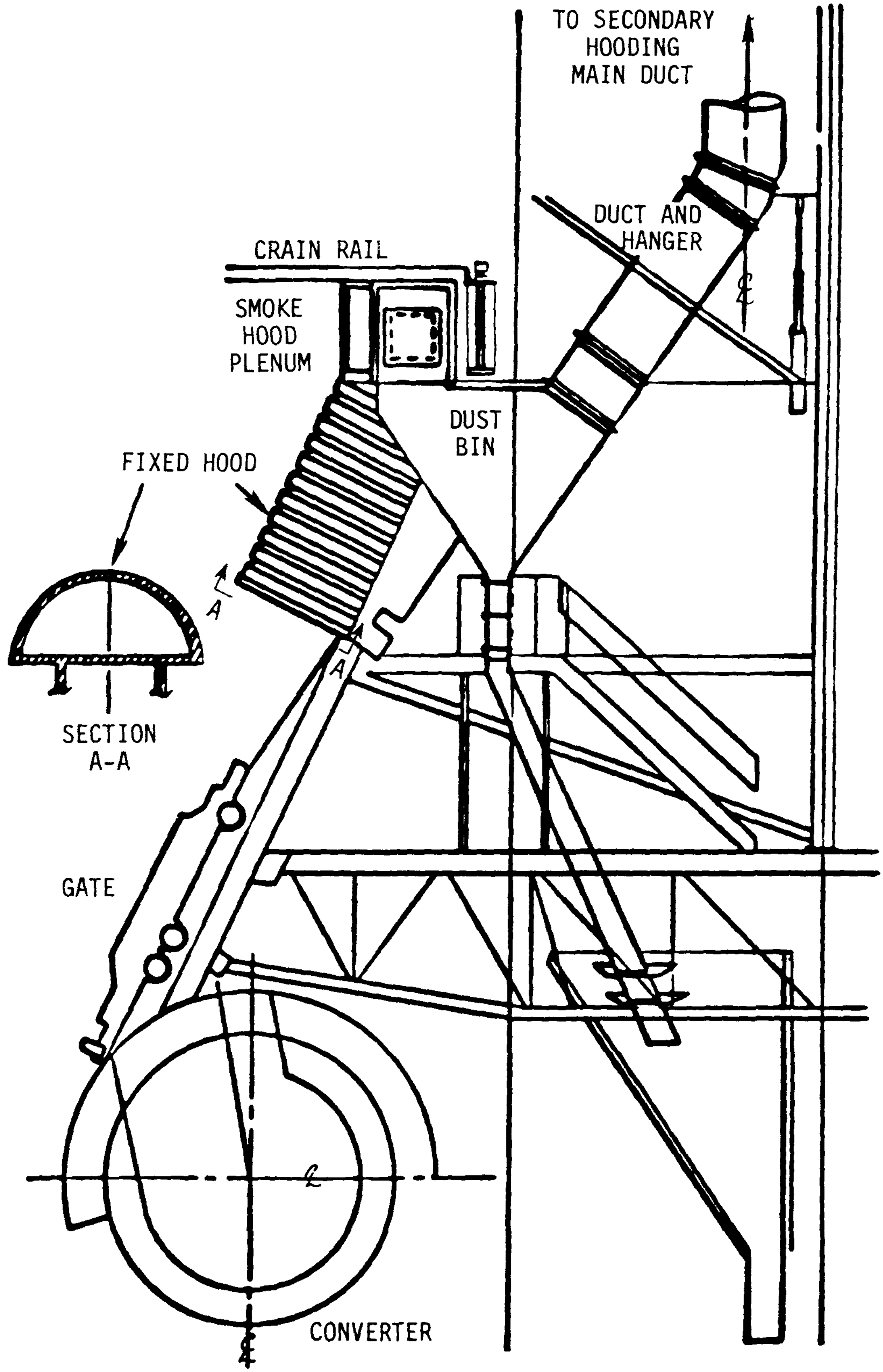


Figure 28. Secondary converter hood configuration.

Figures 29 and 30 show the overall ducting (approximately 210 m) of the secondary emissions system at Ajo. Each converter has two secondary inlets, one on each side of the primary hood.

Each fixed hood (Figure 28) is approximately 4 m long, 6 m wide, and 2 m high. The hoods, which are half-oval in cross-section, are affixed to the upper front sides of the converter primary uptake hoods. Each secondary duct handles approximately 68,000 m^3/h at temperatures of about 93°C.

Requirements at the Ajo plant cannot be regarded as applicable to other smelters, whose layouts may necessitate different quantities and configurations of ducting to contain fugitive emission effectively.

Movable Hoods

Movable secondary hoods are used in conjunction with a fixed hood at several primary copper smelters in the United States.

A type of movable hood system that could be used for control of converter emissions is shown in Figures 31 through 34. The movable hood would be made of steel, elliptical in shape so as to fit over the fixed hood with a clearance of 10 to 15 cm. It would have its own track for movement.

In the retracted position, the movable hood should not extend farther horizontally than the fixed hood. In the extended position, it would mate with the lip of the fixed hood to provide continuity of ducting of secondary emissions. The movable hood would have its own retracting and lowering mechanisms, which would be controlled by the converter operator.

The hood attached to the gate also would be a type of movable hood. The gate hood (Figures 33 and 34) would be of steel construction and possibly elliptically shaped to fit under the movable hood when in a retracted position. Again, as with the movable hood, the gate hood would be protected by the fixed hood in its up position, i.e. no part of the retracted gate hood would extend horizontally beyond the fixed hood (Figure 34).

In the extended position, the gate hood would continue the ducting of secondary emissions to the movable and fixed hoods.

Swing-Away Hood

A brochure of the Nippon Mining Company shows a deflector converter hood of the swing-away type with a retractable secondary hood above (Figure 35). In the foreground, the deflector hood is shown in the operating position during a blister copper pour. The emissions are deflected into the retractable hood.

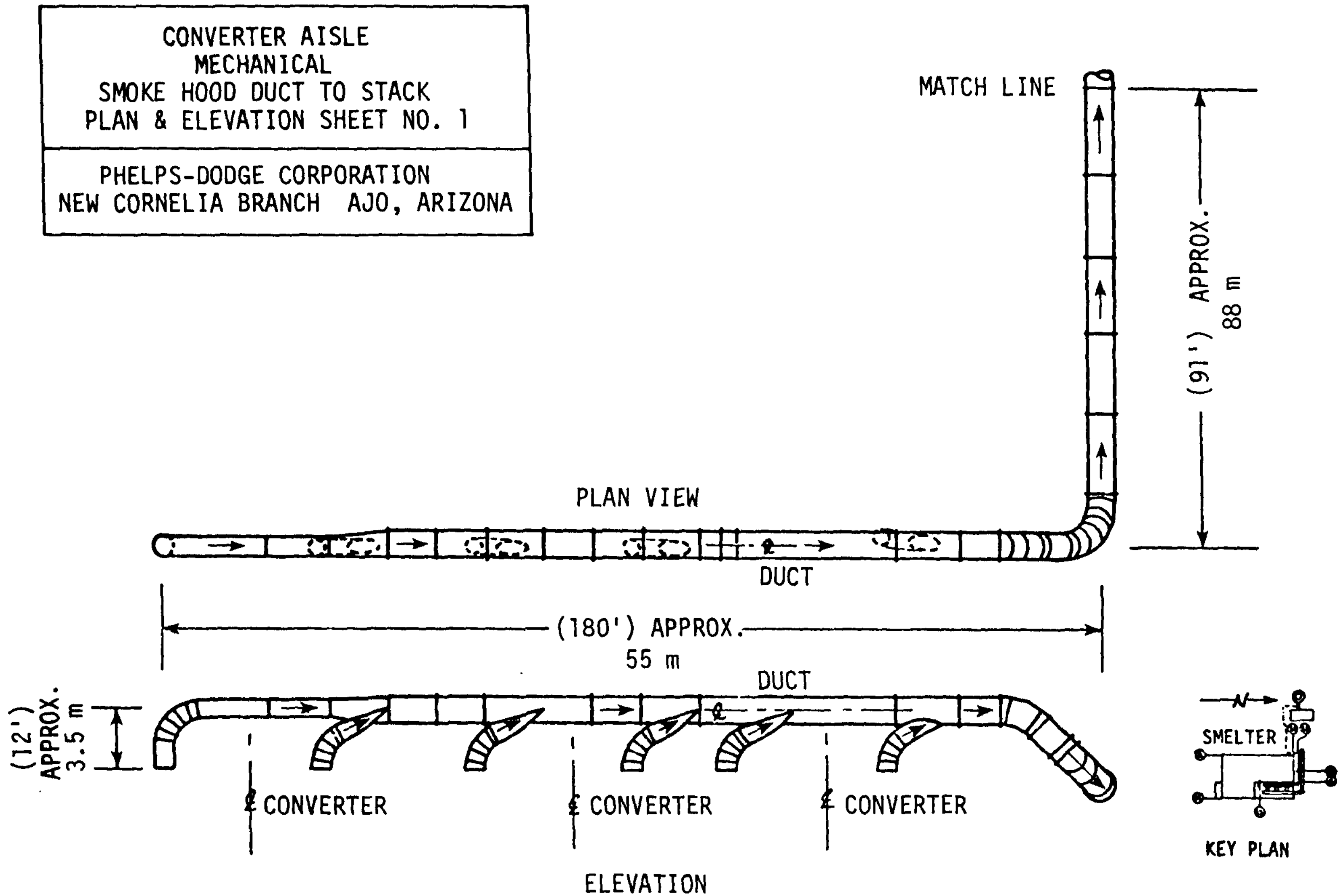


Figure 29. Ajo Smelter, secondary emission collection system, sheet 1 of 2.

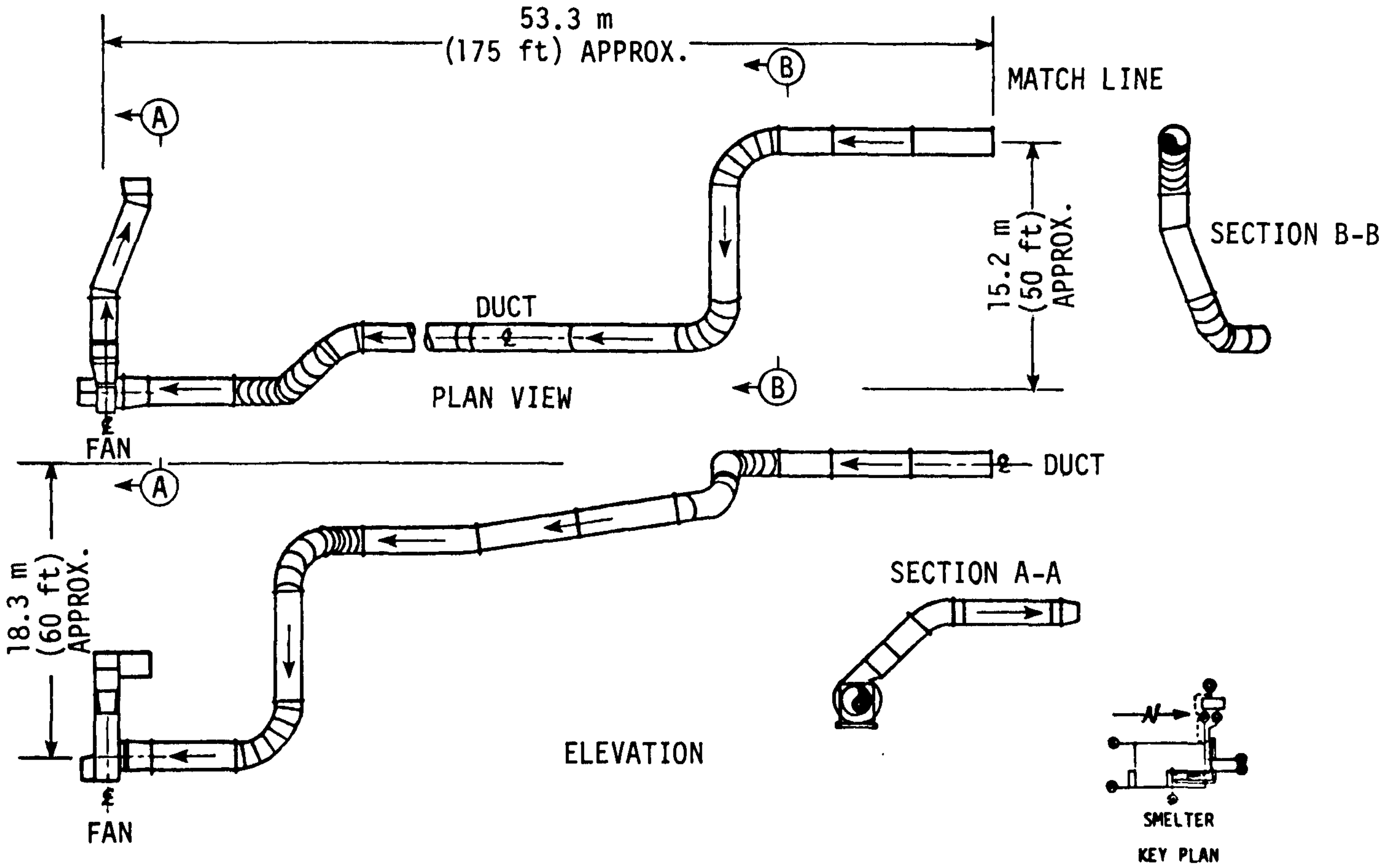


Figure 30. Ajo Smelter, secondary emission collection system, sheet 2 of 2.

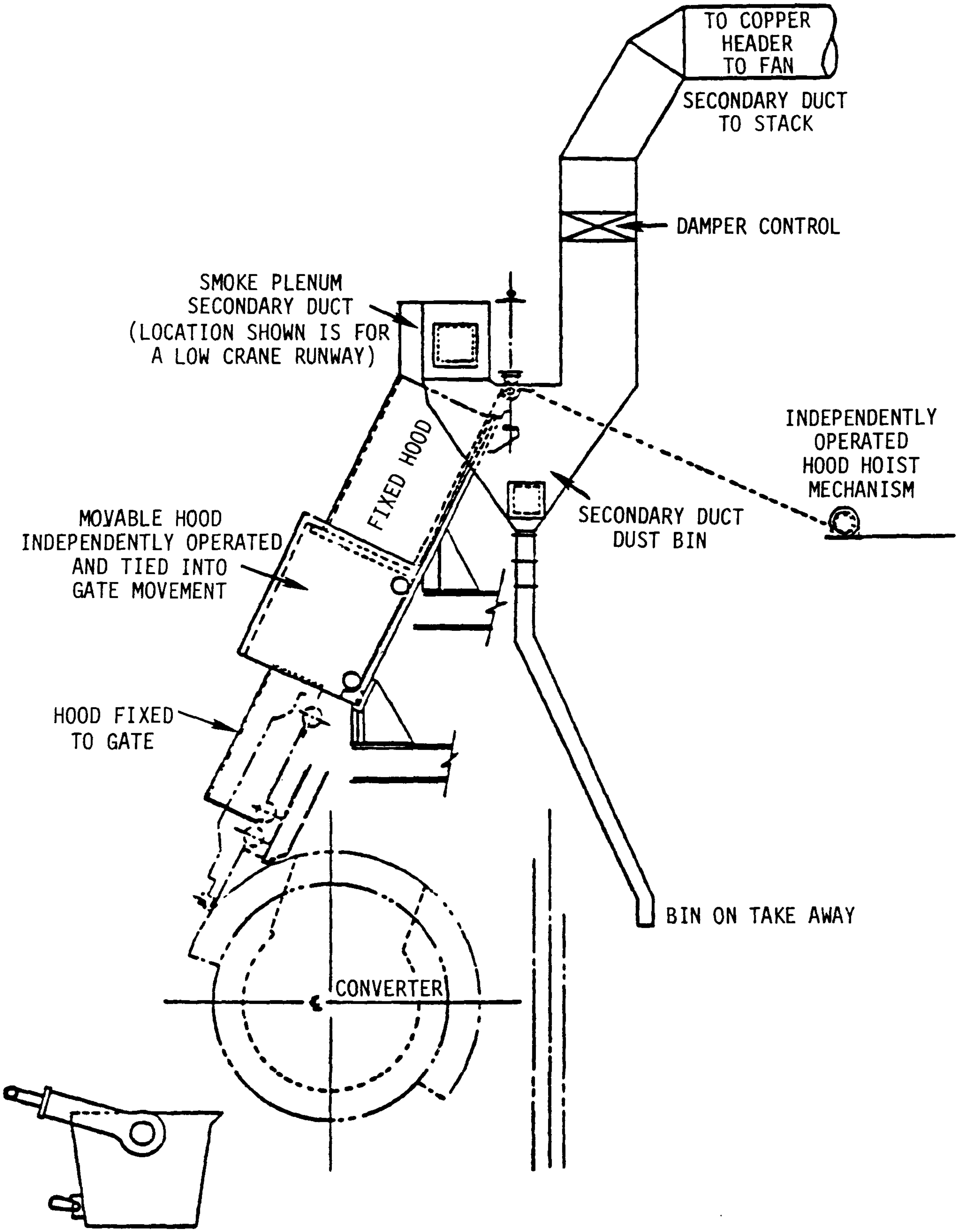


Figure 31. Side view of Peirce-Smith converter with hooding in position.

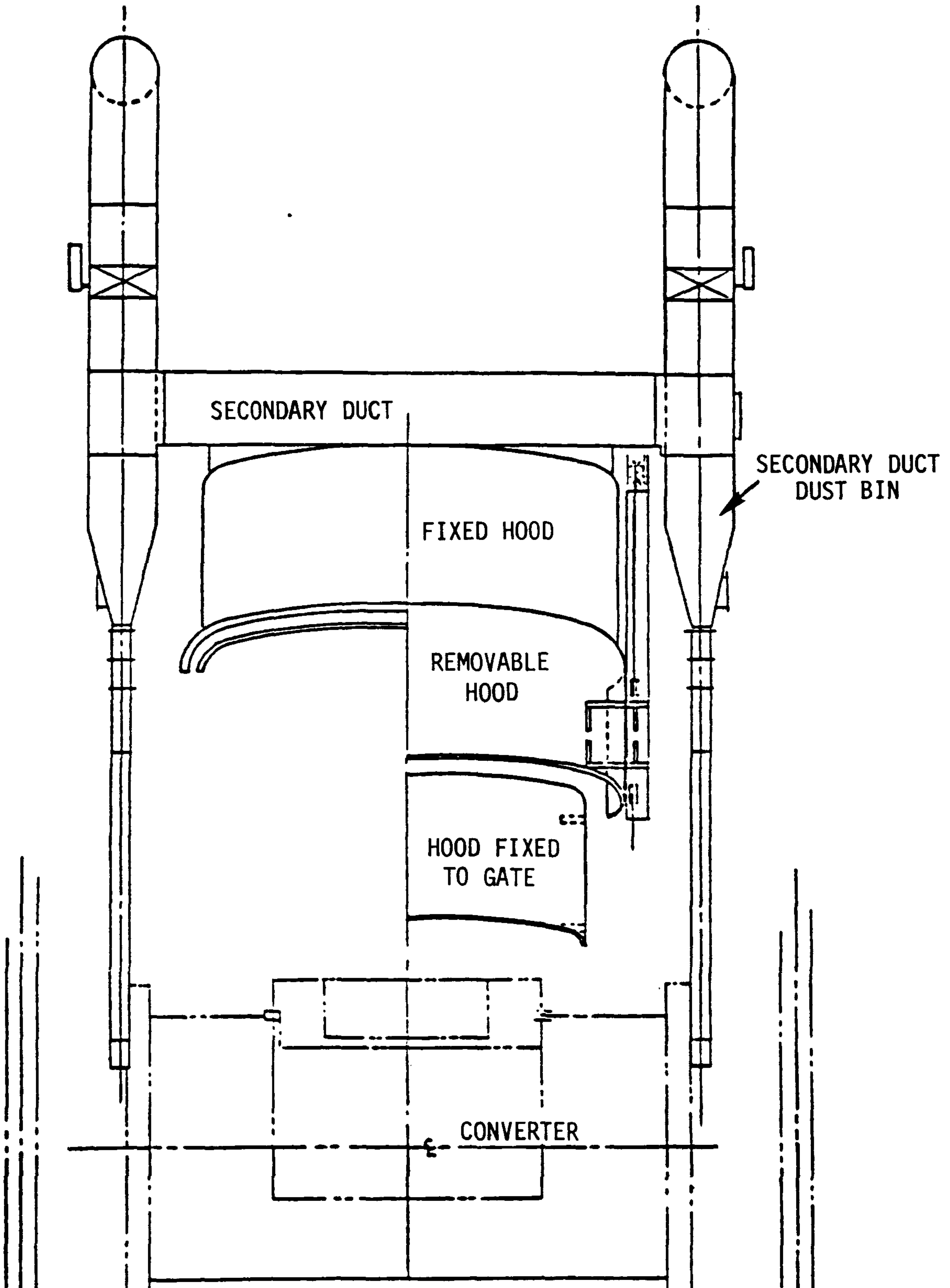


Figure 32. Front view of fixed, movable, and gate hoods.

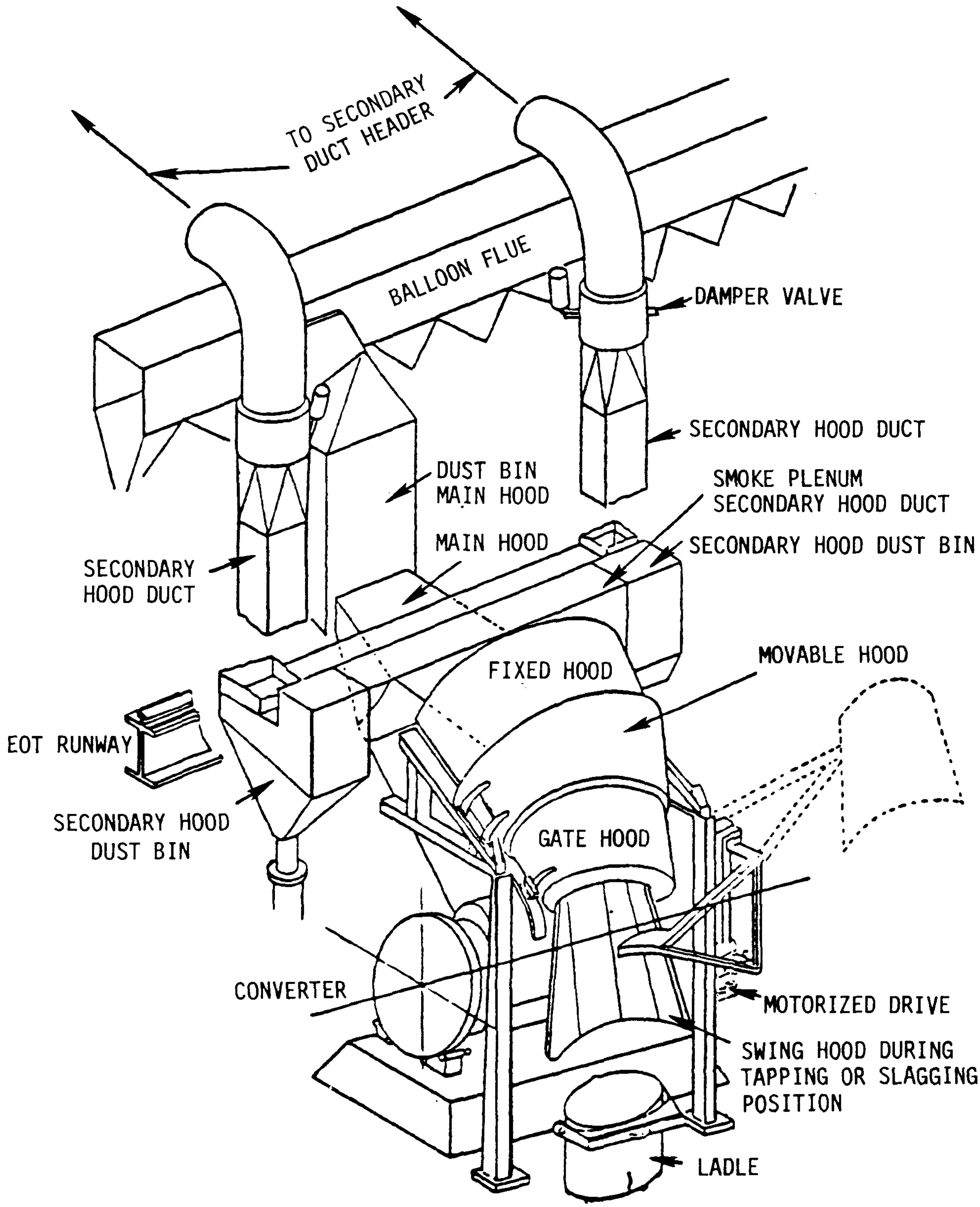


Figure 33. Peirce-Smith converter with hooding extended.

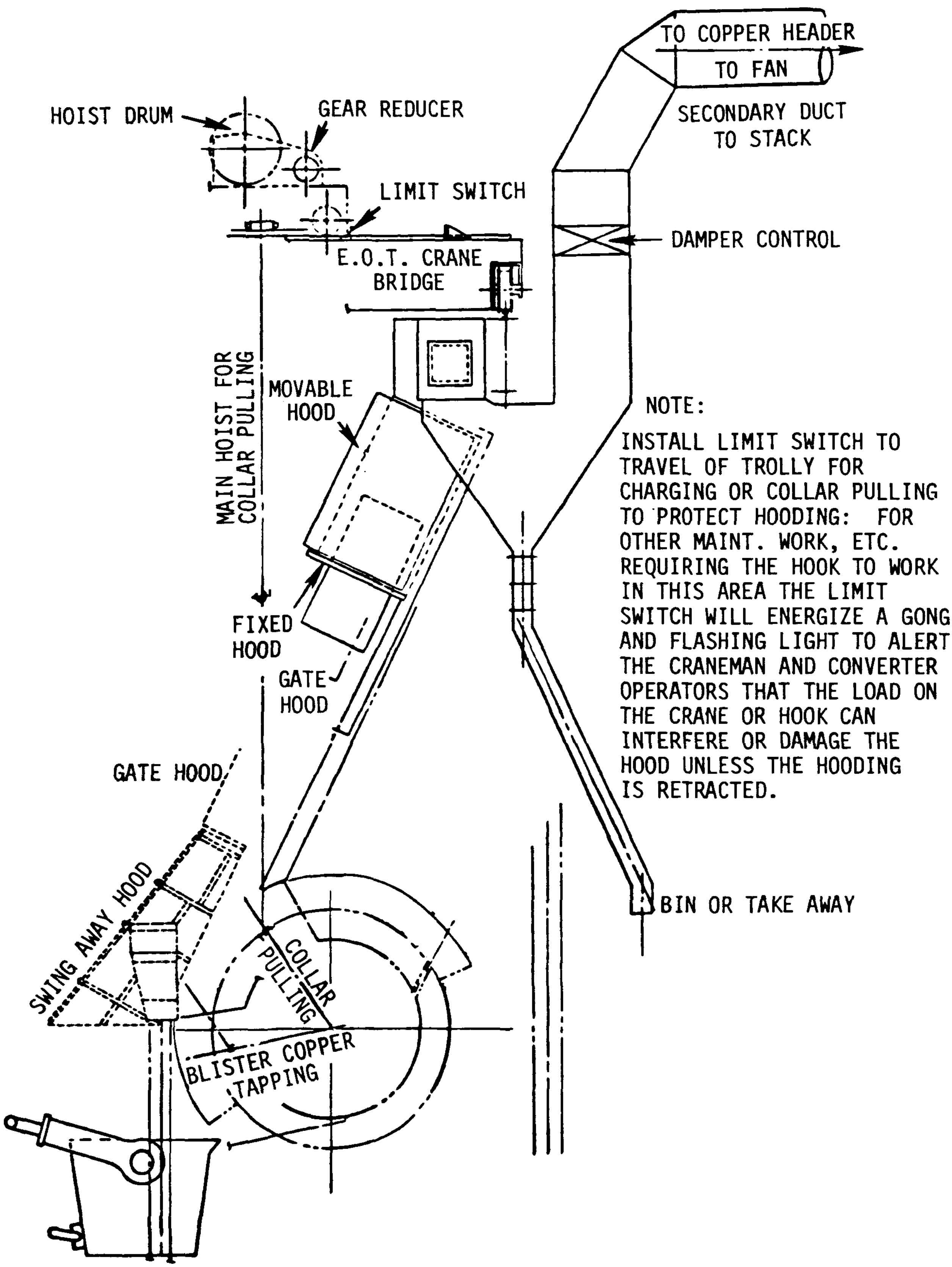


Figure 34. Side view of Peirce-Smith converter during collar pulling or blister copper ladle removal, with hooding retracted.

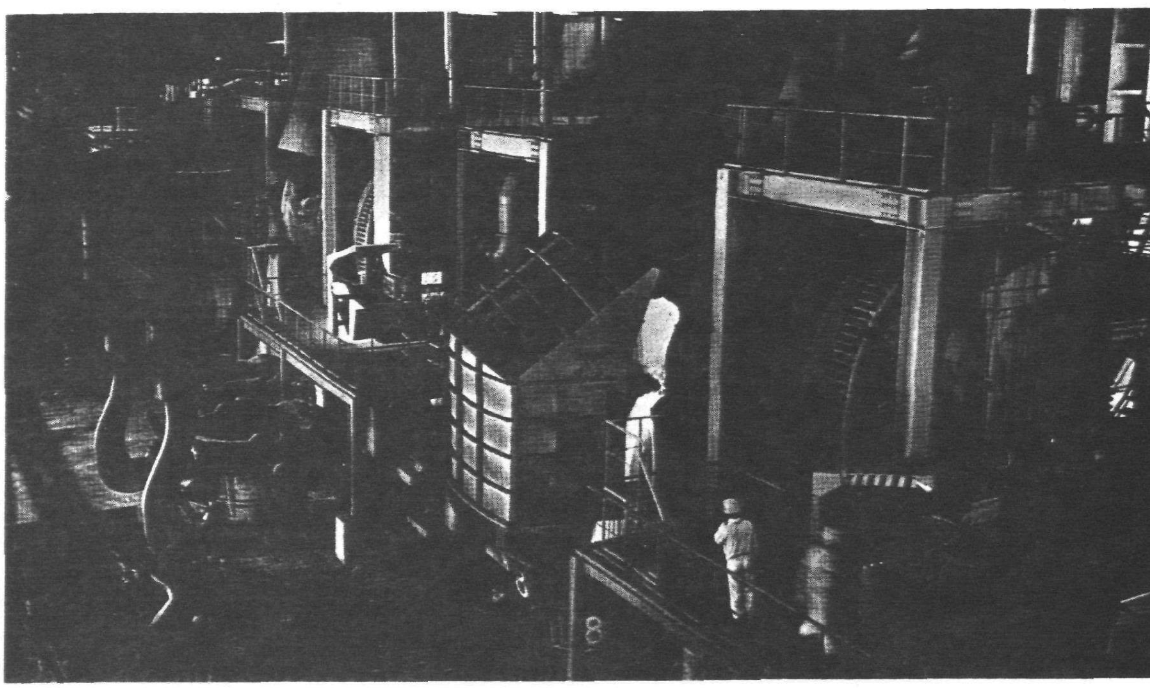


Figure 35. Enclosed swing-away converter hood of Nippon Mining Company.

Movable hoods must be retractable to a position that does not interfere with the overhead crane or plant operations and that is not subject to abuse from moving objects, such as the EOT crane hoist block or cables, charging ladles, or the collar-pulling rig.

Some operating procedures and system design factors for secondary hood systems are discussed below. These procedures and designs should minimize damage, breakdowns, and delays and should optimize working conditions for plant employees.

The movable, swing-away, and gate hoods should be under control of the converter operator. The EOT crane operator and converter personnel should be aware whenever the EOT crane hook, cables, and hook load (ladle, collar puller, etc.) pass into the area adjacent to the hoods as indicated at the Nippon plant in Figure 35. At such times the trolley of the EOT crane could trip a limit switch to energize an intermittent sounding horn and/or flashing lights on the underside of the EOT crane cab stairs. These warning signals would operate as long as the trolley of the EOT crane is in the area of the hoods. The horn signal should be of such intensity and sound as to be distinquishable from other EOT crane or maintenance sirens. Whenever this sound is heard, one of the converter personnel would check to see what hoods may need to be retracted.

At the end of each shift, the departing EOT crane operator would check the warning or interlock system by a trial run of the trolley into this converter area and record its condition on his daily check list. Before putting the EOT crane into service, each incoming crane operator would also check out the warning system.

For matte additions and collar pulling procedures, the converter operator would position the converter gate and retract all the hoods. After the matte addition, the converter would be rotated to its blowing position in the primary converter uptake hood. All hoods would be extended into position; blowing would commence, and flux could be added to the converter as required.

For slagging or skimming operations, the swing-away hood would be retracted. In the event of interference between the EOT crane hook or cables and the gate hood, the gate hood too must be retracted. After the ladle is removed or repositioned, the EOT crane would be backed off and the appropriate hoods positioned in place. The same movements of the hoods would be required in blister copper pouring for the positioning or removing of a ladle.

Positions of the secondary hoods during the various converter operations are shown in Table 3. Estimated efficiencies of the different types and combinations of hoods during the various converter operations are shown in Table 4. The valves

TABLE 3. POSITIONS[a] OF MOVABLE HOODS

Type	Matte addition	Blowing or holding	Skimming	Rabbling	Collar pulling	Pouring
Movable	Retracted	Extended	Extended	Partially or fully extended	Retracted	Extended
Gate hood	Retracted	Extended	Extended	Partially extended	Retracted	Extended
Swing-away	Retracted	Retracted or in operating position	Operating position	Retracted	Retracted	Operating position

[a] Definitions of hood positions:

Retracted - hood in its highest or extreme position away from the converter.
Extended - hood in its lowest position.
Partially extended - hood extended as far as practical to maximize secondary emissions control.

TABLE 4. PEDCO'S ESTIMATED RETROFITTED HOODING EFFICIENCIES[a]
(values in percent)

Hood type	Matte or hot metal addition	Blister or hot metal pouring	Skimming or slagging	Blowing
Fixed	30-50	30-50	30-50	60-70
Fixed and movable	30-50	40-70	40-70	70-90
Fixed and swing-away	30-50	80-90	50-70[b]	80-90
Fixed, movable, and swing-away	30-50	80-90	60-80[b]	80-90
Enclosed building	50[c]-95	50[c]-95	50[c]-95	50[c]-95

[a] Most system efficiencies would be higher if air motion (i.e., open doors, man-cooling fans, monitors, etc.) could be eliminated. Skimming is removal of slag from the converter by tilting of the converter. Slagging is removal of slag from the converter by tilting of the converter and manual use of a rake to work the molten bath.

[b] Efficiency during slagging would be similar to that in blister pouring.

[c] Low efficiency due to air motion; with inadequate design of monitors and airflow, efficiency could be as low as 75%; with doors left open efficiency could drop to 50%.

for collection efficiency are based on retrofit installations. Collection efficiencies would be greater if the installation were incorporated into design of a new plant.

ROOF MONITORS

The standard design in buildings where emissions can cause operational problems usually consists of monitors at the peak of the roof. A monitor runs the length of building to permit convective removal of the emissions, which rise slowly. As an alternative, smaller fan-powered monitors are sometimes installed above each emission source.

As the emissions drift upward, they may impair the visual or respiratory functions of the EOT crane operator; for this reason he may wear respiratory equipment or the EOT crane cab may be air conditioned. The heavier particles usually settle out on process equipment such as the primary uptake hood and EOT crane, and on structures such as runways and roof trusses.

Fugitive emissions may cause buildup of haze in the upper portion of naturally ventilated buildings. Even if powered monitors are used, pockets of dead air and haze may form at certain locations.

These monitors could be tied into a collection system by enclosing the sides of the roof monitors and ducting to induced-draft fans and baghouses. The powered monitor units could also be enclosed and ducted similarly.

BUILDING ENCLOSURE

Because of the problems with the currently used hooding systems, which are ineffective when the converter is pouring or receiving charge, the concept of total building enclosure has arisen. One smelter is experimenting with total enclosure of the converter building, discharging the fugitive emissions by means of a powered roof monitor system. This approach, however, entails some problems. Currently, only three of the five sectioned roof monitors at this smelter are powered; the other two sections are gravity-type monitors. Also, when building access doors are left in the open position, the design concept of total building enclosure is not realized. With proper design, maintenance, and cooperation of plant personnel, collection of converter fugitive emissions by total building enclosure could be very effective. Proper design would involve such factors as tightness of the building, capacity of the takeaway fans, movable truck and rail doors, lighting, and ventilation throughout the entire building to ensure worker safety. Maintenance, particularly of the air moving equipment, would be important to minimize problems with system ventilation imbalances. Training and daily performance of employees would be very important to ensure that doorways, louvers, or other openings are kept closed at all times when not in use. Such careful practice by plant personnel would minimize disturbance of airflows.

The merit of total converter building enclosure is the possibility that such a system could capture nearly all of the fugitive emissions. The scheme entails several problems, however. Design of a total evacuation system that could provide the proper air changes in all working areas would be difficult. Even with an all-powered roof monitor system to pull the air through the building, pockets of dead air are likely, especially in corners and around objects that obstruct the airflow. With a totally enclosed building, air movement must be properly distributed, even in the difficult areas such as around the converters, crane runways, EOT crane repair areas, and converter aisles. Building access doors and mandoors must be opened at times to permit movement of materials and personnel. Proper ventilation of a totally enclosed building may require the use of air ducts to convey air to specific locations. Such a system would be "short-circuited" by passage of air through opened doorways, and disruption of the airflow patterns would reduce the effectiveness of the ventilation system in removing fugitives and changing the building air.

Other problems with total building enclosure are difficulty of retrofit and high cost. Building structure and support may have to be reinforced to handle the stress of added side and end walls and a roof monitor system. Capital cost of totally enclosing an existing converter building could be very high.

AIR CURTAINS

Mitsubishi Metal Corporation at Onahama, Japan, controls some of the fugitive emissions from a Peirce-Smith converter by the use of secondary hooding and an air curtain. (Details are given in Appendix B.) This technique could be modified for application to multihearth roasters, reverberatory furnaces, and other smelter process equipment.

Fugitive emissions at the Onahama smelter are controlled by a combination of secondary hooding, air curtains, and building enclosure and evacuation. Visual observation indicates that the system is 80 to 90 percent effective. With additional building evacuation, a 90 percent capture level could be maintained. The present system does not impede converter operations. Although visibility from the EOT crane cab is not seriously impaired by gases and particulate, additional building ventilation could improve visibility and reduce potential exposure in the EOT crane cab.

A proposed secondary hood system incorporating the air curtain technique (Figure 36) would consist of steel side panels with a back and top panel, forming a partial enclosure. The front, or side at which the crane with ladle approaches the converter, is open. The top panel would have an opening sufficient to permit cables from the EOT crane to pass into the enclosure without damaging the structure or crane cables when charging matte, removing ladles of slag or blister copper, or performing other operations.

Air is blown from one side of the top panel opening and collected on the other side of the opening. The air flow rate across the opening is 1000 Nm^3/min. The air entrains the rising fugitive emissions, then is discharged to a collection system.

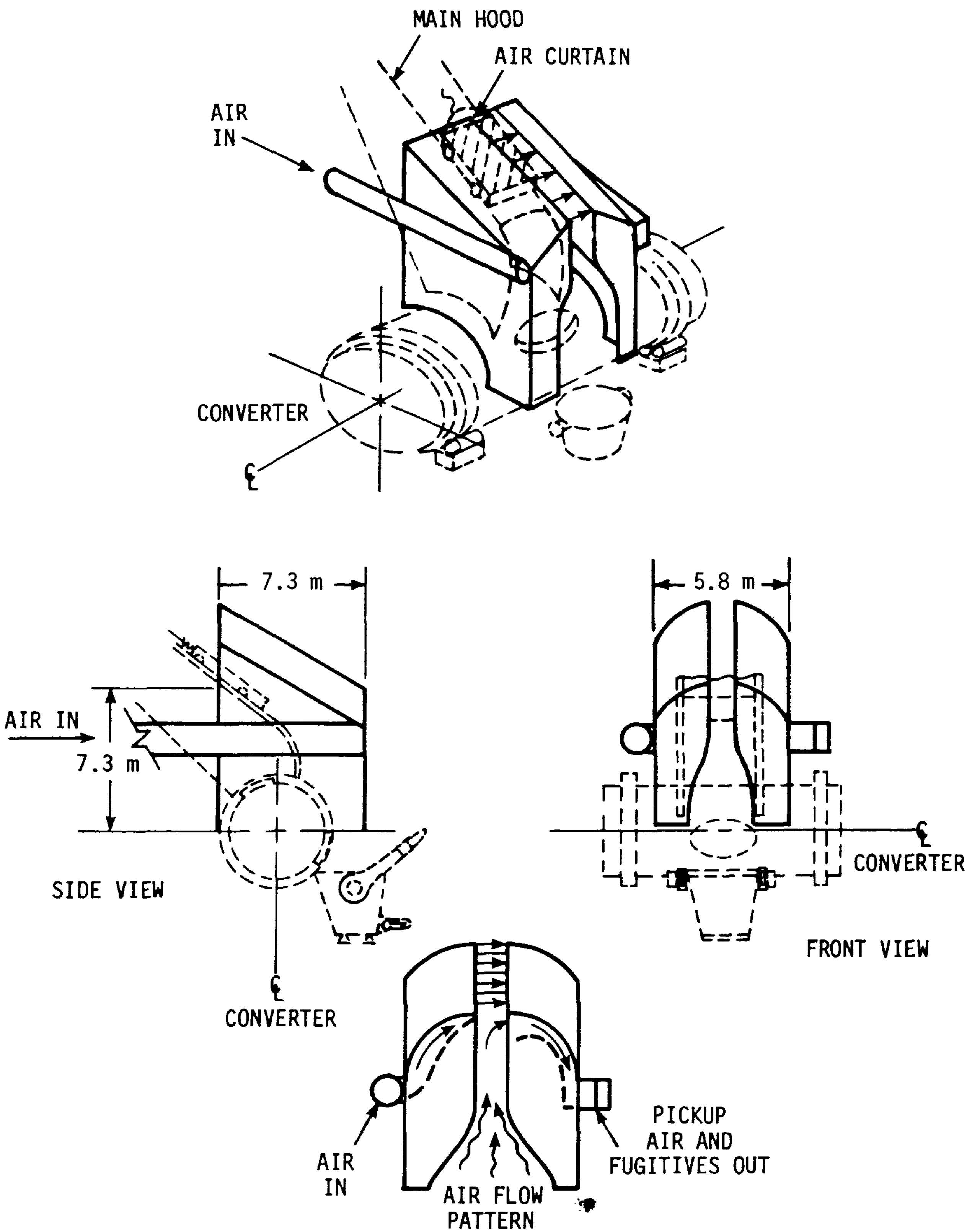


Figure 36. Air-curtain fugitive control system.

SECTION 7

PROPOSED ALTERNATIVE CONTROLS AND PROCESS CHANGES

This section describes some of the alternative process technologies that could reduce the fugitive emissions from copper smelting operations. One alternative system involves the use of a cascading arrangement, in which matte from a smelting furnace (Noranda, Outokumpu, electric arc) is discharged by gravity via launders with hoods to a holding furnace, and then by gravity via launders or runners to a Hoboken converter with a swing-away secondary hood (for use during slagging or blister copper pouring). Another alternative is use of a furnace similar to a Q-BOP furnace in the steel industry. Emissions from ladles could be controlled by use of an EOT crane evacuation system or by use of bottom-pour ladles with covers. These and other possible means of fugitive emissions control are described below.

CASCADING SYSTEM, STAGGERED SYSTEM, AND INDUCTION PUMPING

A cascading system, unlike other systems discussed in this section, would require a change in the traditional smelter layout but should nearly eliminate all sources of fugitive emissions. In a cascade smelter design the roasters (if used), smelting furnaces, converters, and anode or refining furnaces would be arranged to receive products or slags by way of covered launders, runners, refractory-lined pipes, or similar equipment, without the use of ladles (Figure 37). Green feed would be transported to the roasters, and calcined feed would be screw-conveyed or possibly conveyed pneumatically to a storage bin and then to a smelting furnace. Matte could flow by gravity from the smelting furnace to a holding furnace and then to an anode furnace. The furnaces would be arranged in a stepped manner. Slags from the smelting furnaces could be tapped and disposed of as is currently done.

A cascading system would require the lifting of raw materials a considerable height and would require a building somewhat higher and wider in some areas than those in current use. It also would require additional energy to transport the materials initially to a higher level. Operational procedures would be changed in that less movement of the EOT cranes would be required. The use of intermediate holding vessels would be

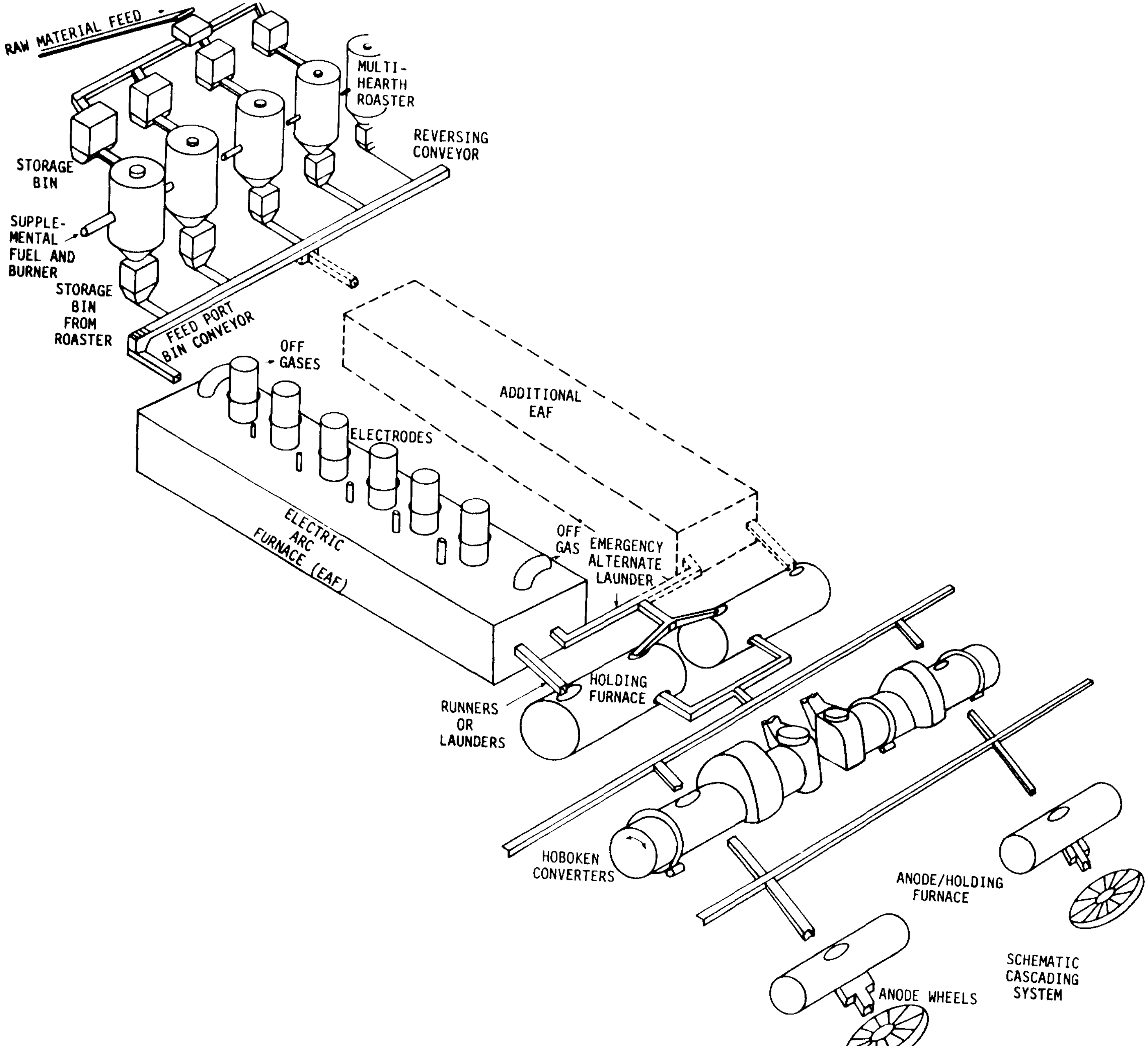


Figure 37. Cascading gravity flow.

expanded, and furnaces would be aligned in a fanning layout rather than in rows. Transportation from unit to unit would be by launders, except for charging to the first unit in the process. Maintenance and operations could be somewhat restricted by the proximity of the various process flows. Overall, control of emissions would be facilitated by the holding furnaces and elimination of the EOT crane for transport, which would eliminate process holdups. A pendant floor-controlled EOT crane would be installed over each process furnace for use in emergencies and for cleanup, repairs, and maintenance. Such a system using evacuated canopied hoods (Figure 38) would minimize fugitive emissions.

In operation of a new smelter designed with the cascading arrangement, problems would arise because of the new operating techniques. Moreover, one could expect buildup of accretions in the transfer launders, and similar operating problems. Such difficulties are common in operation of systems based on new technology; with a problem-solving approach, the system designers and operators could determine and implement the needed modifications.

As an alternative to the cascade design, a horizontal layout could be designed for use with induction pumping to move hot fluids vertically or horizontally without the use of ladles and without open exposure of the fluids. The induction pumping would require large expenditures of power to move the materials from one point to another, but control of gases and fugitive emissions could be achieved with the relatively simple control equipment. The envisioned fugitive emission control system would consist of hooding at the transfer points with the necessary ducting, fans, and baghouse (Figure 39).

Operation of the induction pumping system would require a building similar in height to the current structures but slightly wider. The basic process furnaces would be at ground level, as at present. Again the furnaces would be arrayed in a staggered pattern rather than in a line. The operator would control the flow to and from adjacent holding vessels; EOT cranes would not be needed for handling of ladles. as with the cascade system, an EOT pendant floor-controlled crane over each furnace would be used for cleanup, repairs, and maintenance. Control of the process and of emissions would be facilitated, but power consumption, equipment costs, and maintenance requirements would be greater.

OXYGEN ENRICHMENT

When a Peirce-Smith converter is blown with air, the oxygen in the air reacts with sulfur in the matte to form SO_2 and liberate heat. Besides removing the undesirable sulfur as SO_2,

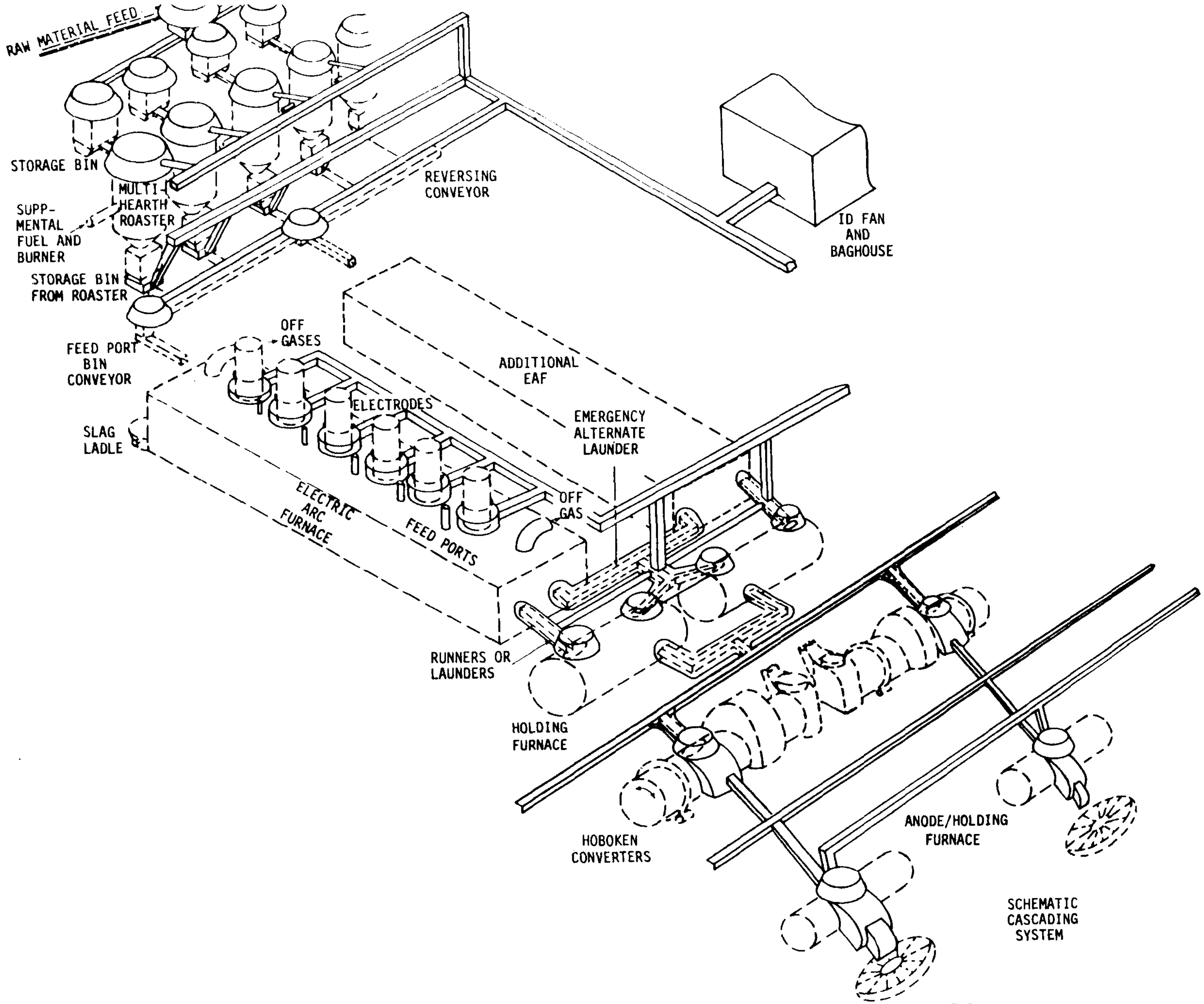


Figure 38. Fugitive emission collection system for cascading gravity flow.

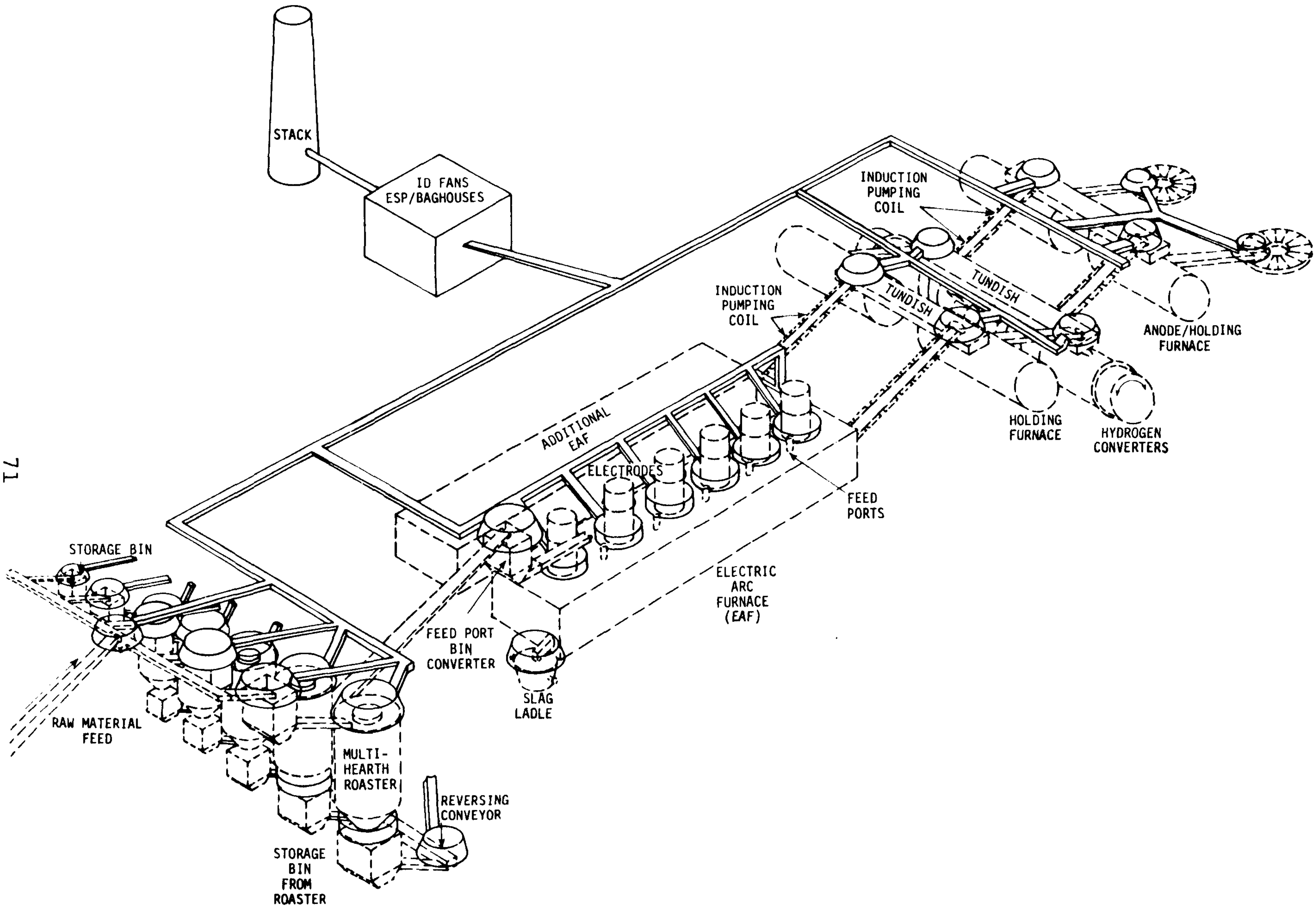


Figure 39. Fugitive emission collection system for cascading/induction/gravity flow.

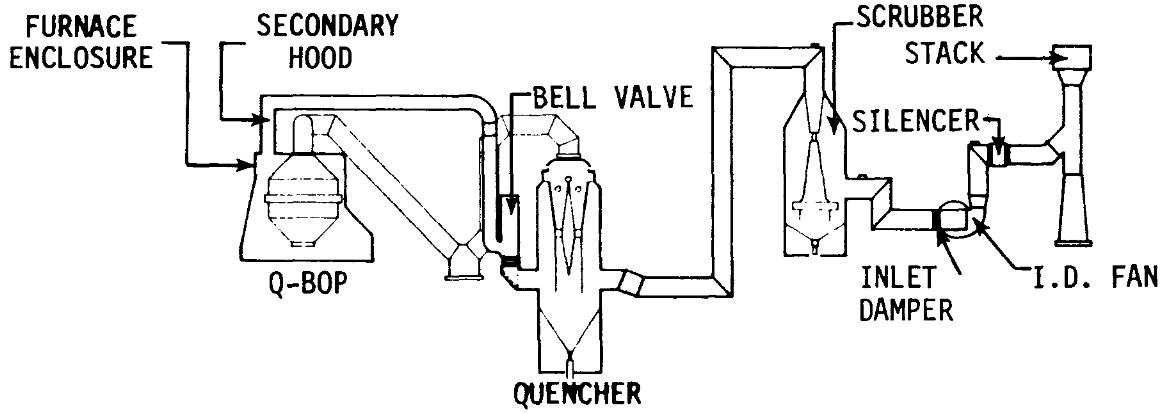
oxygen also reacts with iron in the presence of silica to form the slag. Large amounts of air must be blown into the converter to provide enough oxygen for these reactions. The blowing rate also must be high enough to produce the blister copper in a designated time cycle. Release of fugitive emissions through clearances, such as that between the primary hood and converter, depends partially on blowing rate, the emissions increasing with increasing rate of blowing.

With oxygen enrichment, the blowing rate to a Peirce-Smith converter could be reduced, with resultant reduction of fugitive emissions. Enrichment of blowing air with 5 percent oxygen would theoretically allow reduction of a 68,000 m^3/h blowing rate by about 2700 m^3/h. In addition to reduction of fugitive emissions, the reduced blowing rate of air with higher concentrations of oxygen also increases the SO_2 concentration in the gas stream to an acid plant.

Q-BOP FURNACE

The Q-BOP (Figure 40) furnace used in the steel industry could be applied as a converter furnace at a copper plant, with oxygen or O_2-enriched air as the blowing medium use. Use of the Q-BOP furnace in the steel industry has both increased production and reduced production costs. In addition to these advantages, the Q-BOP can be operated in an enclosure or "doghouse" to prevent escape of fugitive emissions. Such a furnace with fugitive emissions control system is installed and operating at the South Chicago District of Republic Steel Corporation. A schematic of that furnace and its gas cleaning system is shown in Figure 40. Where open hearth (reverberatory type) furnaces were used without oxygen enrichment, the production rate was 23 Mg of steel per hour; with oxygen enrichment production rose to 40 Mg per hour. With the Q-BOP furnace, production has increased to 163 Mg per hour.

Secondary emissions still must be controlled around the other process furnaces. Use of this system at copper smelters might permit the use of fewer converters and could yield a stronger SO_2 stream, but would require holding furnaces for good process control. Because refractory lining would be required, two units would be needed to maintain continuous operation. Maintenance requirements would be somewhat greater with the refractory-lined unit, in which temperatures may be slightly higher. The building structure could be smaller than those in current use.



GAS CLEANING SYSTEM

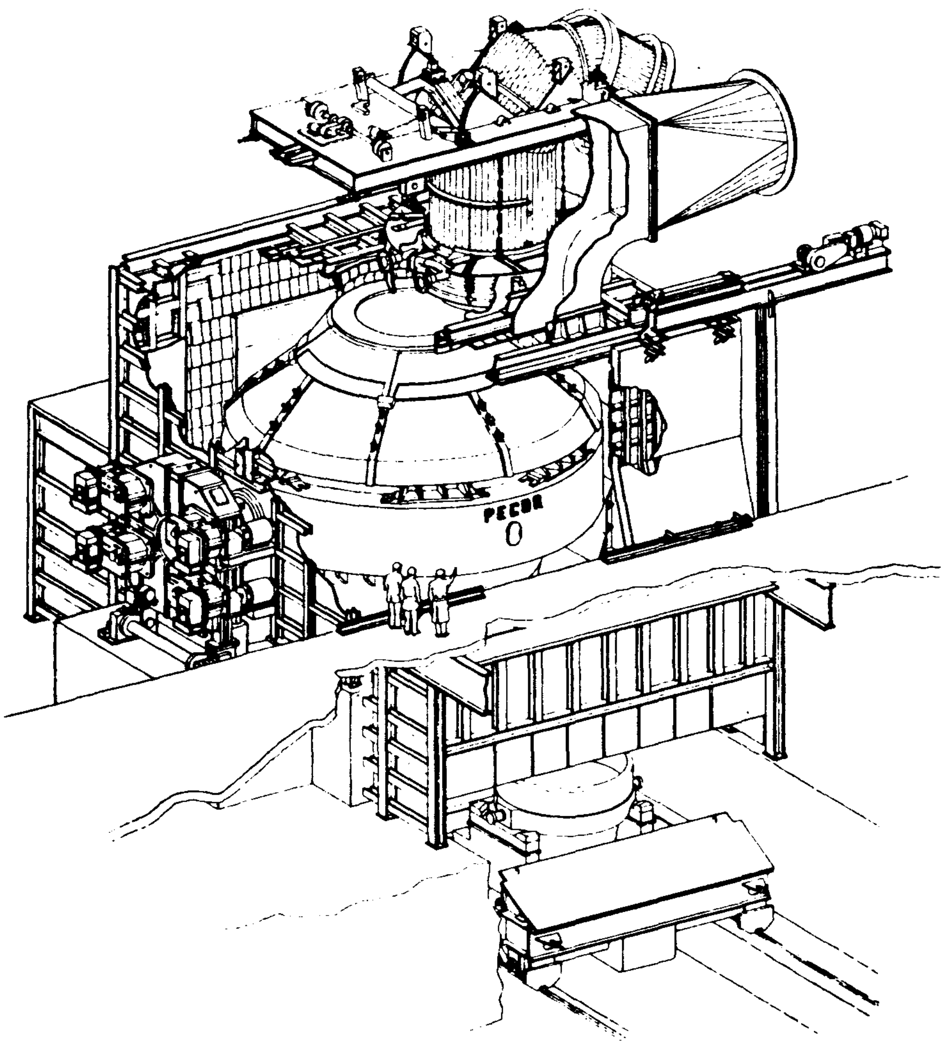


Figure 40. Q-BOP furnace enclosed in a "doghouse" to prevent fugitive emissions (from Iron and Steel Engineer, November 1978).

CRANE EVACUATION OF LADLE EMISSIONS

The EOT could be modified to minimize fugitive emissions from ladles during transport, filling, and pouring by use of a capture hood fixed to the spreader beam (Figure 41). The capture hood would also be fixed to a sectionalized telescopic column, similar to that used in the steel industry's stiff leg EOT crane for moving ingots to and from a soaking pit.

An exhaust fan would be mounted on the trolley and connected with ducting to the telescopic column duct. The EOT crane would position itself over the ladle or slag pot and as the molten materials are discharged into the receptacle, the evacuation blower on the trolley would draw the emissions to the trolley deck. This stream could be discharged to the overhead building monitor or ducted to the crane walkway and then to the building column line for discharge to a collector system. The evacuation system would operate during transportation and pouring of the ladles. Maintenance requirements could be high if the telescopic unit were damaged by poor crane operations.

FLOOR-OPERATED CHARGING

A floor-operated charging machine, illustrated in Figure 42, could replace the EOT cranes that transport ladles of matte or slag. A charging machine could lift a ladle of matte, move back and pivot 180 degrees, then move the ladle of matte to the converter. The arms would life the ladle to the desired height and tilt it for discharge of the matte into the converter. The converter would be contained or totally encapsulated within a hood to capture the fugitive emissions. A portion of the hood for the converter would be retractable for maintenance when the EOT crane must be used. Use of the floor-operated charging machine would be limited to new installations and the converter aisle would be wider than those in the present converter buildings.

Emissions escaping the primary hood would be ducted from the top of the encapsulated structure. This floor-charging technique would minimize fugitive emissions in the converter aisle during charging.

TOP-COVERED BOTTOM-POUR LADLES

Whether EOT cranes or charging machines are used, the transport of molten materials by means of open ladles generates visible fugitive emissions. These ladle emissions could be reduced by use of fitted covers that could be placed on or removed from the ladle with minimal effort (installation of such covers could pose some problems at some existing plants.)

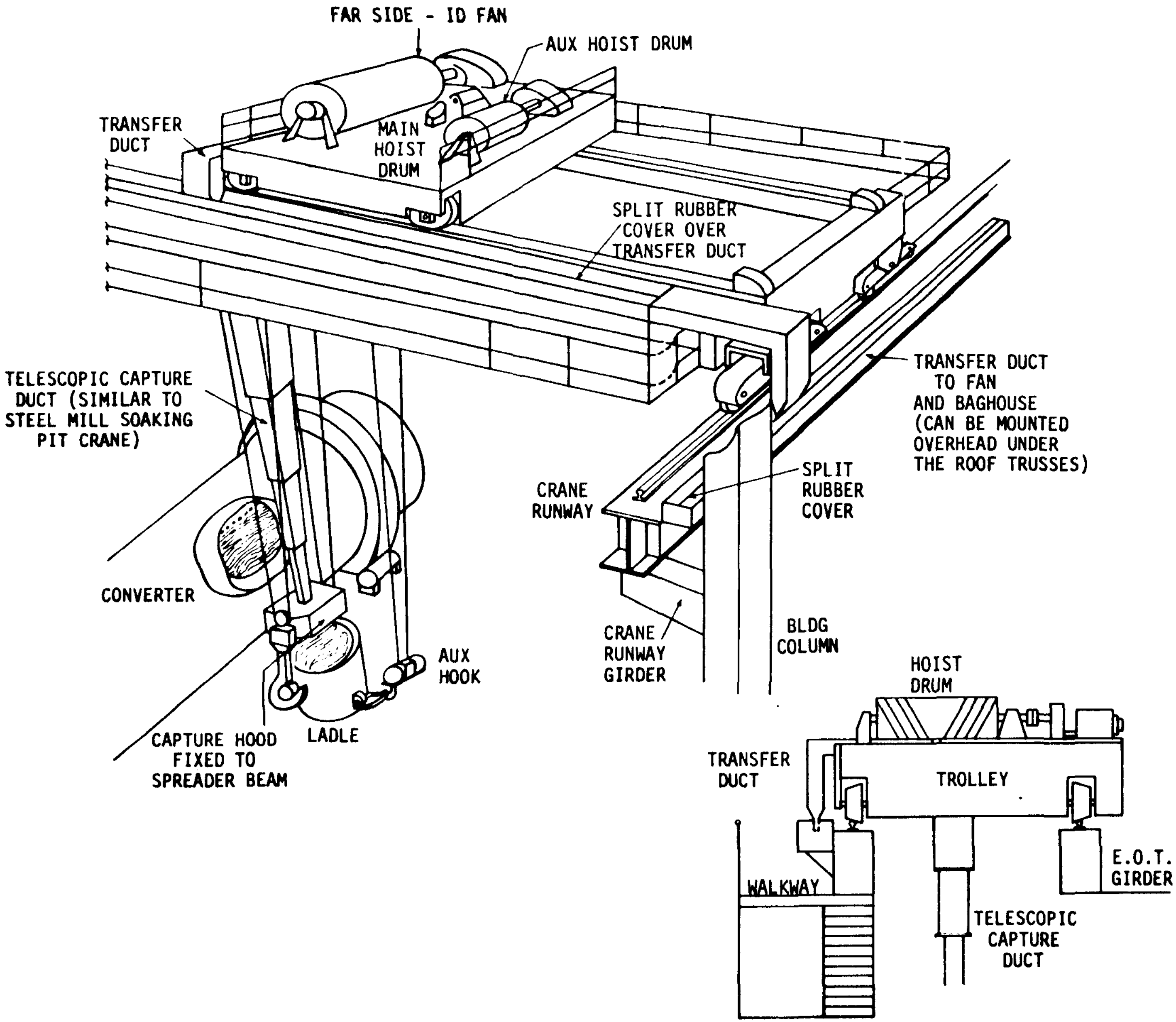


Figure 41. EOT crane with telescopic stiff leg.

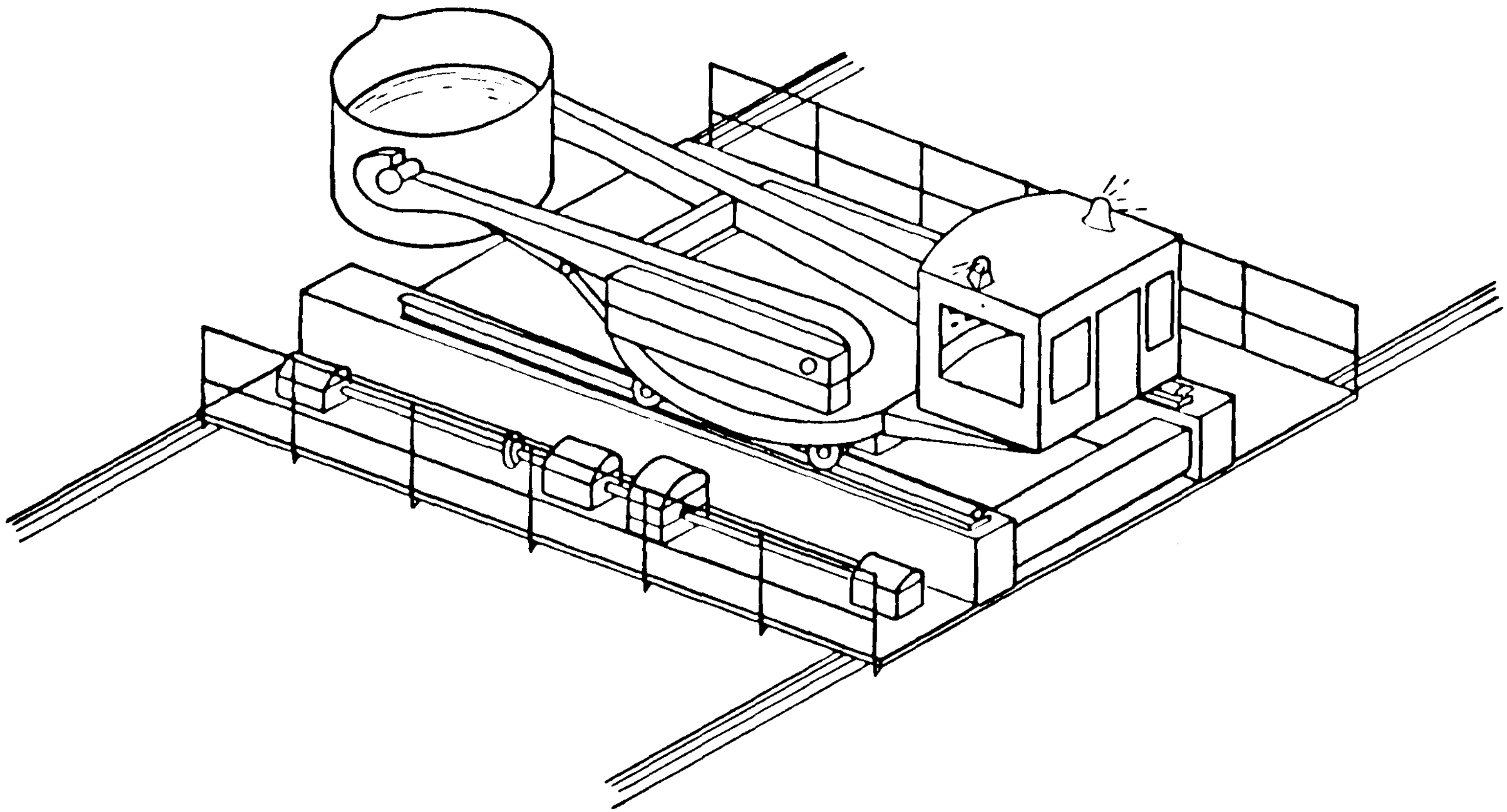


Figure 42. Modified charging machine.

In conjunction with covers, a specially designed bottom-pour ladle with stopper rods (similar to that in Figure 43) might be used at some smelters for transport of molten metal. Such a ladle, with an autopour unit attached to the EOT crane (Figure 43), is used in the steel industry, and the autopour unit controls the operation of the stopper rod mechanism when the steel is transferred into a mold or tundish, etc. At a copper smelter the EOT crane operator could control the flow of matte directly into a converter furnace or the flow of blister copper into an anode furnace. With a cover over the ladle or the use of an overhead crane equipped with an evacuation system, fugitive emissions would be greatly reduced. A bottom-pour ladle could be seated on a ladle support, and the ladle tapped into a specifically designed opening in the end of a converter through a refractory-lined airtight joint/cover. In such a system, the mouth of the converter could be made smaller; it would be used solely for pouring and not for receiving.

INDIVIDUAL FURNACE ENCLOSURES

In another fugitive emission control scheme, each furnace (e.g. a Peirce-Smith converter) would be operated in its own enclosure (Figure 44), with separate exhaust systems that could tie into a central point. For each furnace an EOT crane would be remotely controlled by an operator outside of the enclosure; during maintenance the operator would enter the enclosure at the platform levels and would use the captive EOT crane for maintenance around the furnace. Matte, slag, and blister copper ladles would be transferred to and from the enclosure via a four-wheel rail-mounted car, also remotely-controlled, that would pass through a sliding door.

Individual furnace enclosures would require the use of closed-circuit television. Gas evacuation systems would be provided to handle the fugitive emissions from each furnace. Such individual furnace enclosures offer a major advantage over total building enclosure in that emissions do not spread throughout the building and can be better controlled. Also, with separate furnace enclosures all of the building air is not evacuated, but only that needed to remove fugitive emissions to provide the required air changes in the smaller enclosure.

Maintenance requirements would be greater than in current systems, and failure of a captive EOT crane would require rapid remedial maintenance. Another problem would be supplying proper ventilation within the area during maintenance if the unit is on stream. It is envisioned, however, that the operator would usually work outside of the enclosure and would observe the area by means of closed-circuit television. Emissions would be minimal.

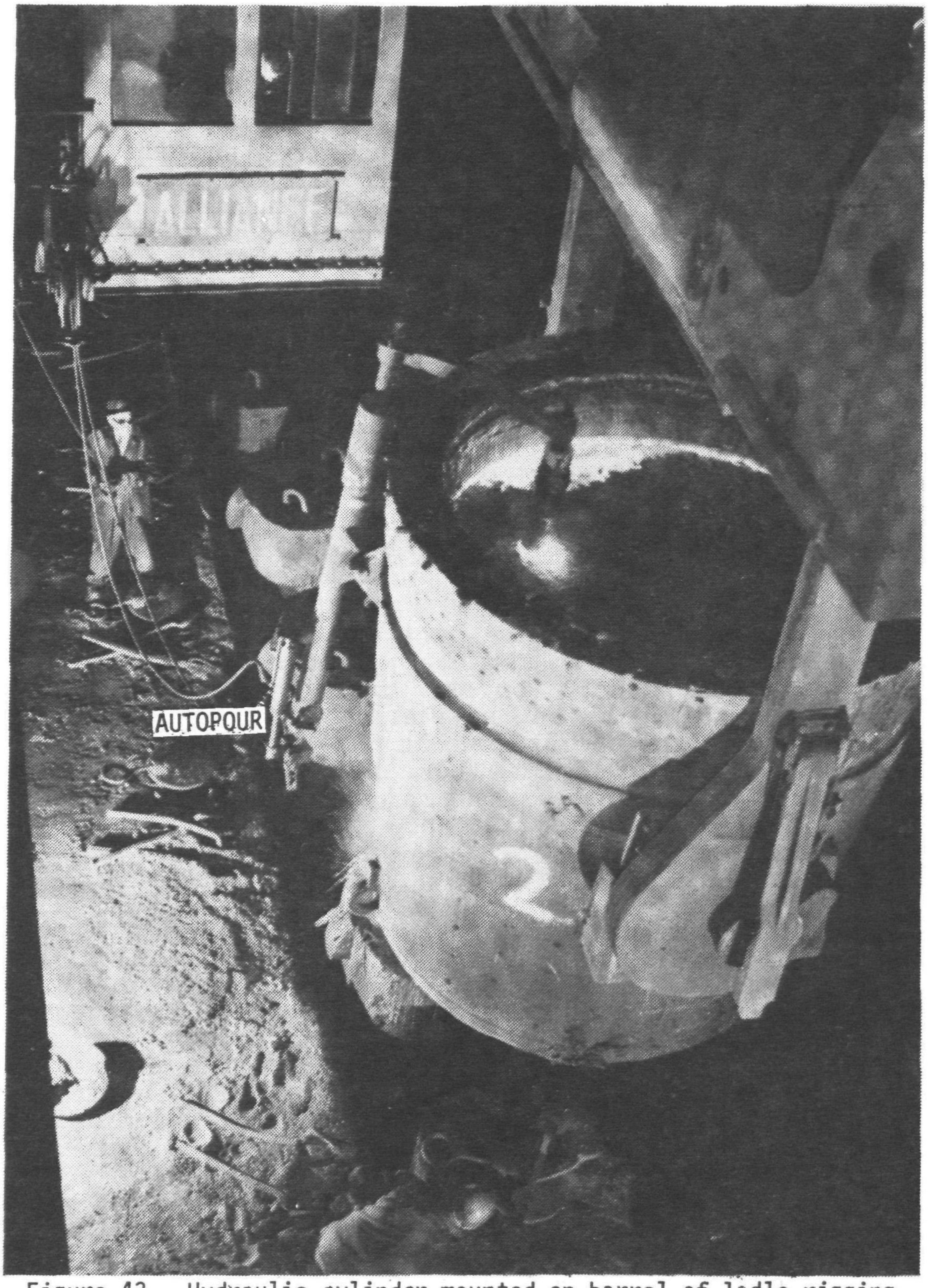


Figure 43. Hydraulic cylinder mounted on barrel of ladle rigging raises and lowers stopper rod to control flow of molten steel from ladle to ingot mold. (Courtesy, Blaw Knox Equipment, Inc.)

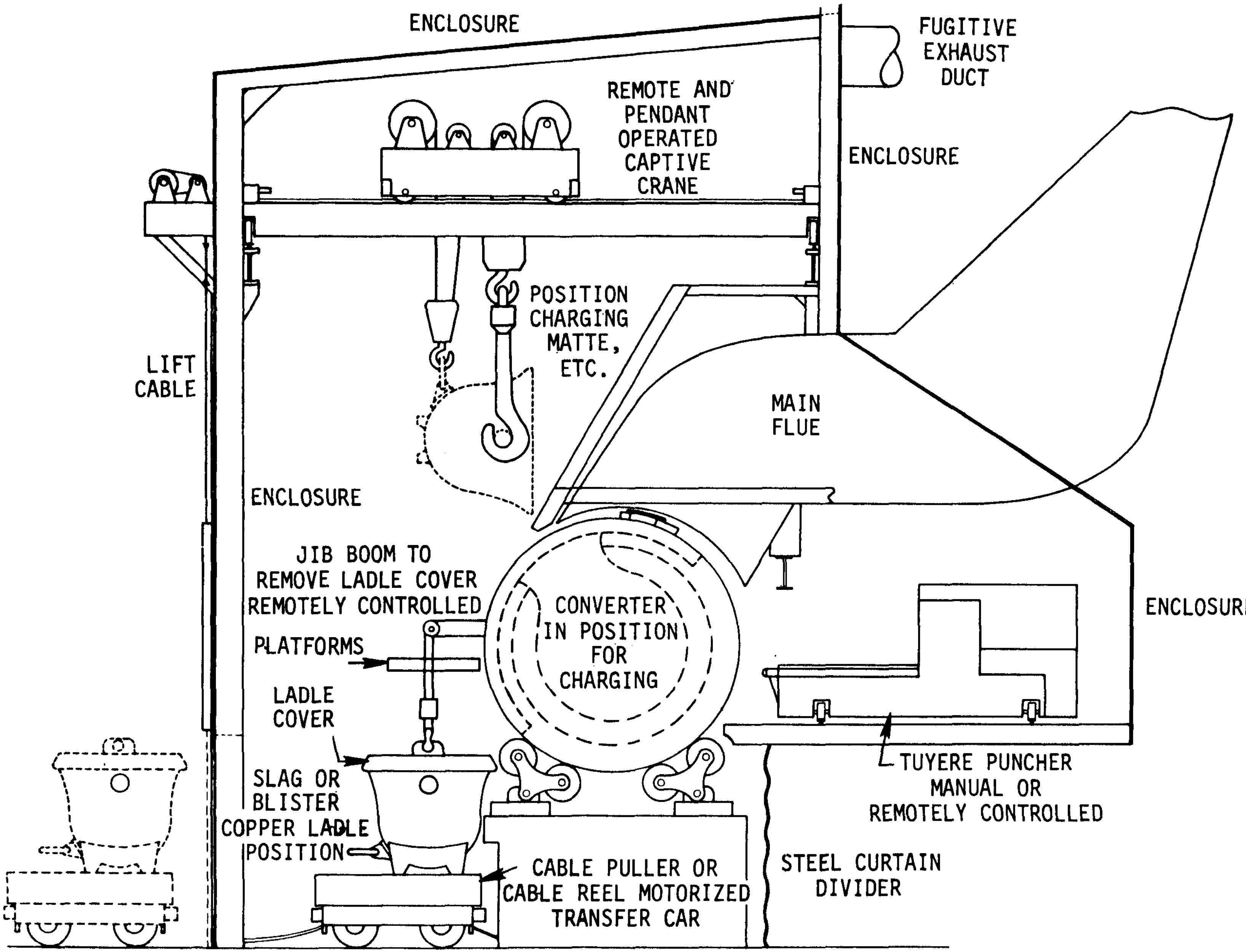


Figure 44. Individual furnace enclosure.

SECTION 8

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APPENDIX A

COST ANALYSIS

Detailed cost estimates are given for secondary hooding and air curtain controls. Brief summaries are given also for relative costs of other alternative control systems and estimated energy requirements. The measurements cited are in English units, as in the original cost studies.

COSTS OF SECONDARY HOODING SYSTEMS FOR CONVERTERS

This section is a capsule discussion of costs of the following major types of secondary hoods:

- Fixed type: made of structural steel with an elliptical cross-section. It is attached to the primary or uptake hood.
- Fixed and movable: consists of a movable intermediate hood and a hood fastened to the gate. Both hoods are made of structural steel with elliptical cross-sections so that they telescope in the retracted mode.
- Swing-away type with fixed overhead hood: made of structural steel, refractory lined, and supported by pivot arms with a motorized drive to permit positioning during blowing and pouring operations.
- Combination of fixed and movable swing-away type.

Cost Parameters

This section describes the various items that must be installed or modified to achieve control of fugitive emissions from the Peirce-Smith converter, source of a great quantity of secondary emissions. It does not include costs of certain operating procedures that would minimize fugitive emissions, e.g., maintaining minimal clearance between the primary uptake hood and the apron of the converter, or maintaining proper matte charges to provide for direct flow of gases from the mouth of the converter to the centerline of the primary uptake hood.

Following are descriptions of the items evaluated.

- The fixed hood has an elliptical cross-section of 17 feet 6 inches on the major axis and 7 feet on the minor axis; it is 9 feet 6 inches long. The plate is 3/8-inch carbon steel, with stiffeners of 7-inch channels. The fixed hood is bolted to the primary hood and to the smoke plenum of the secondary duct system.

- The movable hood in the retracted position is parked above the fixed hood. It has an independent track system with a five-speed, double-grooved hoist unit and slack cable limit switch. The movable head is 9 feet long and is elliptical, with a major axis of 18 feet 6 inches and minor axis of 7 feet 6 inches. These dimensions provide a 3-1/3-inch clearance between the movable and fixed hoods. There are mating plates on the lower end of the fixed hoods and on the top of the movable hood. The lower end of the movable hood is fitted with a 12-inch thick asbestos-type curtain that follows the elliptical perimeter to form a seal with the gate hood. The movable hood is constructed of 3/8-inch carbon steel with stiffeners of 7-inch channels.

- The gate hood is elliptical in cross-section with a major axis of 16 feet 6 inches and a minor axis of 6 feet 6 inches; it is 9 feet long. Clearance between the fixed hood and the gate is thus 3-1/2 inches. The hood would be bolted to the gate. The place is 3/8-inch carbon steel reinforced with 7-inch channels.

The dimensions listed above would be modified for each converter layout to provide the required clearances. Design considerations may dictate that the fixed hood is the largest unit, with the movable hood under it and the gate hood under the movable hood.

- If height of the crane runway rail presents a problem, the smoke plenum of the secondary hooding duct system could be fitted as follows: The plenum would span the primary uptake hood and would have a secondary hood dust bin affixed on each end. The dust bins would be equipped with pneumatic dust valves and discharge pipes. This evaluation includes no provision for removal of dust in the dust bins. The smoke plenum for this study is 4 feet by 4 feet 8 inches by 21 feet. It is constructed of 3/8-inch steel with 6-inch channel stiffeners.

- The secondary hooding duct system would have an uptake from each dust bin adjacent to the converter and then pass to its main ducting header for fugitive emissions. The damper valve shown would be adjacent to the main ducting header and would be closed when the converter is out of service. Existing facilities would determine the path of retrofit. The gases go to a dust bin ahead of the fans and from there to the breeching into the main converter gas duct and to the stack. For this study, the main duct runs are 600 feet long.

- The fans considered in this estimate are Buffalo Forge Type 1320 BL, single inlet, Arrangement 1, Class 3, with 145 bhp, 785 rpm, 80,000 ft^3/min at 200°F. There would be one fan for each converter in the plant; as many fans as are required would be tied into the system.

- Support items for this system include piping, wiring, foundations, supports for ducting every 20 feet, expansion joints, miscellaneous platforms, and walkways. Valves, fans, dust bins, and similar items are flanged for ease of maintenance.

The retrofit factor was considered as being midway between a new installation and an existing "difficult" installation.

Estimates of total installed costs are based on current (1978) costs for major components of specified sizes, as provided by equipment suppliers. Estimates of fabrication costs and installation in the southwest are based on general accepted engineering practice (e.g., as given in Richardson's, Mean's, the Chemical Engineering Index, and K. M. Guthrie) and on data from PEDCo engineering files.

A 5.0 percent contingency factor is applied to the total of the direct and indirect costs to allow for changes in equipment or design changes. An escalation factor of 7-1/2 percent per year is used to account for increases in cost of equipment, labor, and services before and during construction. Direct capital costs include equipment, instrumentation, piping, electrical, structural, foundations, site work, insulation, and painting. Indirect capital costs include engineering costs, contractor's fee and expenses, interest (accrued during construction on borrowed capital-estimated at 9 percent per year), freight, offsite expenditures, taxes (sales tax of 4 percent of equipment cost), startup or shakedown, and spares.

Annualized costs include both operating costs and fixed capital charges: utilities, labor and fringe benefits, maintenance, plant overhead and total fixed costs, which amount to

19.97 percent of total installed costs and consist of depreciation over 15 years at 6.67 percent unless otherwise indicated, property insurance at 0.3 percent, property taxes at 4 percent, and interest on borrowed capital at 9 percent.

Capital Costs

Table A-1 shows the direct costs, indirect costs including one year of contingency and escalation, and total capital costs for plants containing one to nine converters without a baghouse in the fugitive emission discharge system. Table A-2 includes the system of Table A-1 with addition of a baghouse and appropriate increase in the fan pressure design.

Operating Costs

Operating costs include operating labor at $8 per man-hour, supervision at 15 percent of labor, maintenance for labor and supplies at 2 percent of total capital costs, maintenance materials at 15 percent of maintenance labor and supplies, electricity at 35 mills per kWh, plant overhead at 50 percent of operations, and payroll at 20 percent of the operating labor costs. The fixed costs include a straight-line depreciation over 15 years, 0.3 percent for insurance, 4 percent for taxes, and 9 percent for capital costs. Table A-3 lists the operating costs for a multiconverter plant without a baghouse in the fugitive emission discharge system. Table A-4 includes the system of Table A-3 with addition of a baghouse and appropriate increases in energy and maintenance costs.

Additional handling of slag and blister ladles may cause delays in operation of the movable and swing-away converter hoods. It is estimated that a delay of 5 to 15 seconds may occur with each ladle movement, equivalent to a total delay of 3 to 10 minutes per day or a 0.23 to 0.7 percent slowdown in production. This loss is calculated on an annual basis as part of the operating cost since it is negligible in comparison with other delays that are encountered (e.g., delay because matte is unavailable, because the anode furnace cannot accept more blister copper, or because of maintenance of converters or furnaces.

Each of these systems is connected to a main discharge duct and an exhaust fan that exhausts to an existing stack.

COSTS OF AIR CURTAIN CONTROLS

Order-of-magnitude costs (± 35 percent) of an air curtain for control of fugitive emissions were developed in a manner somewhat similar to that for secondary hooding systems.

TABLE A-1. ESTIMATED CAPITAL COSTS OF SECONDARY HOODING AT MULTICONVERTER PLANT WITHOUT BAGHOUSE
(dollars)

No. of converters	Direct costs	Indirect costs	Total costs
1	760,000	532,000	1,292,000
2	1,216,000	785,000	2,001,000
3	1,532,000	963,000	2,495,000
4	1,771,000	1,255,000	3,026,000
5	2,211,000	1,463,000	3,674,000
6	3,219,000	2,122,000	5,341,000
7	3,601,000	2,383,000	5,984,000
8	3,880,000	2,545,000	6,425,000
9	4,402,000	2,850,000	7,252,000

TABLE A-2. ESTIMATED CAPITAL COSTS OF SECONDARY HOODING AT MULTICONVERTER PLANT WITH BAGHOUSE (dollars)

No. of converters	Direct costs	Indirect costs	Total costs
1	1,122,000	736,000	1,858,000
2	1,616,000	1,007,000	2,623,000
3	2,127,000	1,298,000	3,425,000
4	2,587,000	1,714,000	4,301,000
5	3,103,000	1,966,000	5,069,000
6	4,850,000	2,936,000	7,786,000
7	5,523,000	3,466,000	8,989,000
8	6,093,000	3,792,000	9,885,000
9	6,877,000	4,244,000	11,121,000

TABLE A-3. ESTIMATED ANNUAL OPERATING COSTS OF SECONDARY HOODING AT A MULTICONVERTER PLANT WITHOUT BAGHOUSE (dollars)

No. of converters	Labor and supervision	Maintenance, labor, supplies, and materials	Overhead plant and payroll	Utilities	Fixed costs	Total annual costs
1	10,000	30,000	22,000	1,000	258,000	321,000
2	21,000	46,000	38,000	3,000	400,000	508,000
3	31,000	57,000	50,000	4,000	498,000	640,000
4	41,000	70,000	64,000	5,000	604,000	784,000
5	51,000	85,000	78,000	6,000	734,000	954,000
6	62,000	124,000	105,000	8,000	1,077,000	1,376,000
7	72,000	138,000	119,000	9,000	1,195,000	1,533,000
8	82,000	148,000	131,000	10,000	1,283,000	1,654,000
9	93,000	167,000	149,000	11,000	1,448,000	1,868,000

TABLE A-4. ESTIMATED ANNUAL OPERATING COSTS OF SECONDARY HOODING AT A MULTICONVERTER PLANT WITH BAGHOUSE
(dollars)

No. of converters	Labor and supervision	Maintenance, labor, supplies, and materials	Overhead plant and payroll	Utilities	Fixed costs	Total annual costs
1	17,000	43,000	33,000	80,000	371,000	544,000
2	28,000	60,000	50,000	122,000	524,000	784,000
3	38,000	79,000	66,000	226,000	684,000	1,093,000
4	48,000	99,000	83,000	343,000	859,000	1,432,000
5	58,000	117,000	99,000	404,000	1,012,000	1,690,000
6	72,000	179,000	140,000	523,000	1,555,000	2,469,000
7	82,000	207,000	161,000	878,000	1,795,000	3,123,000
8	92,000	227,000	178,000	1,039,000	1,974,000	3,510,000
9	103,000	256,000	252,000	1,209,000	2,221,000	4,041,000

The design evaluated here differs from the Mitsubishi air curtain system in that the side panels are extended to contain the area of the ladle awaiting blister copper or slag from the converter. Also, partial covering is placed on the approach side (Figure 37).

The air flow across the top is set at 1000 Nm^3/min, but the collection side is designed with its own suction fan to handle up to 2500 Nm^3/min so as to create a flow pattern that will capture the fugitives during charging, teeming, slagging, and similar operations.

Tables A-5 and A-6 show the total installed capital and annual operating costs.

RELATIVE COSTS AND ENERGY REQUIREMENTS

Table A-7 summarizes the costs of alternative control systems relative to the systems now in widespread use. Table A-8 lists the estimated energy requirements.

TABLE A-5. ESTIMATED CAPITAL INSTALLED COSTS OF AIR CURTAIN TYPE HOODING WITH BAGHOUSES AT MULTICONVERTER PLANT (dollars)

No. of converters	Direct costs	Indirect costs	Total costs[a]
1	$ 372,200	$ 586,700	$1,082,000
2	736,900	926,700	1,878,000
3	987,800	1,222,200	2,495,000
4	1,365,500	1,708,800	3,470,000
5	1,621,200	2,010,200	4,099,000
6	2,111,300	2,725,500	5,460,000
7	2,688,700	3,406,300	6,880,000
8	3,242,000	4,189,800	8,389,000
9	3,591,200	4,630,300	9,280,000

[a] Includes 5% contingency and 7-1/2% escalation for 1 year.

TABLE A-6. ESTIMATED ANNUAL OPERATING COSTS OF AIR CURTAIN TYPE HOODING WITH BAGHOUSES AT MULTICONVERTER PLANT
(dollars)

No. of converters	Labor and supervision	Maintenance, labor, supplies, and materials	Overhead plant and payroll	Utilities	Fixed costs	Total annual costs
1	17,000	25,000	20,000	59,000	220,000	341,000
2	28,000	43,000	29,000	67,000	382,000	549,000
3	38,000	57,000	36,000	80,000	506,000	717,000
4	48,000	80,000	47,000	145,000	704,000	1,024,000
5	58,000	94,000	54,000	168,000	832,000	1,206,000
6	72,000	126,000	76,000	225,000	1,109,000	1,608,000
7	82,000	158,000	92,000	240,000	1,398,000	1,970,000
8	92,000	193,000	110,000	297,000	1,704,000	2,396,000
9	103,000	213,000	120,000	349,000	1,886,000	2,671,000

TABLE A-7. RELATIVE COST EVALUATION OF ALTERNATIVE AND EXISTING SYSTEMS

Alternative system	Compared with							BLDS costs[b]			Problem areas	Capital installed costs[b]			Annualized operating costs[b]			Emissions[b]		
	Roaster	Dryer	Reverb	EAF[a]	Flash shelters	P-S converter	Anode furnace	Higher	Same	Less		Higher	Same	Less	Higher	Same	Less	Minimal	Less	Same
Cascade	X[c]	X	X	X	X	X	X	X			Maintenance Equipment location Proximity of furnaces Operating techniques	X			X			X		
Staggered with induction pumping	X	X	X	X	X	X	X	X			Maintenance Equipment location Proximity of furnaces Operating techniques	X			X			X		
4-BOF			X	X	X	X				X	Maintenance Enclosure design Operating techniques			X		X			X	
Crane evacuation			X	X	X	X	X	X			Maintenance	X			X				X	
Covered bottom-pour laoles			X	X	X	X	X				Maintenance	X			X				X	
Floor operated charger			X	X	X	X	X	X			Maintenance Operating techniques	X			X			X		
Flash smelter			X																	
Hoboken converter						X			X		Maintenance/operating techniques			X		X			X	
Individual furnace enclosure	X	X	X	X	X	X	X	X			Maintenance/operating techniques Air movement or changes	X			X			X		
Building evacuation	X	X	X	X	X	X	X	X			Maintenance Air movement or changes	X			X				X	

[a] Electric arc furnace.

[b] X indicates relative cost of alternative system; annualized operating costs include fixed operating and maintenance.

[c] X indicates conventional equipment with which alternative is compared.

TABLE A-8. ESTIMATED ENERGY REQUIREMENTS FOR CONTROL OF FUGITIVE (GASEOUS AND PARTICULATE) EMISSIONS AT INTAKES AND DISCHARGE POINTS

Process equipment	Energy requirement,[a] kWh/h		
	Secondary hoods; capture hoods minimum[b]/maximum[c]	Individual enclosure[c,d]	Building evacuation[c,d]
Roaster	192/200	27	271
Dryer	156/162	38	38
Outokumpu furnace	30/149	90	411
Noranda furnace	60/149	50	172
Electric arc furnace	89/176	110	444
Reverberatory furnace	53/149	90	355
Peirce-Smith converter	168/216	11	365
Hoboken converter	17/108	17	305
Anode furnace	55/108	11	365
EOT crane	7/73		

[a] Energy expended per unit; gas stream discharged after passing through a baghouse.

[b] Operating only when emissions are being captured.

[c] Operating continuously.

[d] Air changes estimated at 12 per hour.

APPENDIX B

TRIP REPORT: MITSUBISHI METAL CORPORATION, ONAHAMA, JAPAN

27 July 1979 DCN 79-201-010-05-X

TRIP REPORT

To: Alfred B. Craig, Jr.
Project Officer, IERL-Ci

From: Richard T. Coleman, Jr.
Project Director, Radian Corporation

Subject: Site visit to the Mitsubishi Metal Corporation primary copper smelter in Onahama, Japan

Trip date: 9 and 10 July, 1979

Contacts: Izumi Sukekawa, Group Manager, Technical Sales Dept.
Shun-Ichi Ajima, Manager, Technical Sales Dept.
Hiroshi Kono, Manager of Operations, Onahama Smelting & Refining, Ltd.
Yoshiyuku Tsuji, Metallurgist, Asst. Superintendent of R&D, Onahama S&R

Purpose: Investigate copper converter fugitive emission controls

Summary:

The combination of secondary hooding, air curtains, and building enclosure and evacuation used to control fugitive emissions at the Onahama smelter is approximately 80 to 90 percent effective based on visual observation. Additional building evacuation would certainly enable a 90 percent capture level to be maintained. The present system does not impede converter operations. Visibility from the crane cab is not seriously impaired by gases and particulate. However, additional building ventilation could improve visibility and reduce potential exposure in the crane cab.

Mitsubishi personnel did not seem to favor any detailed study of their facility by EPA. Any study of a system similar to Mitsubishi's would have to be a demonstration conducted at a U.S. primary copper smelter. It is recommended that this system be investigated in detail by both EPA and NIOSH as a potential solution to the fugitive emissions problem and as an engineering control for SO_2, particulate, and volatile metals (Pb, As, etc.) in primary copper smelters.

General:

The initial meeting with Messrs. Sukekawa and Ajima on July 9 took place in Mitsubishi Metal's Tokyo office. We discussed the purpose of our trip, emphasizing our interest in fugitive emission controls. We explained IERL-Ci's involvement in non-ferrous metals research during the past four years and indicated that demonstration of effective controls is a key R&D need.

Their reaction during that meeting and the subsequent trip to Onahama on July 10 was that they have demonstrated that their control technology works effectively and it is commercially available. They did not indicate any willingness or interest in participating in an EPA-funded testing or demonstration project.

Messrs. Sukekawa and Ajima were very cooperative and they and Messrs. Kono and Tsuji at the Onahama smelter provided a lot of information concerning both the converter controls and the MI (Mitsubishi) smelting process. This information is discussed below.

Converter Fugitive Emission Controls:

Emissions from the five converters in the Onahama smelter are controlled in four ways. These are:

- Primary hoods,
- Secondary hoods,
- Air curtains, and
- Converter aisle building exhaust ventilation.

This combination of controls was observed to work effectively and captured an estimated 80 to 90 percent of the fugitive gases leaving the converter. Only the building exhaust ventilation appeared to be slightly underdesigned. An increase in the gas volume evacuated from the building would probably assure a 90 percent control efficiency.

The converter control system is pictured schematically in Figure B-1. During normal blowing and standby, approximately 950 Nm^3/min of process gas are drawn through each primary hood. The primary hoods are not water cooled but still fit tightly on each converter without severe buckling. A gap of only two to three inches is maintained between the primary hood and the converter furnace. This enabled almost complete capture of SO_2 to be maintained during the blowing cycle. Fugitive emissions during blowing were observed to be minimal.

During charging and pouring, the primary hood draft is reduced by approximately 65 to 80 percent. The gas concentration being fed to the acid plant is used as a control on the primary hood damper. The draft on the primary hood is sufficient to allow perhaps 30 to 50 percent control of fugitive emissions during the period when the converter was rolled out. This is possible only because no air is introduced into the tuyeres during these periods. Control of the remainder of the fugitives during charging and pouring is provided by the secondary hoods, air curtains, and building exhaust ventilation.

The secondary hoods are sheet metal panels on either side of the converter which rise approximately 14 meters (50 feet) from the floor of the converter aisle. At the top of these panels, three fans are fitted into the hood. The fans create an air curtain across the slot in the top of the secondary hood. The slot is large enough to allow the crane cables to be maneuvered during both charging and pouring. On the other side of the slot is a duct which collects the exhaust from the three air curtain fans. The exhaust duct for the secondary hood is at the top of the sheet metal panels on the side farthest from the converter aisle. This arrangement is depicted in Figure B-2.

The secondary hoods operate at all times. During normal blowing and standby, when the primary hood covers the converter mouth, only 800 to 1200 Nm^3/min are exhausted from the secondary hood. When charging or pouring is in process, the gas volume exhausted is increased to between 2200 and 2500 Nm^3/min. These volumes are sufficient to establish a good air flow pattern into the hood exhaust duct and air curtain. Some gases, however, do spill out of the front of the hood and rise into the converter aisle. This occurs mainly during charging and pouring. The secondary hood was observed to capture an estimated 30 to 50 percent of those gases not collected by the primary hood during charging and pouring, and nearly all fugitives during normal blowing and standby.

The design basis for the secondary hood gas treatment is 2000 ppm SO_2 concentrations range between 400 and 500 ppm SO_2. These gases are mixed with the reverberatory furnace gases and are scrubbed in an MgO absorption process (see attached report).

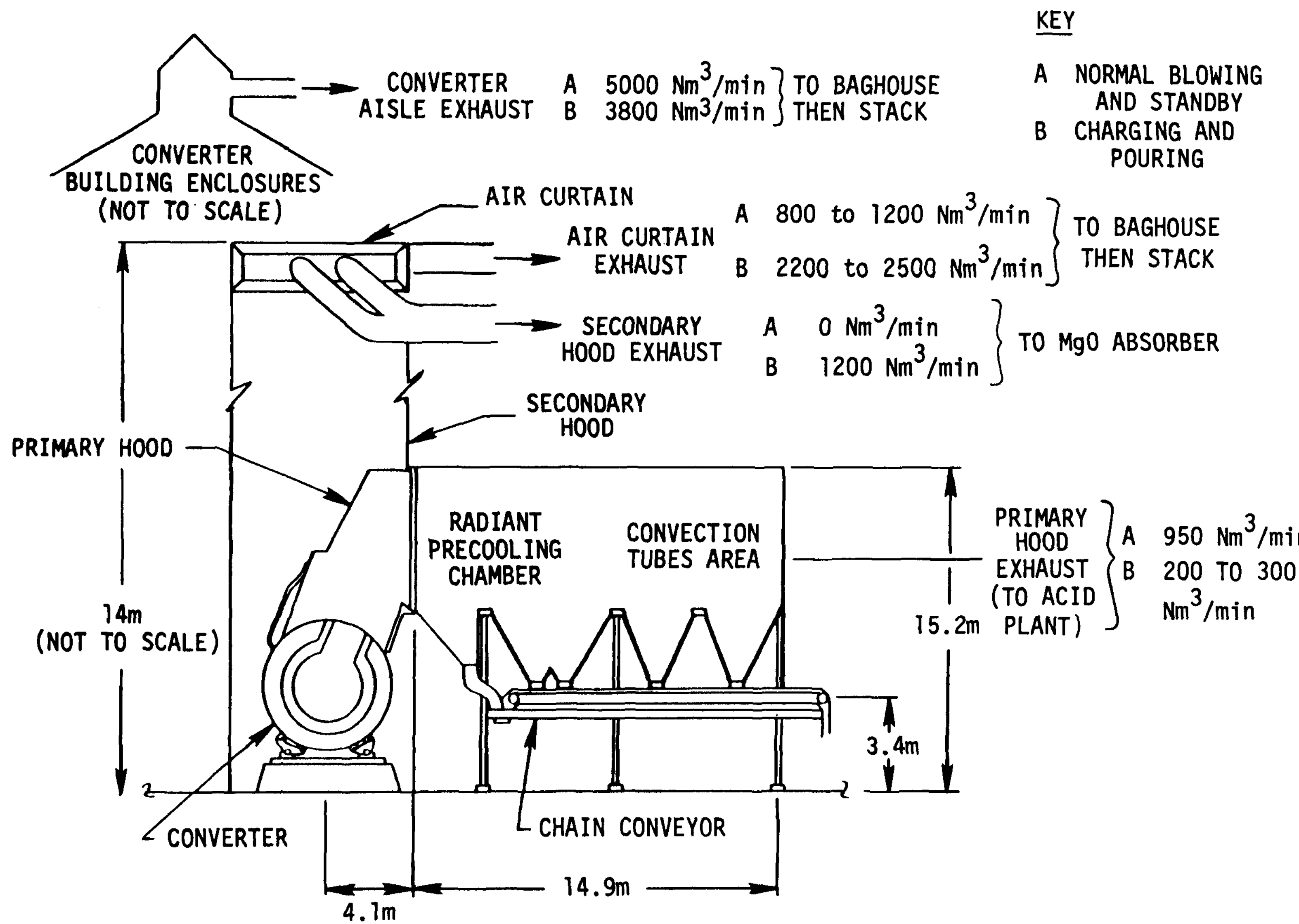

Figure B-1. Schematic of the Onahama converter emission control system.

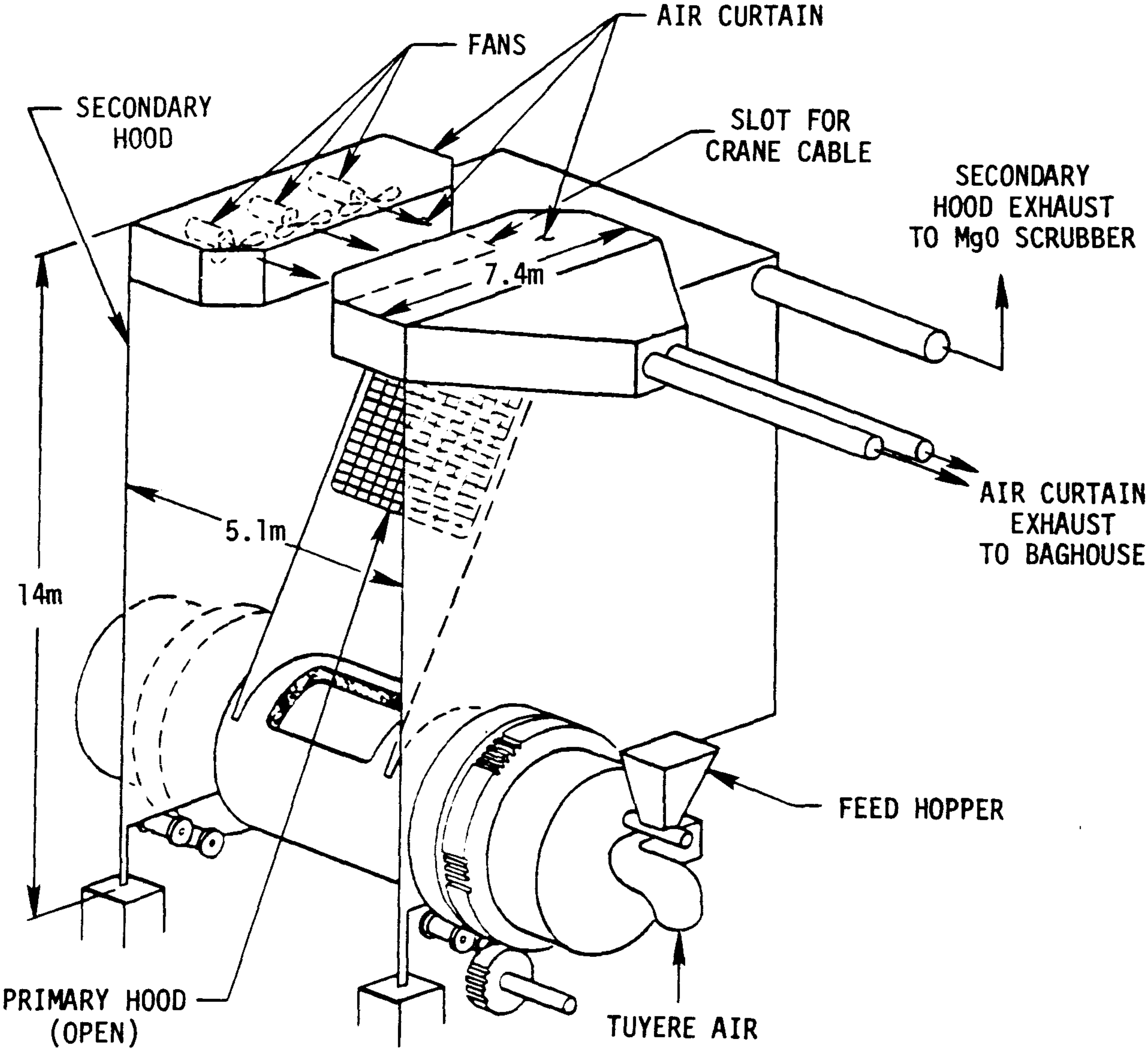

Figure B-2. Converter hooding arrangement.

The air curtain across the slot in the secondary hood captures those hot gases which would normally pass through the secondary hood slot. Three fans create the stream of air which passes above the slot in the secondary hood and enters the capture duct on the opposite side of the slot. This "push/pull" technique is almost 100 percent effective in collecting fugitive which would normally escape through the slot. Only those gases which spill out of the front of the hood remain uncaptured.

The gases collected by the air curtain range between 100 and 200 ppm SO_2. They are ducted to a baghouse for particulate removal and then are discharged through a stack.

The converter aisle building exhaust completes the converter fugitive emission control systems. The converter aisle has been almost completely enclosed with sheet metal partitions. The major effort involved separating the reverberatory furnace building from the converter aisle. Only the major access doors at either end and middle of the converter aisle remain open. The reverberatory furnace building is relatively free of SO_2 as a result of isolating it from the converter aisle. However, a significant quantity of SO_2 remained uncollected at the top of the converter aisle.

During normal blowing and standby, 5000 Nm^3/min are exhausted from the building. This volume of air is ducted to the same baghouse which handles the air curtain exhaust. However, each time an air curtain is activated, the building exhaust is reduced by 1200 Nm^3/min, the quantity used by the air curtain. This system is not quite adequate for complete building ventilation. A significant increase in the converter aisle exhaust would probably be needed to clear the residual SO_2 which collects at the top of the building. The residual SO_2 is not a major problem and does not impair visibility from the converter aisle crane cabs.

With three converters operating, one on standby and one under repair, approximately 10,500 Nm^3/min are exhausted from the converter building. This results in a building air change once every 5½ to 6 minutes. This is sufficient to maintain the converter aisle work areas relatively free from any significant SO_2 concentrations.

Mitsubishi (MI) Process:

Mitsubishi personnel provided information on their continuous smelting process. The information provided confirmed that reported in "Emerging Technology in the Primary Copper Industry." The MI process is located at the Naoshima smelter. We were not able to arrange a visit to Naoshima because a new plant management was taking over responsibility for the smelter the same week we made our visit.

The MI process is a three furnace smelting system. A 65 percent copper matte is produced in the smelting furnace. Coal is mixed with the concentrate to provide more uniform heat for smelting, reduce the flame temperature required in the furnace, and increase the slag fluidity. Supplementary oil fired burners are also needed to heat the furnace. These are two changes from the original intent of operating an autogeneous smelting furnace. They have been made necessary because water cooling of the furnace refractory was added to reduce refractory wear. This loss of heat made supplemental fuel necessary.

Slag and matte from the smelting furnace flow by gravity in a launder to the electric slag cleaning furnace. The copper content of the slag is maintained at 0.5 to 0.6 percent Cu by maintaining a reducing atmosphere in the slag cleaning furnace. The slag then flows by gravity to the converter furnace.

The converter furnace produces anode copper which flows to a holding furnace prior to anode casting. Slag from the converter furnace is granulated separate from the slag cleaning furnace slag. The converter furnace slag is then recycled to the smelting furnace.

Mitsubishi has licensed their process to Gulf Western for a 65,000 metric ton per year smelter in Timmins, Ontario. This is a 30 percent increase in capacity over the 50,000 mt/yr smelter at Naoshima. Both smelters will be similar in that the concentrates processed contain very little arsenic (<100 ppm). The Timmins, Ontario smelter, however, will process concentrate containing up to 6 percent zinc and 5 percent lead. Consequently, the particulate removal equipment will be much larger than at the Naoshima smelter.

Mitsubishi is presently marketing the MI process. The major advantages cited are: energy efficiency, low atmospheric emissions, and no converter aisle. Eliminating the converter aisle lowers the required smelter capital investment and eliminates the fugitive emissions from crane operations.

TECHNICAL REPORT DATA

(Please read Instructions on the reverse before completing)

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15. SUPPLEMENTARY NOTES

16. ABSTRACT

This report deals with fugitive emissions from copper smelting and with related emission control measures. The study involved evaluation of the controls now used in the copper smelting industry and development of suggestions for alternative control devices and practices.

A brief overview of copper smelting processes is followed by a more detailed analysis of the conventional processes identifying portions of the operating cycle that produce fugitive emissions. Emphasis is placed on Pierce-Smith Converting, which is one of the major emission sources in copper smelting. Some alternate processes now in limited used in the U. S. are described including estimations of fugitive emissions from these conventional and alternative copper smelting processes.

A specific report on the utilization of the Hoboken Converter is being prepared at the time of this report. The USEPA should be contacted if a copy of this report is desired.

17. KEY WORDS AND DOCUMENT ANALYSIS

a. DESCRIPTORS	b. IDENTIFIERS/OPEN ENDED TERMS	c. COSATI Field/Group
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