CHAPTER 11.6

Conversion and Refining

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This chapter introduces the pyrometallurgical processes of conversion (also called "converting") and refining, and discusses how they are broadly applied, with a focus on current primary metallurgical practice. To convey the relative importance and specific goals of conversion and refining, these processes are placed in the context of the periodic table of elements and with smelting processes covered in Chapter 11.5, "Smelting." The main sections covering unit operations include a brief description of key developments up to the present day and typical modes of operation of furnace technologies used at modern smelting plants for the major metals.

The chapter is organized based on some specific definitions. *Conversion* is defined as the pyrometallurgical removal of elements from a melt originally produced by a smelting process to allow subsequent refining of the crude metal so produced. *Refining* is defined as the removal of generally low levels of elements deemed deleterious to the refined metal properties or subsequent steps required to refine the metal to commodity or market specifications.

METALS REQUIRING CONVERSION AND REFINING

Conversion processes cover the nonferrous practice of removing iron and sulfur from copper-nickel mattes produced in any of the relevant smelting processes described in Chapter 11.5. In practice, conversion is based on reacting oxygen or oxygen-enriched air, either with comminuted solid matte using suspension or flash technology or introduced to a molten matte bath using submerged tuyere, top submerged lance, or top lance furnace technologies to produce crude metal/alloy or enriched sulfide. Conversion generally lowers the levels of major deleterious elements, but final control of such elements must usually be dealt with in subsequent refining steps (Figure 1).

Refining processes generally implemented at primary smelting facilities involve the removal of deleterious elements from crude metal/alloy resulting from the conversion of matte or are produced directly from concentrate. Elements are considered deleterious if they affect final metal quality marketability or if they negatively impact further metal refining steps, such as in copper electrorefining to cathode, or final metal/alloy quality. The key processes involved in refining are oxidation (slagging and gasifying of oxidized impurities), liquation (solidification of higher melting point impurities), drossing, vacuum (vaporizing impurities), and stirring/injection (slagging of impurities and inclusions).

HISTORICAL BACKGROUND

This section provides a short historical sketch on conversion and refining, with particular reference to copper and nickel through the late 1800s. Up until the introduction of the bottom-blowing Bessemer converter in 1856 (Weeks 1896; Bessemer 1905; Wilder 1949), conversion and refining operations in iron and steel and nonferrous metallurgy were carried out in relatively small furnaces, such as the reverberatory-type furnaces (Bloxman 1871), along with manual manipulation of the melt by metal bars and/or paddles. These early operations evolved over centuries from small bowl-shaped furnaces, some originally operated with hand-operated bellows.

Following the success of the Bessemer process for steelmaking, similar approaches were attempted for copper and nickel sulfide conversion. Among other challenges, the high thermal conductivity of copper caused freezing on the vessel bottom. Working at the Éguilles copper plant in France in 1880 with a small Bessemer converter, metallurgists Pierre Manhès and Paul David modified the placement of air-injection tuyeres from the bottom of the vessel to the side and above the copper melt, making conversion of the melt possible without freezing (Pelletier et al. 2009). They subsequently developed a horizontal barrel converter. Toward the end of the 1800s, both horizontal and upright configurations were being used with acidic (silica-based) refractory linings, which also served as the source of flux (Figure 2). The potential benefits of using basic refractory linings were long appreciated, but commercial success was proving elusive (Mathewson 1913). The breakthrough occurred with development of the Peirce–Smith converter in 1909, still in use in modernized form today (Southwick 2009). Early nickel conversion furnaces were similar to those utilized for copper and were deployed in

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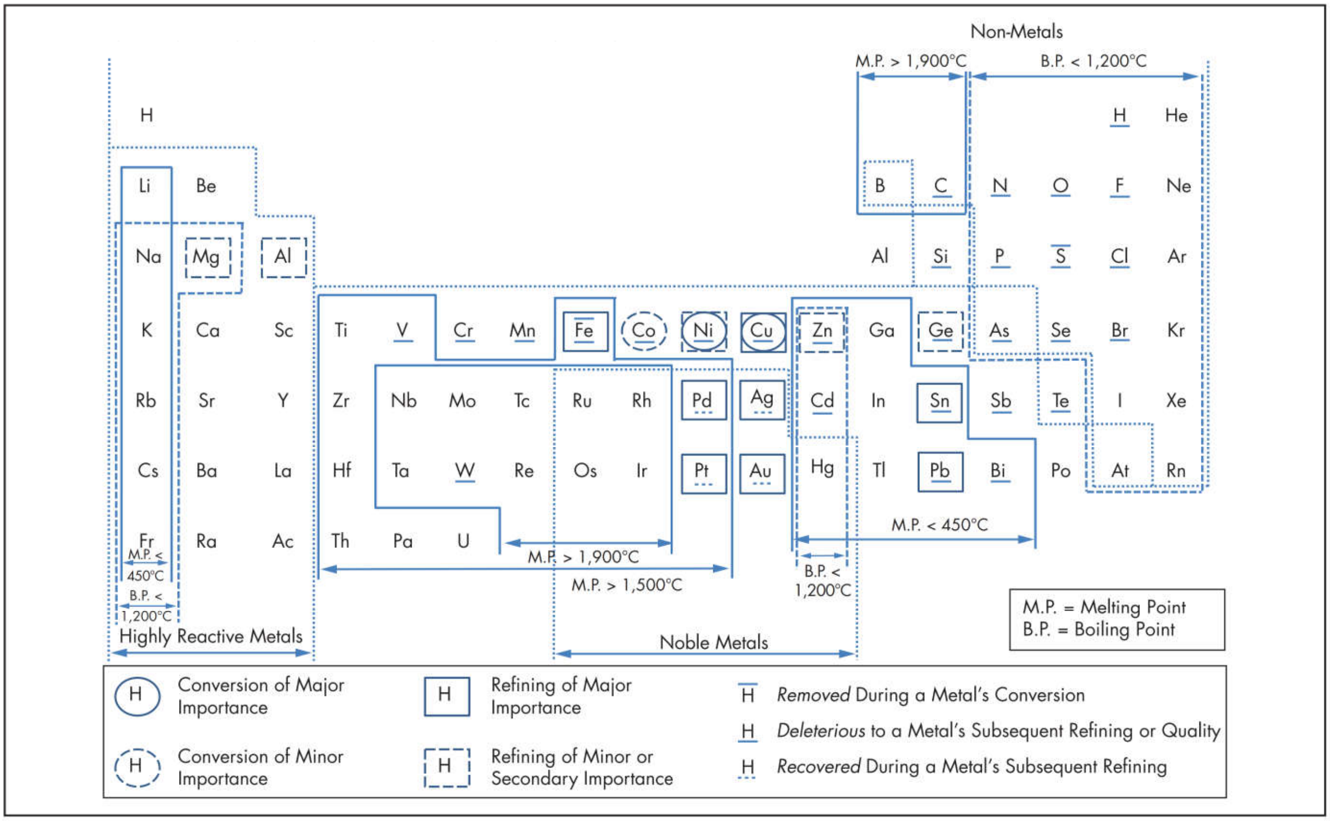


Adapted from Dannatt and Ellingham 1948

Figure 1 Abridged periodic table of the elements highlighting those metals subjected to conversion and refining and common deleterious element associations

Norway and Canada in the late 1800s (Royal Ontario Nickel Commission 1917).

Lead metallurgical evidence can be found as far back as the seventh millennium BC. Early pit/bowl furnaces were used to smelt oxidic and sulfidic lead ores with charcoal. Given that the crude lead nearly always contained a significant concentration of silver (0.01% to 0.5%), "cupellation" was gradually carried out to separate lead and other impurities by oxidation, to produce silver and a litharge (lead oxide) slag. By 1850 the "Parkes process" was developed in the United Kingdom, which uses zinc additions to remove the silver in a more concentrated form. This process remains in use today.

THEORETICAL ASPECTS OF MODERN CONVERSION AND REFINING

Conversion Theory

This section provides a short overview of the common theoretical concepts of conversion for copper and nickel.

Copper Conversion

For suspension or flash conversion of copper matte (George 2002), feed is ground to a size sufficient to ensure early ignition and combustion, typically P_{80} <75 µm. Sufficient oxygen is introduced to ensure near complete overall sulfur elimination after reactions in the bath below the flash shaft. The main reactions are presented here, together with the corresponding heat of reaction:

$$Cu_2S(s) + 2O_2(g) \rightarrow 2CuO(l\ solution) + SO_2(g) \quad -2.6\ MJ/kg\ Cu \qquad \textbf{(EQ 1)}$$

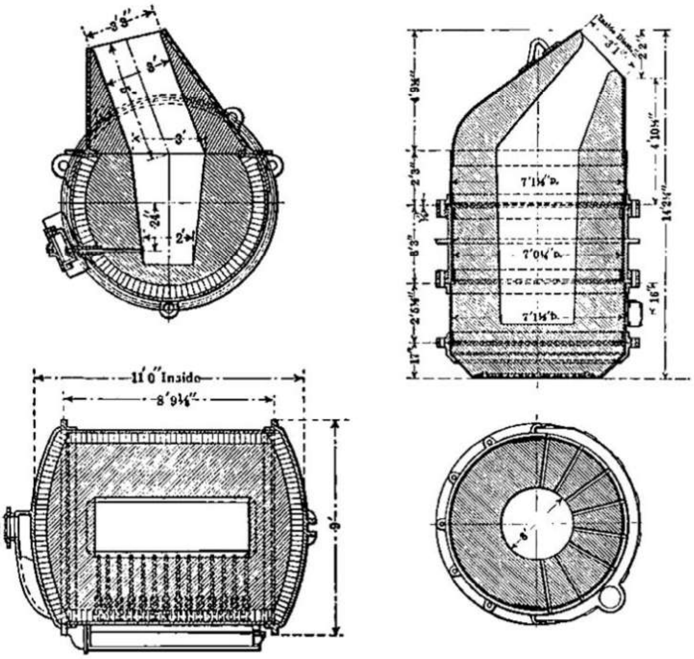
$$2FeS(s) + 7/2\ O_2(g) \rightarrow Fe_2O_3(s\ and\ l\ solution) + 2SO_2(g) \quad -0.8\ MJ/kg\ Cu\ (at\ Fe/Cu\ 0.1) \qquad \textbf{(EQ 2)}$$

$$Cu_2S(s) + O_2(g) \rightarrow 2Cu(l) + SO_2(g) \quad -0.4\ MJ/kg\ Cu \qquad \textbf{(EQ 3)}$$

$$FeS(s) + 3/2O_2(g) \rightarrow FeO(s\ and\ l\ solution) + SO_2(g) \quad -0.6\ MJ/kg\ Cu\ (at\ Fe/Cu\ 0.1) \qquad \textbf{(EQ 4)}$$

A further reaction of FeO with silica and/or lime flux also occurs to make a converter slag. In addition, several subreactions also occur, as discussed in more detail by Davenport et al. (2001).

Reactions 1–4 can also be used in general to describe batch or continuous top lance conversion and batch submerged tuyere conversion (known as the Peirce–Smith converter). In top lance conversion, the feed copper matte is generally molten and the oxygen is transferred to the slag/matte emulsion via a top-blowing lance (at ~1,230°C) with an efficiency of >90%. Lime/limestone as flux (or silica flux) is continuously injected to allow for formation of the conversion slag. Because of the agitated melt, the sulfur dioxide partial pressure is lower compared to suspension or flash conversion. Consequently, sulfur



Note: Units of measurement are in feet and inches.

Source: Hofman 1914

Figure 2 Historic horizontal and upright conversion furnaces with acidic refractory linings

in the blister produced from these operations tends to be lower than that in flash conversion for the same copper level in slag.

Nickel Conversion

Submerged tuyere conversion for nickel matte is very similar to the copper matte blow (at ~1,300°C) but typically with a higher initial Fe/(Ni + Cu + Co) ratio. Generally, the end point in nickel conversion is a low-iron nickel (plus copper) matte, which is subsequently treated hydrometallurgically to recover the contained metals.

Refractory and Containment

For all conversion processes, high-quality chrome-magnesia refractories are invariably utilized. Common to all applications is the wear associated with contacting the refractory with slag containing oxidic copper (calcia and silica based). For the continuous processes, substantial external cooling of the shell and refractory is required to control steady-state liner geometry.

Refining Theory

This section provides some brief comments on the theoretical concepts of refining for ferrous and nonferrous applications.

Iron Refining

Iron produced via the blast furnace must be refined for the production of steel. Liquid iron from the blast furnace typically contains carbon, silicon, manganese, phosphorus, and sulfur levels that must be reduced. The level of these elements will depend on blast furnace iron ore and coke quality (Turkdogen and Fruehan 2012).

Ferroalloy Refining

Ferronickel, as produced in electric furnace smelting of calcined laterite ore, is refined typically in 25-to-30-t (metric ton) batches using a ladle refining system adapted from that used in steel refining. Several refining agents are added to the ferronickel melt along with oxygen lancing to remove elements such as phosphorus, silicon, carbon, and sulfur that may be present. Phosphorous removal is achieved by adding lime with oxygen injection, producing a calcium phosphate slag; carbon and silicon are also removed by oxygen lancing. Sulfur is removed by the addition of calcium carbide, producing a calcium sulfide slag (Thurneyssen et al. 1960, Zamalloa et al. 2009).

Copper Refining

Crude or "blister" copper produced either directly from smelting or, more typically, via a conversion process must have the sulfur and oxygen levels reduced to permit the casting of flat anode shapes suitable for subsequent electrorefining. The oxidation blow for final sulfur elimination in this copper raises the melt oxygen to a level that must subsequently be reduced to allow casting of quality anode shapes. This "reduction step" is executed through the same tuyeres using either natural gas, ammonia, or, more efficiently, superheated steam/natural gas reduction.

Lead Refining

Lead produced directly in a blast furnace or other smelting furnace is subjected to many fire-refining processes to achieve required final metal quality. Typically, these are carried out sequentially in a series of heated ladle-type furnaces referred to as "kettles." In this way, elements such as copper (also nickel and cobalt), arsenic, tin, zinc, antimony, and bismuth are removed by chemical flux additions and skimming off the resulting dross or slag that is formed (Tan and Vix 2006; Dawson 1989; Ashman et al. 2000; Leroy et al. 1970; Blanderer 1984; Kapoulitsas et al. 2000).

Tin Refining

Crude tin produced in either reverberatory or submerged lance furnaces is cooled to approximately 400°C, with concurrent air agitation for refining. Elements such as copper, lead, and zinc are also removed by chemical flux addition and removal of the resulting dross (Smith 1996).

Gold, Silver, Platinum, and Palladium Refining

Precious metals such as gold, silver, platinum, and palladium tend to deport to copper and nickel mattes in pyrometallurgical smelting, conversion, and refining processes (Avarmaa et al. 2015). Consequently, many of the processes associated with copper and nickel primary production are also utilized for the recovery of these precious metals. In these processes, matte and alloy target compositions are often dictated by the refining processes used to recover the precious metals (Barrett and Knight 1989).

UNIT OPERATIONS FOR NONFERROUS CONVERSION

Significant progress was made in the early 20th century in improving conversion productivity through the successful adoption of larger vessels, high-quality basic refractories, improved process control, and better primary and secondary hooding. Although the batch Peirce–Smith converter dominated, alternative designs modeled after the submerged tuyere concept were also introduced. The Hoboken converter (Belgium) was developed to give improved off-gas capture with some measure of success (Johnson et al. 1979) but has not been widely adopted. The Noranda converter (Canada) and the Shuikoushan/bottom-blowing converter (SKS/BBC) and submerged lance conversion and refining (SLCR) converters (China), which are also based on the submerged tuyere concept, were recently developed for continuous conversion, thus offering an improved conversion approach compared to conventional batch conversion. Nevertheless, the classic Peirce–Smith converter remains the predominant submerged tuyere configuration, representing almost three-quarters of the current global reported copper and nickel conversion capacity.

The last half of the 20th century also saw the steady development and commercialization of top lance and suspension or flash conversion, such as the Mitsubishi Materials Corporation (MMC) and Outotec–Kennecott processes applied to copper (Kapusta 2004). More recently, batch and continuous top submerged lance conversion have been deployed, such as the IsaSmelt and Ausmelt processes on copper and high-platinum-group-metal (PGM) nickel mattes (Warner et al. 2007).

An inspection of Table 1 shows that in the last 25 years, 36% of the newly installed or modernized 11.0 Mt (million metric tons) Cu/yr global conversion capacity was based on alternatives to Peirce–Smith converters. The last decade saw a similar level of alternative converter technology adoption, amounting to 36% of the installed capacity of 6.8 Mt Cu/yr. The pace of new technology adoption in the last five years has increased to represent 54% of the 4.0 Mt Cu/yr of newly installed, modernized, or under-construction capacity.

Submerged Tuyere Conversion Furnace

The submerged tuyere conversion furnace refers to furnace technology whereby the process gas is injected into the melt via submerged tuyeres to effect the required metallurgical reaction. The vessel is fitted with a turning mechanism so it can be turned about 50° to bring the tuyeres out of the melt for operations such as charging matte or skimming (by rotating the vessel further).

Included in this category of furnaces are the copper and nickel-copper converters, commonly known as Peirce–Smith converters, the Noranda converter, the SKS/BBC (Yie 2014), and the SLCR (Cui et al. 2016) in China. The Peirce–Smith converters typically operate in batch mode, as will be described here; however, several versions on new submerged tuyere converter furnaces operating continuously have been recently developed. These include the Noranda converter developed in Canada and the SKS/BBC and SLCR developed in China. Today, there are more than 200 Peirce–Smith converters worldwide at more than 60 smelters (considering those smelters that produce over 50,000 t Cu/yr).

Peirce–Smith Converter Furnace

Since development of the Manhès converter, vessels were gradually increased in size (Wraith et al. 1994; Mackey 2014). A major breakthrough in converter design and operation came in 1909 with development in the United States of the Peirce–Smith horizontal converter, so named after the two developers. This vessel, which was a huge success and widely adopted, included magnesite brick lining, silica flux feeding, means for tuyere punching, and a robust turning mechanism.

This unit became the main converter "workhorse" of the copper industry from that time to the present day, and it is expected to operate well into the future. Up until the 1990s, all major copper smelters used a Peirce–Smith converter for copper conversion. These vessels are also used in nickel-copper conversion. The units increased in size to match the ever-increasing demand for copper through the 20th century (Wraith et al. 1994; Mackey 2014). By about the 1940s, the industry generally adopted a "standard" vessel size of 4.0 m diameter by 9.1 m—first introduced in about 1910—and vessels this size still operate today at some smaller plants.

Most large smelters now use bigger vessels—a large converter today is typically 4.5 m diameter by 13.8 m long, having an inside volume 100 times that of the first units developed in France in the 1880s. As an example, such large vessels were installed in 2000 at the Ronnskar smelter, Sweden; typical blowing rates are ~46,000 Nm^3/h. The modern converter today has similar overall features as the original barrel units, such as submerged air injection, flux addition, tuyere punching, and so forth, but has developed into a robust, high-efficiency unit with advanced process controls and very efficient fugitive gas capture (Figure 3).

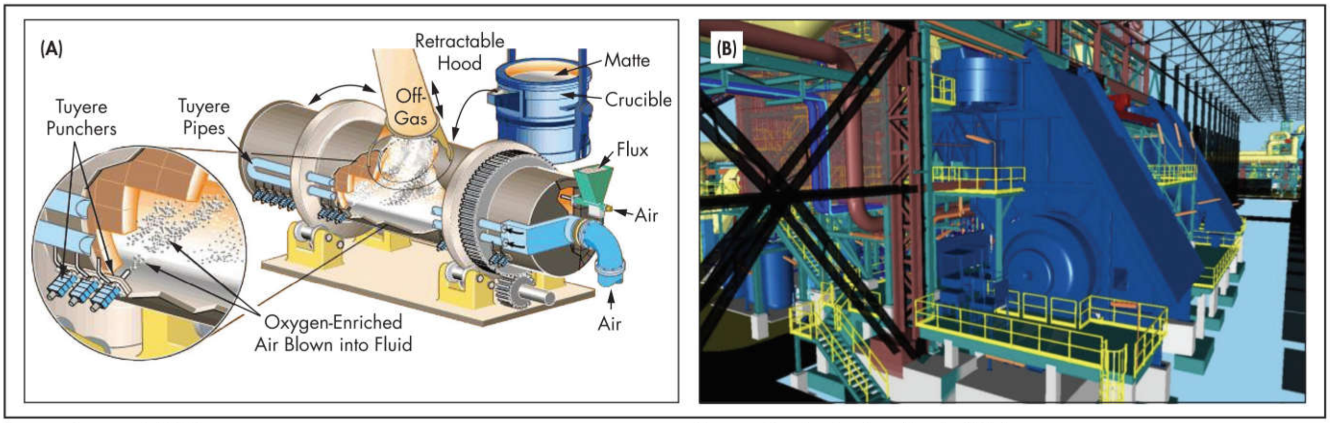
Key ancillary features of the Peirce–Smith converter include

- Air delivery (single-stage compressor, 110–130 kPa),
- Drive mechanism and emergency turnout (generally electric drive),
- Hood (see Figure 3),
- Automatic tuyere punching machine,
- Off-gas handling, and
- Advanced process control instruments and controls.

Table 1 Conversion technology copper plus nickel/PGM new or modernized capacity

Company	Country	Plant (Name) Location	Annual Production, t/yr Cu	Converter Furnace Type	Start and Last Expansion Year
Mitsubishi Materials Corporation	Japan	Naoshima	270,000	**Top lance—Mitsubishi**	(1974) 1991
Mount Isa Mines	Australia	Mount Isa	232,000	Peirce–Smith	1992
Freeport-McMoRan	USA	Miami, Arizona	175,000	Hoboken	1992
Glencore	Chile	(Altonorte) La Negra	375,000	Peirce–Smith	1992
Rio Tinto Kennecott Copper	USA	(Kennecott) Salt Lake City, Utah	272,200	**Suspension/flash—Outotec**	1995
Anglo American	Chile	Chagres	180,000	Peirce–Smith	1995
Pan Pacific Copper	Japan	Saganoseki	472,650	Peirce–Smith	(1973) 1996
Aurubis	Bulgaria	Pirdop	210,000	Peirce–Smith	1996
Glencore	Canada	(Horne) Rouyn-Noranda, Quebec	188,145	**Noranda converter**	1997
Jinlong Copper Company (JV Sumitomo)	China	Tongling, Anhui	300,000	Peirce–Smith	1997
Hindalco Birla Copper	India	Dahej, Gujarat	250,000	Peirce–Smith	1998
PT Smelting (Mitsubishi Materials Corporation JV)	Indonesia	Gresik	271,000	**Top lance—Mitsubishi**	1998
LG-Nikko Copper Inc.	South Korea	Onsan II	262,000	**Top lance—Mitsubishi**	1998
Zhongtiashan Non-Ferrous Metals Company	China	Houma, Linfen, Shanxi	75,000	**Top submerged lance—Ausmelt**	1999
Boliden Mineral	Sweden	Rönnskär	230,000	Peirce–Smith	2000
Yunnan Copper Company	China	Kunming, Yunnan	220,000	Peirce–Smith	2002
Anhui Tongdu Copper	China	Tongling, Anhui	187,500	Peirce–Smith	2003
National Iranian Copper	Iran	Khatoon Abad	80,000	Peirce–Smith	2004
Hindalco Birla Copper	India	Dahej, Gujarat	250,000	**Top lance—Mitsubishi**	2005
Sterlite Industries	India	Tuticorin	325,000	Peirce–Smith	2005
Mopani Copper Mines	Zambia	Mufulira	162,500	Peirce–Smith	2006
Russian Copper Company	Russia	(Karabash) Chelyabinsk	137,500	Peirce–Smith	2006
Anglo American	South Africa	(Waterval) Rustenburg (Ni/PGM)	22,000	**Top submerged lance—Ausmelt**	(2003) 2006
Jiangxi Copper Company	China	Guixi, Yingtan, Jiangxi	700,000	Peirce–Smith	(1985) 2007
Mitsubishi Materials Corporation	Japan	Onahama	180,000	Peirce–Smith	(1965) 2007
Chifeng Jinfeng Copper Industry Company	China	Chifeng, Inner Mongolia	100,000	Peirce–Smith	2007
China Nonferrous Metal Mining Company	Vietnam	Bát Xát, Lào Cai	10,000	Peirce–Smith	2007
Grupo Mexico	Peru	(Southern Copper) Ilo	316,500	Peirce–Smith	2007
Jinchuan Non-Ferrous Metals Corporation	China	Jinchuan, Jinchang, Gansu (Ni/Cu)	125,000	Peirce–Smith	2008
Chifeng Jinjian Copper Group	China	Chifeng, Inner Mongolia	100,000	Peirce–Smith	2008
Dongying Fangyuan Nonferrous Metals Company	China	Dongying, Shandong	125,000	Peirce–Smith	2008
Yunnan Copper Company	Zambia	Chambishi	170,000	Peirce–Smith	2009
Yunnan Copper Company	China	Chuxiong, Yunnan	100,000	Peirce–Smith	2009
Daye Nonferrous Metals Company	China	Daye, Huangshi, Hubei	300,000	Peirce–Smith	2010
Shandong Humon Smelting Company	China	Laishan, Yantai, Shandong	75,000	Peirce–Smith	2010
Yanggu Xiangguang Copper Company	China	Yanggu, Liaocheng, Shandong	400,000	**Suspension/flash—Outotec**	(2007) 2011
Yunnan CopperCompany	China	Liangshan, Sichuan	100,000	Peirce–Smith	2011
Zijin Copper Company Ltd	China	Shanghang, Longyan, Fujian	200,000	Peirce–Smith	2011
Chifeng Fubang Copper Industry Company	China	Linxi, Chifeng, Inner Mongolia	60,000	Peirce–Smith	2011
Xinjiang Xinxin Mining Industry Company	China	Xinxin, Xinjiang	90,000	Peirce–Smith	2011
Kazzinc Ltd	Kazakhstan	Ust-Kamenogorsk	70,000	Peirce–Smith	2011
Baotou Huading Copper Development Company	China	Baotou, Inner Mongolia	100,000	Peirce–Smith	2012
Yunnan Tin Corp	China	Gejiu, Honghe, Yunnan	100,000	**Top submerged lance**	2012
Tongling Non-Ferrous Metals Group	China	Tongling, Anhui	400,000	**Suspension/flash - Outotec**	2012
Wuxin Copper Company Ltd.	China	Ürümqi, Xinjiang	100,000	Peirce–Smith	2013
Jinchuan Non-Ferrous Metals Company	China	Fangchenggang, Guangxi	400,000	**Suspension/flash—Outotec**	2013
Yuguang Gold and Lead Company	China	Jiyuan, Henan	100,000	**Submerged tuyere—SKS/BBC ENFI**	2013
Zhongtiashan Group	China	Yuanqu, Shanxi	125,000	Peirce–Smith	2014
Huludao Copper Group	China	Huludao, Liaoning	100,000	Peirce–Smith	2014
First Quantum Metals	Zambia	Kansanshi	300,000	Peirce–Smith	2014
Rudarsko Topioicarski Basen	Serbia	Bor	100,000	Peirce–Smith	(1961) 2015
Henan Zhongyuan Gold Smelter Company	China	Sanmenxia, Henan	375,000	**Suspension/flash—ENFI**	2015
Dongying Fangyuan Nonferrous Metals Company	China	Dongying, Shandong	375,000	**Submerged tuyere—SKS-SLCR**	Commission
China Minmetal Corporation	China	Hengyang, Hunan	100,000	**Peirce–Smith**	Construction

Note: Alternatives to Peirce–Smith conversion are identified in bold; operations that have been shut down are not included in the table.



Source: Butts et al. 1999 Source: Dundee Precious Metals 2014

Figure 3 (A) Cutaway view showing the main operating features of the Peirce–Smith converter; and (B) modern, fully enclosed hooding devices to capture fugitive emissions

Flow sheet and process description. This section describes the typical process sequence in the operation of the copper Peirce–Smith converter. With the vessel "out of blowing"—turned so the tuyeres are out of the melt—the charge of molten matte produced in the smelting furnace is added by ladle through the mouth of the vessel (Figure 3). Typical copper matte today contains about 60% Cu, 15% Fe, 23% S, and 1%–2% O; there may also be small quantities of trace elements present, such as lead, zinc, and so on. The vessel is then turned to blowing position with the air turned on along with the introduction of silica flux, and the conversion operation commences. Blowing continues in stages until the level of iron is reduced by forming an iron-silicate slag; this blowing cycle is referred to as the "slag blow." The slag is skimmed off and additional ladles of matte added. This blowing–skimming sequence continues two or three times until the iron level is reduced to a few percent, the vessel is considered full, and all the slag has been skimmed. The resulting matte is then essentially present as copper sulfide (also referred to as "white metal"). The "copper blow" then commences, whereby the blowing air oxidizes all the sulfur to form metallic copper with the final melt referred to as blister copper. The bath temperature increases over this cycle because of oxidation of Cu_2S, and cold material such as copper scrap material can be added.

Great skill is required to detect the "end point," and even though new instrumentation is now available to measure the end point accurately, many plants still rely on the long-ago method of sampling the copper bath near the end of the cycle (by an iron bar thrust through a tuyere) and observing the surface of the sample upon cooling. The Peirce–Smith converter effectively handles the elimination of minor elements present in matte; published data are available on this topic (Schlesinger et al. 2011).

The converter off-gas is cooled and cleaned of dust in an electrostatic precipitator and directed to the sulfuric acid plant. Plants in Europe and Japan have high sulfur capture, in particular, and typically capture 99.3%–99.8% of sulfur input to the plant. The converter slag is cooled and copper typically recovered by a grinding-flotation process (Nesset 2011). The resulting high-copper slag concentrate is recycled to the smelting furnace. At some smelters, converter slag is cleaned in an electric furnace.

A typical heat balance for the copper blow of a Peirce–Smith converter is presented in Table 2. This is given in terms of both 1,000 kg of white metal and for a typical 4 m-by-9 m vessel, which until recently was the standard vessel size used in industry. The data in the table can be scaled to any size of vessel (and correcting for heat losses). Generally today, software tools such as HSC Chemistry (Outotec) or FactSage (Thermfact and GTT-Technologies) are used for converter computations. As the converter process is autogenous, fossil fuel energy requirements are low—the main energy requirement is the electrical power to drive the air blowers and fans to control off-gases.

A typical flow sheet for the Peirce–Smith converter within a modern smelter complex is illustrated in Figure 4. It can be seen that, in addition to treating matte from the flash furnace, the Peirce–Smith converters here also treat matte from the electric furnace and the Kaldo/TBRC (top-blown rotary converter) plant. A smelter normally has two or more converters; typically, one (or more) is blowing and one is charging (one will also be down for brick re-lining). The usual arrangement of converter vessels is in the area of the plant known as the "converter aisle."

Furnace size and productivity. Selected physical parameters of a large, modern submerged tuyere converter (Peirce–Smith converter) are presented in Table 3. The total cycle time for a vessel such as the one shown in the table (from empty converter ready to take matte to empty converter after skimming blister copper) is approximately 7–8 hours. There may be two or more such blows per day, equivalent to over 600 t Cu/d. Careful scheduling is required to maximize crane movement for the highest plant efficiency. Specialized software developed for the type of discrete event scheduling encountered here is helpful in optimizing plant operations (Coursol et al. 2009).

Furnace configuration. The Peirce–Smith converter is lined with chrome-magnesite refractory bricks. This lining slowly wears over the course of a vessel campaign; the 4.5 m by 13.4 m vessel presented in Table 3 can have a campaign life of about 3 months, representing more than 130 charges, after which a new refractory lining is installed.

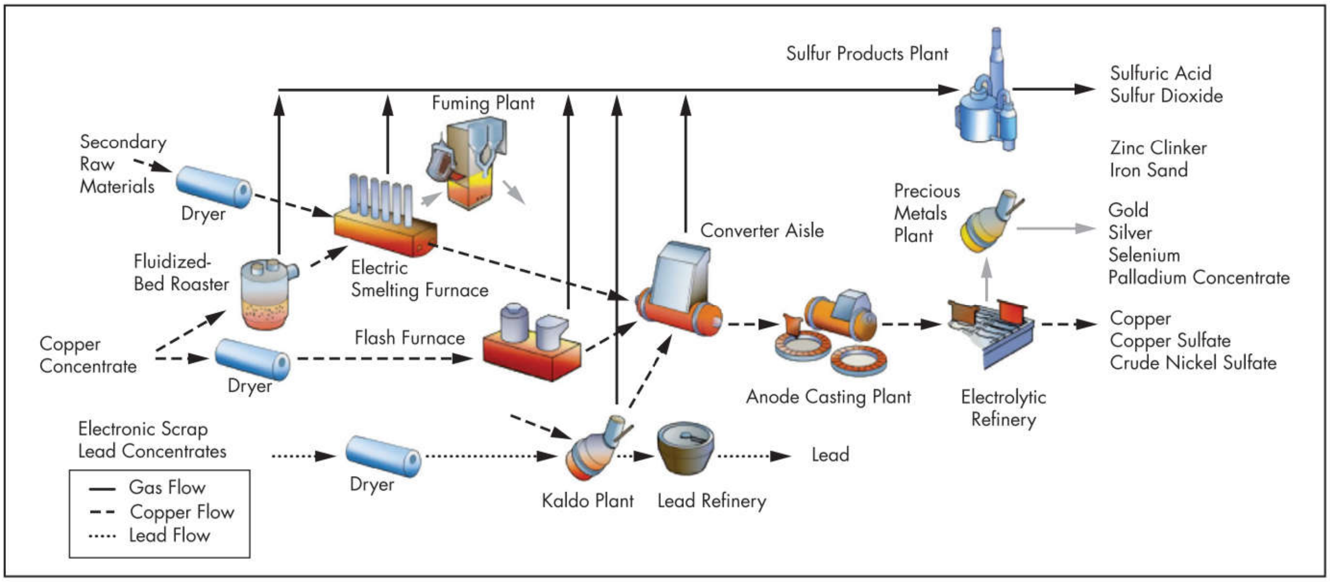
Process control. The modern converter vessel is equipped with advanced process controls for the highest efficiency. All input and output materials are recorded, and all process and

Table 2 Heat balance in the copper blow of a Peirce–Smith converter

Heat Input			Heat Output*		
	Heat Amount, million kcal			Heat Amount, million kcal	
Item	1,000 kg White Metal (Cu_2S)	Charge in Industrial Converter (2-hour blow)	Item	1,000 kg White Metal (Cu_2S)	Charge in Industrial Converter (2-hour blow)
Sensible heat in white metal†	0.209	21.6	Sensible heat in blister copper	0.148	15.3
Heat of reaction	0.326	33.8	Sensible heat of off-gases	0.329	34.0
			Converter heat losses	0.024	2.5
			Heat absorbed by cold copper	0.034	3.5
Totals	**0.535**	**55.3**	**Totals**	**0.535**	**55.3**

*Heat capacity data is from Schuhmann 1952.

†White metal is assumed here at 1,300°C and the blast air is assumed to be at 25°C (heat input).



Adapted from Sundqvist 2010

Figure 4 Peirce–Smith converters in a typical smelter flow sheet

off-gas flows as well as temperatures are also monitored and recorded. Modern instruments are now available to monitor off-gases by optical analysis techniques, typified by the Semtech instrument (Ek and Olsson 2005), to monitor and control the converter end point. The vessels are fully hooded for process and fugitive gas collection.

Advanced process modeling tools such as FactSage or HSC Chemistry are now used to accurately model and track the process during the blow. The paper by Tan (2009) illustrates the good correlation between the observed vessel temperature and that predicted by the model during a slag blow at the Mount Isa smelter in Queensland, Australia. The model can generally predict temperature changes due to stoppages to add matte and/or skim slag over the converter cycle.

Continuous Conversion—Submerged Tuyere Processes

A major advancement in conversion by submerged tuyere technology is the development of a continuous conversion process whereby air or oxygen-enriched air blowing is continuous, with matte additions typically added on an intermittent but regular basis. Blister copper is produced and removed continuously or tapped intermittently. Three commercial processes based on submerged tuyere technology are the Noranda continuous converter, the SKS/BBC, and the SLCR processes.

Noranda continuous converter. The Noranda converter commenced on a commercial scale at the Glencore Horne smelter in Canada in 1998 following several years of testing and piloting (Boisvert et al. 1998; Prevost et al. 1999). The Noranda converter vessel is 4.5 m in diameter and 19.8 m long and is fitted with up to 42 submerged tuyeres for introducing air or oxygen-enriched air, an end-wall port for feeding solid charge, a matte feed mouth, an off-gas mouth, two copper tapholes in the vessel barrel, and a single slag taphole (Figure 5). The operation of the vessel is continuous with air or oxygen-enriched blown continuously, copper and slag tapped periodically while blowing, and high-grade matte from the adjacent Noranda process reactor is transferred by crane and added periodically. The copper is transferred for finishing, then handled conventionally in the anode furnace. After tapping, the slag is treated for copper recovery by milling and flotation. The off-gas, rich in sulfur dioxide, is treated in the smelter acid plant.

Typical metallurgical data for the Noranda converter are presented in Tables 4 and 5. Table 4 shows a typical mass balance, and Table 5 presents a typical heat balance for slightly different operating conditions. In Table 5, the molten matte charge was 1,166.4 t/d (48.6 t/h), the solid charge was 165.6 t/d (6.9 t/h), and the solid carbon addition was 24 t/d (1 t/h). The copper tapped was 864 t/d (36 t/h), and the slag tapped was 182.4 t/d (7.6 t/h). The tuyere blowing rate was 34,750 Nm^3/h, and there was 5,000 Nm^3/h of air at the feed port. The tuyere oxygen enrichment at 23.5% was lower than shown in Table 4; hence, relatively less solid charge could be charged.

Oxygen bottom-blowing continuous converter. The SKS/BBC and the SLCR converter were developed in China over the past few years to provide for continuous conversion by submerged tuyere injection using a horizontal cylindrical furnace akin to a Noranda reactor vessel. The SKS/BBC was first commercialized at the Yuguang copper smelter in China in 2013 (Figure 6; Yie 2014). The smelter incorporates an SKS smelting vessel and the new SKS/BBC. The continuous converter is 4.1 m in diameter and 18 m long with a design capacity equivalent to 100,000 t/yr of cathode copper. The vessel is fitted with the same type of concentric tuyeres as on the SKS smelting vessel. The single continuous conversion vessel is positioned at a slightly lower elevation compared to the SKS smelting furnace so that molten matte at about 74% Cu content flows directly by launder to the conversion vessel, entering via a port in the end wall (Figure 6). Matte addition by launder is generally continuous but can be intermittent for a short period. Several similar continuous conversion vessels have since been commissioned in China.

A continuous bottom-blowing converter process (called SLCR) with a similar concentric tuyere design has recently been implemented in a different mode of operation and on a larger scale at the Dongying Phase II smelter in China. Here, combining conversion and refining in a single vessel, two such vessels are installed that operate in tandem to handle matte from one SKS type smelting vessel, now referred to as the submerged lance smelting (SLS) process. The new converter vessels, 4.8 m diameter by 23 m with 17 tuyeres, referred to as the SLCR process, carry out conventional conversion and then anode refining in the same vessel (Figure 7) so that the final product is anode copper (Luraschi and Schachter 2015; Cui et al. 2016). The Dongying Phase II plant commenced operations in September 2015.

Table 3 Selected physical parameters of a modern, large Peirce–Smith converter treating matte of about 65% Cu

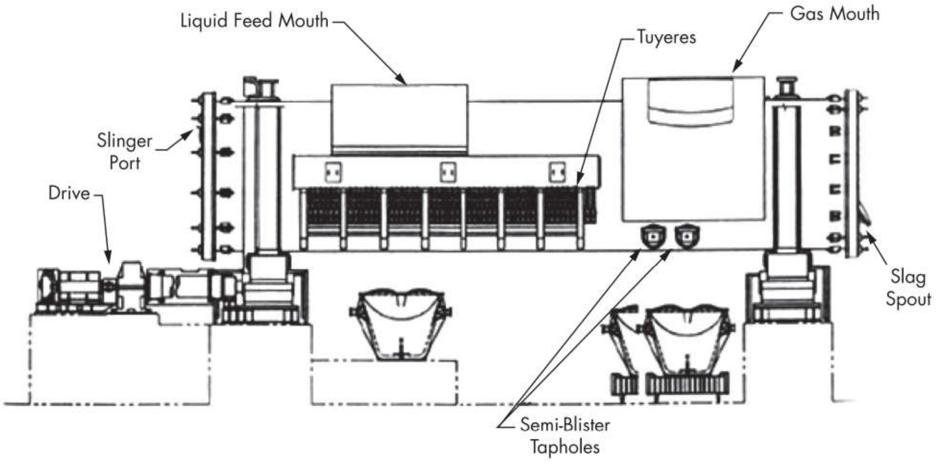
Item	Generic Furnace
Charge capacity, total matte/charge, t	350
Diameter (inside steel dimension), m	4.5
Length (inside steel dimension), m	13.4
Number of tuyeres	60
Tuyere type	Single pipe
Refractory weight, t	210
Blister production,* t/charge	310
Nominal blowing volume, Nm^3/h	46,200
Nominal off-gas volume, Nm^3/h	110,000

Source: Lehner and Wiklund 2001; Ek and Olsson 2005
*Includes the recycle of some scrap copper.

Table 4 Typical mass balance operating data for the Noranda converter

Item	Data
Molten matte from Noranda reactor, t/d	887
Solid coolant addition (solid matte, scrap, etc.), t/d	551
Silica flux, t/d	78
Solid carbon addition, t/d	20
Tuyere blowing rate, Nm^3/h	18,600
Oxygen enrichment, %	41.6
Number of blowing tuyeres in operation	13
Operating temperature, °C	1,218
Copper production rate, t/d	887
Slag production rate, t/d	252

Source: Prevost et al. 2013



4.5 m Diameter x 19.8 m Long
(inside shell)

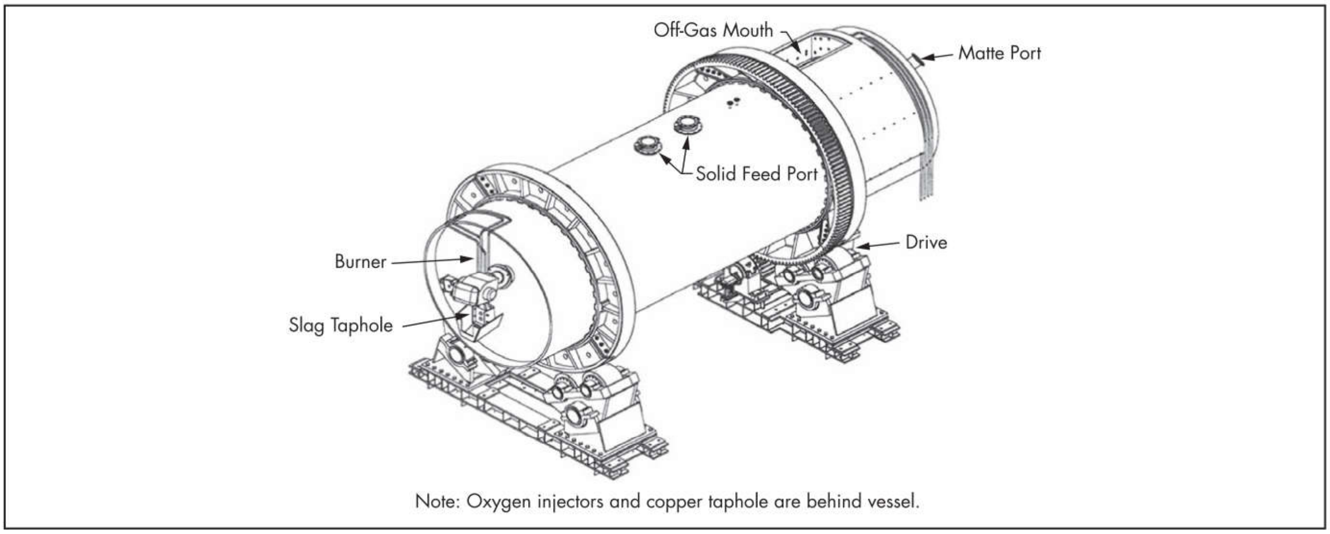
Source: Boisvert et al. 1998. Copyright 1998 by The Minerals, Metals & Materials Society. Used with permission.

Figure 5 Longitudinal section of Noranda continuous converter

Table 5 Typical heat balance operating data for the Noranda converter

Heat Input		Heat Output	
Item	**Million kcal/h**	**Item**	**Million kcal/h**
Sensible heat in molten matte and injected air	8.91	Sensible heat in tapped copper	6.27
Heat of reactions (including coal combustion)	21.22	Sensible heat of off-gases	18.50
		Sensible heat of slag	2.13
		Vessel heat losses	3.28
Total	**30.13**	**Total**	**30.13**

Source: Mackey et al. 1995



Note: Oxygen injectors and copper taphole are behind vessel.

Adapted from Yie 2014

Figure 6 SKS/BBC continuous converter at the Yuguang smelter, China

At Dongying Phase II, matte from the SLS (also known as SKS/BBS) furnace is laundered to one of two SLCR units. The operation of these vessels is as follows: One of the two units starts to be filled with matte to a level that allows concurrent conversion to commence by blowing oxygen-enriched air. After ~400 t of contained copper is received, the matte is redirected to the other unit, converting slag is removed, and refining commences. The sulfur level is lowered, then natural gas and air are used to transform the blister copper to anode copper. Following casting, the unit is readied to take a new charge of matte when the other unit has completed its conversion cycle. Typical operating data for the SLCR process are given in Table 6.

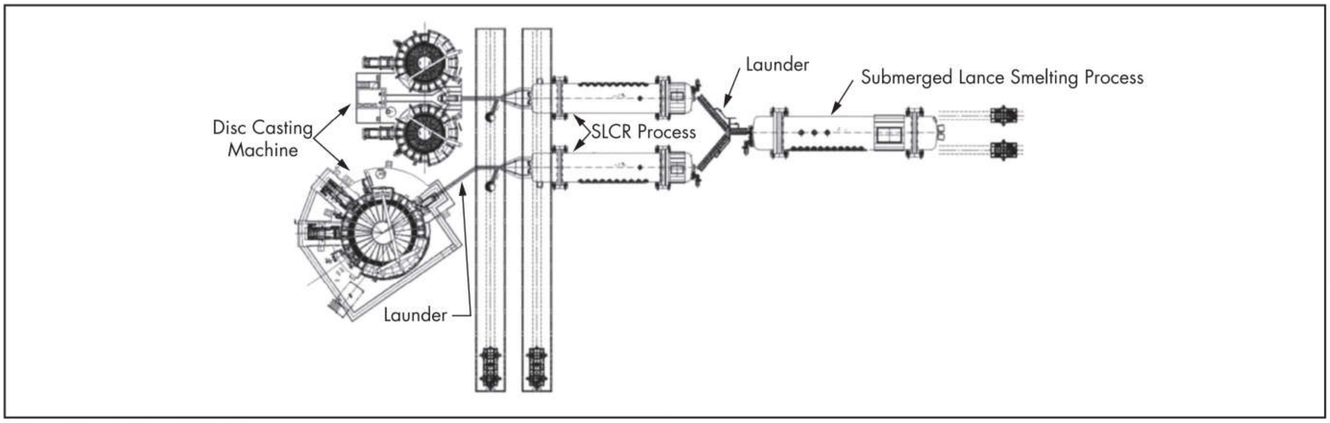
Suspension/Flash Conversion Furnace

Suspension or flash conversion (FC) involves continuous combustion of comminuted solid matte above a molten bath where the combustion heat supports furnace operation at the temperatures required. The first patent to directly describe the general principles of this approach was granted to Haglund in 1940; however, it would be about 55 years later before a commercial plant treating copper matte would be commissioned.

In the late 1940s, Inco investigated applying the principles of their existing suspension smelting technology to conversion (Victorovich 1993). Chalcocite concentrate containing about 2%–4% Ni derived from the selective flotation of copper-nickel matte leaves a mushy nickel oxide residue when converted in the submerged tuyere Peirce–Smith furnace. The objective was to flash-convert the chalcocite concentrate at higher temperature to reduce residue formation. Further work was to wait until the late 1970s, with deployment in 1985 at the Copper Cliff (Ontario, Canada) smelter. The technology used a surplus Peirce–Smith converter modified to convert 3% nickel chalcocite concentrate by a type of FC arrangement and was operated for eight years before being replaced by a larger unit (Queneau and Marcuson 1996).

In the late 1970s, Kennecott started development of the so-called solid matte oxygen conversion process (Richards et al. 1983). Kennecott subsequently entered into a development and marketing agreement with Outotec in the early 1980s (George 2002). In early 1991, a large-scale granulation and subsequent pilot conversion program was initiated to mature the process and furnace technology. Outotec's success with the commercial application of direct-to-blister furnace technology at Glogow (Poland) and Olympic Dam (Australia) increased confidence in scale-up and operation.

In 1995, Rio Tinto Kennecott commenced operation of the first commercial application of suspension conversion using FC technology at their Salt Lake City (Utah, United States) smelter. Outotec FC technology was commissioned at the Yanggu smelter in 2007 (expanded in 2011), the Tongling smelter in 2012, and the Fangchenggang smelter in 2013 (all in China). See Table 7 showing key physical parameters for exemplar FC furnaces. The Sanmenxia (China) smelter commissioned a modified FC technology in 2015 that utilizes a cyclonic burner design developed at the Yanggu smelter.



Adapted from Cui et al. 2016

Figure 7 SLCR conversion furnaces at the Dongying Phase II smelter

Table 6 Operating data for the SLCR process at the Dongying Phase II smelter, China

Item	Data
Matte charging rate by launder from SLS (73%–78% Cu), t/h	50–60
Solid charge addition	Variable
% O_2 in conversion air	21–30
Tuyere pressure, MPa	0.65–0.75
Copper content of SCLR slag (iron-silicate type), %	8–10
Typical conversion/reduction cycle time, hours	16
Casting rate, t/h	100
Casting time, hours	4
Copper temperature, °C	1,250

Source: Cui et al. 2016, with permission from Dongying Fangyuan

Table 7 Physical parameters of exemplar Outotec flash conversion furnaces

Item	Salt Lake City (Utah, United States)	Tongling (China)
Year started	1995	2012
Shaft height (inside dimensions), m	6.5	7
Shaft diameter (inside dimensions), m	4.25	5
Settler length (inside dimensions), m	18.75	21.8
Settler width (inside dimensions), m	6.5	6.7
Blister produced, t/d	1,100	1,240
Off-gas volume, Nm^3/h	16,000	38,000

Source: George et al. 1995; Zhou and Sun 2013

Flow Sheet and Process Description

At Kennecott, screened, granulated matte from the smelting process is conveyed to a matte storage dome where it is radially stacked and reclaimed in a continuous operation. Stockpiled matte can be introduced and stacked with new matte or physically added to the stacked pile using mobile equipment. Capacity can range from 7,000 to 10,000 t. Reclaimed matte is delivered to a combined dryer and grinding circuit consisting of an air heater, vertical roller mill with integrated classifier (similar to those used to pulverize cement and coal), and baghouse cluster to collect product. The dried and ground matte is pneumatically conveyed to a feed storage bin located above the FC furnace. A second storage bin receives pneumatically conveyed burnt lime flux, and a third obtains recycled dust collected in the furnace off-gas system.

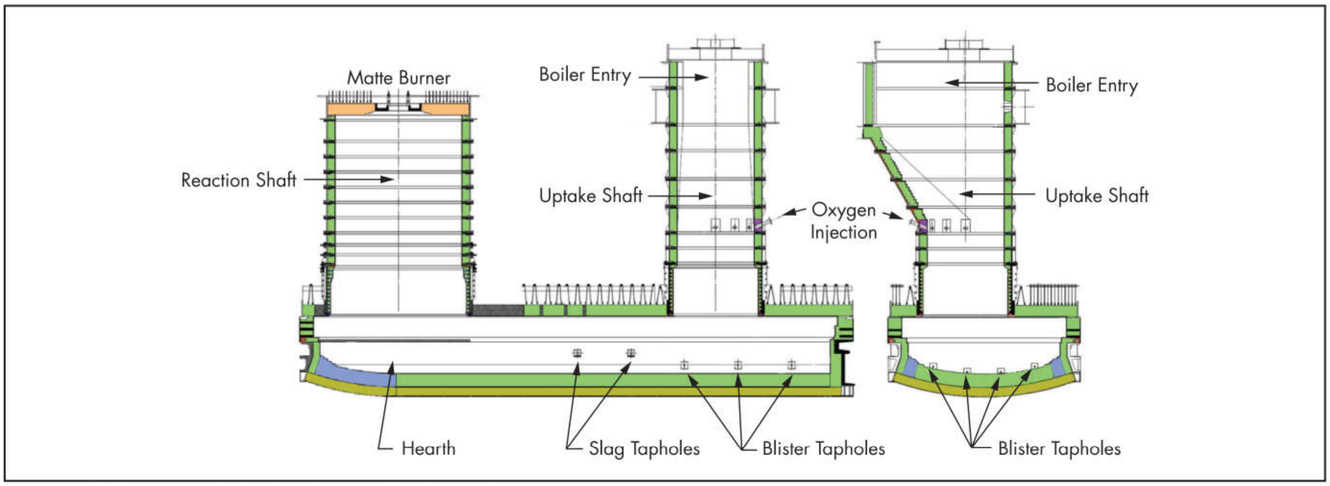
Matte, flux, and dust are fed into the furnace reaction shaft burner using independent loss-in-weight feeders and distribution systems. Oxygen is supplied to the burner (can be greater than 96% in the plenum) to satisfy process requirements, and distribution air is provided at the shaft entry to control plume geometry (effectively reducing process enrichment to ~85% O_2). Blister copper is tapped from the settler into one of two anode furnaces via heated and ventilated refractory-lined and covered launders and catch basins. Slag is tapped and delivered to a granulator via water-cooled copper launders with steel covers to ensure adequate ventilation. Granulated FC slag is screened and conveyed to a storage bin for recycle to the smelting furnace. Provision is made for stockpiling and introduction of this slag as process conditions dictate.

Off-gas from the furnace is directed to the waste heat boiler consisting of radiative and convective sections. The cooled, partially de-dusted off-gas passes by means of a vertical duct to the dry electrostatic precipitator where the majority of remaining entrained dust is removed. The partially sulfated dust that settles in the boiler sections is collected using drag conveyors, commingled with the more fully sulfated dust from the precipitators, and pneumatically conveyed to the furnace dust bin. Depending on process requirements, some of this dust may be diverted to the smelting part of the process.

The cooled and de-dusted gas is ducted to wet gas scrubbers to humidify and cool the gas for subsequent ducting to wet electrostatic precipitators before treatment in the acid plant. System draft is controlled independently using a fan either ahead of or after the wet gas scrubbers.

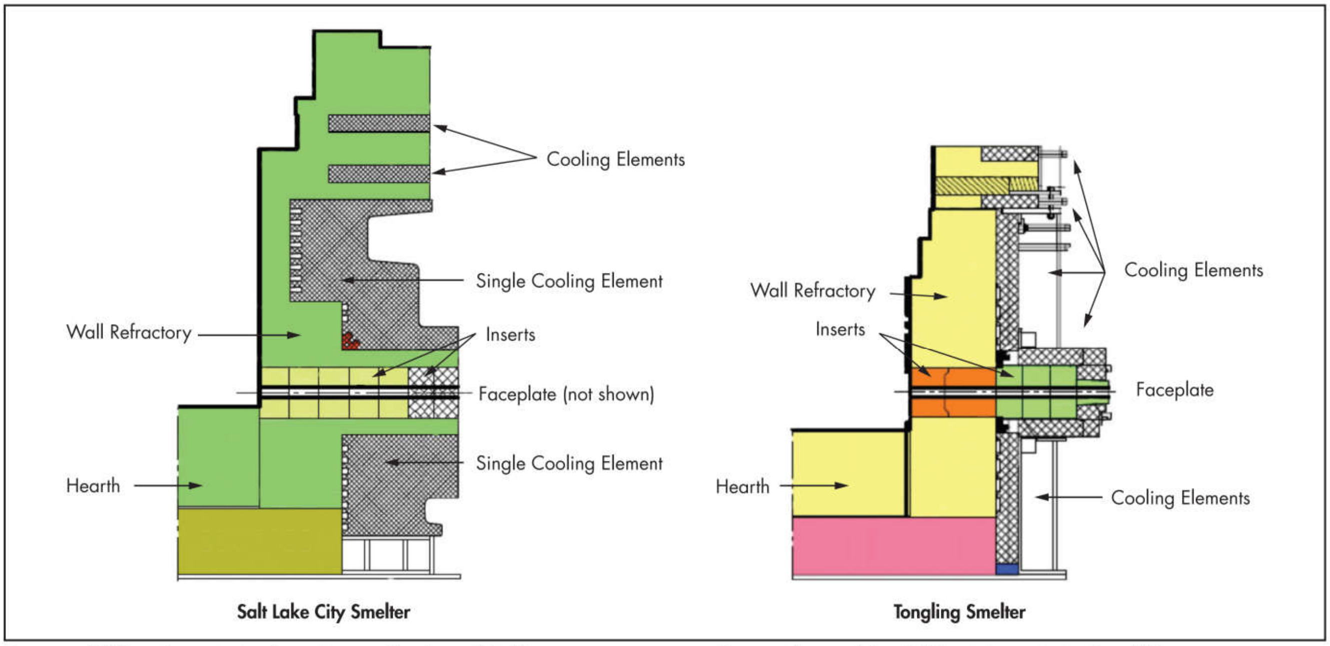
Furnace Configuration

Since start-up of the first flash converter at Salt Lake City, furnace containment design has seen many changes to the original arrangement. The first major permanent changes were performed primarily in response to major melt containment design deficiencies. The changes made are described in more detail by Kennecott authors and are reflected in Figure 8 (Newman and Weaver 2002). Blister taphole design is a critical feature of these furnaces, and Figure 9 contrasts the end wall



Courtesy of RHI, with permission from Outotec, Hatch, and Rio Tinto

Figure 8 Flash conversion furnace elevation longitudinal and lateral sections from the Salt Lake City smelter



Courtesy of RHI, with permission from Outotec, Hatch, and Rio Tinto

Source: Zhou and Sun 2013, with permission from RHI

Figure 9 Flash conversion furnace blister taphole elevation longitudinal sections from two smelters

blister taphole designs for the Salt Lake City and Tongling converters. Both designs are considered reasonably robust.

Ancillary Equipment

Granulation of batch tapped matte and converter slag are also critical support unit operations for flash conversion. Granulated product is typically sized at P_{80} 1 mm. Details of the particular matte and converter slag granulator systems are available (George et al. 2006; Lengyel and Loveless 2015).

Operational Elements

Uniform distribution of matte is important in flash conversion, and the system at Kennecott has evolved to meet this requirement. Blister and slag are typically batch tapped with timing coordinated to ensure that slag layer thickness is maintained as the blister level raises and lowers. Under normal operation, no hearth buildup exists; however, during start-up after an extended outage, the blister layer is cooler and diligence is required in achieving a clear first tap, particularly if sidewall buildup is significant.

Servo and regulatory process control of an FC furnace involves maintaining blister sulfur (set indirectly by control of slag copper content), slag composition, and settler temperature through manipulation of oxygen, flux, and auxiliary fuel, respectively. In addition to fuel, manipulation of enrichment and dust charged can be used for temperature control. These parameters are manipulated at the furnace burner entry with close to constant auxiliary fuel on the settler. To accommodate

Table 8 Example flash conversion furnace mass and heat balance

Material	In or Out	Flow	Temperature, °C	Hf, MJ/h*	Hs, MJ/h*
Flash Converter Shaft					
Dome matte	In	70.000 t/h	25	–47,130.5	0.0
Lime flux	In	1.261 t/h	25	–13,791.6	0.0
Dust charged	In	7.480 t/h	25	–32,287.4	0.0
Shaft air-oxygen	In	10,907 Nm^3/h	25	0.0	0.0
Disperser air	In	1,711 Nm^3/h	25	0.0	0.0
Shaft product	Out	67.259 t/h	1,648	–80,303.4	74,249.4
Shaft flux	Out	1.263 t/h	1,648	–13,800.0	1,924.7
Shaft off-gas	Out	10,762 Nm^3/h	1,648	–119,134.9	39,000.8
Losses	Out	—	—	—	12,552.0
Sum of (out – in)	—	—	—	–120,028.8	127,726.9
Shaft natural gas	In	293 Nm^3/h	25	7,698.1	← Net H
Shaft natural gas off-gas	Out	938 Nm^3/h	1,648	—	4,220.8
Flash Converter Settler					
Shaft product	In	67.259 t/h	1,648	–80,303.4	74,249.4
Shaft flux	In	1.263 t/h	1,648	–13,800.0	1,924.7
Shaft off-gas	In	10,762 Nm^3/h	1,648	–119,134.9	39,000.8
Slag	Out	10.047 t/h	1,250	–42,890.9	11,274.2
Blister	Out	47.330 t/h	1,240	–1,568.3	36,591.6
Dust	Out	4.550 t/h	1,300	–9,874.8	3,921.9
Off-gas	Out	13,011 Nm^3/h	1,300	–148,918.7	36,650.3
Losses	Out	—	—	—	16,736.0
Sum of (out – in)	—	—	—	9,985.7	–10,000.9
Shaft natural gas off-gas	In	938 Nm^3/h	1,648	—	4,220.8
	Out	—	1,300	—	3,465.2
Settler injection gas	In	255 Nm^3/h	25	0.0	0.0
	Out	—	1,300	0.0	485.9
Air ingress	In	850 Nm^3/h	25	0.0	0.0
	Out	—	1,300	0.0	1,556.3
Settler natural gas	In	85 Nm^3/h	25	1,271.5	← Net H
Settler natural gas off-gas	Out	925 Nm^3/h	1,300	—	2,189.2
Waste Heat Boiler					
Furnace off-gas	In	15,978 Nm^3/h	1,300	–164,190.3	44,346.9
Dust	In	4.550 t/h	1,300	–9,874.8	3,921.9
Sulfation air	In	0 Nm^3/h	25	0.0	0.0
Air ingress	In	2,549 Nm^3/h	25	0.0	0.0
Off-gas to electrostatic precipitator	Out	15,558 Nm^3/h	350	–155,753.7	10,816.5
Dust to electrostatic precipitator	Out	4.581 t/h	350	–18,301.8	1,065.7
Dust from waste heat boiler	Out	2.244 t/h	250	–8,979.7	349.3
Losses	Out	—	—	—	1,255.2
Sum of (out – in)	—	—	—	–8,970.1	–34,782.1
Steam produced	—	19.918 t/h	—	—	–43,752.3

*Hf = estimated standard formation; Hs = sensible enthalpies.

these requirements, control systems and procedures similar to those described in Chapter 11.5, "Smelting," for flash smelting furnaces are utilized. Monitoring for process and furnace/equipment integrity is typically similar to that adopted for flash smelting furnaces.

Energy Flows

An example of a mass and heat balance of the Kennecott FC furnace and boiler is shown in Table 8 and is typical of the internal control calculations utilized in real time, discussed earlier. In these examples, separate balances for the reaction shaft and settler sections of the furnace are shown, assuming a nominal shaft sulfur elimination of 80% and shaft oxygen fixed according to Reaction 1. Under these conditions, the calculated shaft exit temperature is quite high compared to the settler. In practice, all control variables are manipulated at the reaction shaft except settler freeboard heating, which is normally targeted to be held constant, resulting in a variable actual reaction shaft exit temperature. This effectively reduces the required balance to a combined, simplified one for control purposes.

Top Lance Conversion Furnace

Top lance conversion involves either batch processing of a charge of solid or molten matte or continuous conversion of a molten matte stream. Top-blown rotary converter (TBRC) furnaces derived from the single-lance Kaldo steel process are used for the batch processes, Outotec and Andritz Maerz being the major technology suppliers. Top lance continuous conversion uses multiple-lance technology with a stationary furnace developed and supplied by MMC.

In 1958, Inco successfully trialed the use of a small Kaldo steel furnace at the Domnaverts Steel Works in Sweden to convert nickel matte to crude nickel (Queneau and Marcuson 1996), leading to subsequent piloting at their Port Colborne (Canada) facility. In 1971, the Copper Cliff smelter in Canada commissioned two TBRCs, producing nickel for subsequent refining.

In 1974, the Tennant Creek smelter in Australia operated two 25-t TBRCs in the belief that good bismuth elimination could be achieved given the efficient mixing, gas contact, and higher operating temperature. While some measure of success was achieved with the conversion operation, the smelter closed shortly after start-up because of operating difficulties coupled with low metal prices. A single 73-t TBRC was installed at the Afton smelter in British Columbia, Canada, for combined smelting and conversion when the smelter commenced in 1978. The plant was generally deemed to have been successful but closed in 1983 because of low copper prices (Nickel et al. 1980).

In 1990, the Stillwater PGM smelter in Montana (United States) commenced operation with two 1.5-t TBRC vessels for converting solid matte produced by an electric furnace (Roset et al. 1992). The plant has expanded several times since then, and larger TBRCs are now used for matte conversion.

TBRCs are now widely employed to treat precious metal–containing alloys from lead production, anode slime from copper electrorefining, and copper scrap/e-scrap. A 10-t TBRC is used at Glencore's Montreal (Quebec, Canada) refinery to process de-coppered anode slimes to produce doré metal (Lessard 1989).

The copper scrap processing facility of Metallo-Chimique NV (now Metallo Belgium) in Beerse, Belgium, treats a range of copper and precious metal scrap materials. Smelting and conversion is successfully carried out in one of the five TBRC units at the plant, including three 150-t units and two smaller ones (Goris 2013).

MMC began development of continuous top lance smelting and conversion processes in 1959 using a small slag fuming furnace at their Naoshima (Japan) facility. A subsequent pilot to demonstration scale-up program from 1968 through the early 1970s at their Onahama (Japan) smelter helped to resolve lance service life and intermediate product transfer issues (MMC 2012).

The Naoshima smelter commissioned the first commercial continuous top lance conversion furnace as part of the integrated Mitsubishi process in 1974 (and expanded in 1991). This was followed by a second captive smelter at Kidd Creek (Canada) in 1981 (shut down in 2010). In 1998, two Mitsubishi-based smelting and conversion facilities were commissioned: one at the Gresik (Indonesia) smelter and one at the Onsan II (South Korea) smelter. The Port Kembla (Australia) smelter replaced their Peirce–Smith converters with a single top lance conversion furnace in 2000 (shut down in 2003). The Dahej (India) smelter adopted the integrated Mitsubishi technology in 2005. See Table 9 showing key physical parameters for exemplar Mitsubishi converters.

Table 9 Physical parameters of Mitsubishi conversion furnaces

Item	Naoshima (Japan)	Gresik (Indonesia)
Year started (expanded)	1974 (1991)	1998
Height (inside dimensions), m	3.6	3.7
Diameter (inside dimensions), m	8.0	9.0
Lance diameter (inside dimensions), m	0.05	0.05
Number of lances	10	10
Blister produced, t/d	950	875
Off-gas volume, Nm^3/h	28,800	27,000

Source: Schlesinger et al. 2011

Mitsubishi Converter

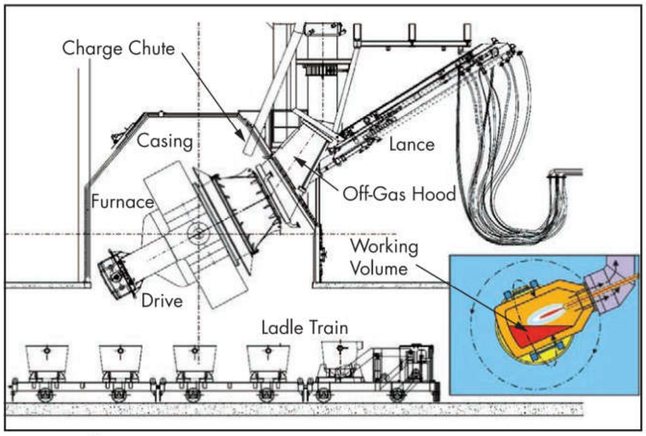
Flow sheet and process description. In a conversion furnace, matte is continuously siphoned from the electric furnace and laundered to the conversion furnace. In the case of a stand-alone converter, the matte is supplied by means of a holding furnace. Oxygen-enriched air (30%–35%) along with limestone flux and granulated slag is, via charge tanks, blown through up to ten rotating process lances to produce blister copper and a calcia-based slag. In addition to using granulated converter slag as a coolant, bundled and shredded scrap is charged to the furnace. The slag continuously overflows from the syphon and is laundered to a granulator for recycling to both the smelting and conversion furnaces. Blister flows through the siphon and is laundered directly to either anode furnaces or a holding furnace.

Furnace size and productivity. Mitsubishi employs advanced furnace design and operation to minimize refractory wear. This includes the installation of specially designed water-cooled elements in critical areas for the furnace (MacRae et al. 1998).

Operation and control. The continuous flow of molten matte to the converter is inferred from measurements made of the launder flowing stream. Total oxygen delivered to the lances is adjusted to achieve target sulfur in blister produced of 0.7%–0.9% indirectly through control of the copper content in slag (14%–15% Cu). Lower sulfur levels are avoided because of the corresponding higher slag aggressiveness to refractory. Expert systems are also used (Goto et al. 1998). Typically, an overall melt depth of 1–1.1 m with 0.12–0.15 m of slag is maintained. A relatively thin layer of slag is targeted to maximize oxygen utilization for a normal lance position of 0.3 m above the bath (Newman et al. 1991).

Top-Blown Rotary Converter

Configuration. The TBRC provides a high degree of mixing of the molten phases and the injected gas with considerable process flexibility. It consists of a rotating vessel (up to 40 rpm) that is tilted to improve mixing and accommodate an inclined oxygen lance (Figure 10).The typical furnace vessel working volume for matte and anode slime applications ranges between 0.8 and 2 m^3. (*Working volume* is defined as the melt volume and is shown in the Figure 10 inset.) Copper and electronic scrap (and also lead concentrate smelting) furnace working volumes are significantly larger and range between 4 and 16 m^3.



Courtesy of Outotec

Figure 10 Elevation of TBRC general arrangement with furnace schematic (inset)

The general TBRC assembly consists of a furnace that is rotated and tilted using hydraulic power transmission, furnace enclosure or casing and charging equipment ventilated to gas scrubbing, water-cooled conversion lance delivering air/oxygen, water cooled off-gas hood, and ladle train to receive process products. Some facilities use a water-cooled utility lance for flame ignition and process monitoring.

Operation. Because the entire operating vessel is encapsulated inside the ventilated casing, the working environment of a TBRC plant is generally very good. Process off-gases are usually wet gas cleaned. If dust is produced as a filter cake material, such material cannot be introduced to an operating unit (normally added to empty vessel and time must be allowed to dry). Smelting and conversion of anode slime starts with charging of free-flowing material via a chute from charging bins with the furnace vessel in the upright position. This type of unit has the flexibility to achieve both conversion and reduction steps as required. After final refining, the doré metal bath is ready, with levels of antimony, bismuth, selenium and tellurium typically less than 0.01% of each (Lessard 1989).

Roset et al. 1992 describe the Stillwater smelter early experiences with using silica-based slags for nickel-copper-PGM matte conversion. Due to operational issues associated with the frequency and magnitude of excessive melt foaming, practice evolved toward using lime as a flux. This resulted in better melt stability throughout the conversion stages. This plant also now handles considerable quantities of electronic scrap (Roset et al. 2014).

Top Submerged Lance Conversion Furnace

Top submerged lance (TSL) conversion involves either batch processing of a charge of molten matte or continuous melting and conversion of comminuted solid matte. The process has evolved as a logical progression from the analogous smelting applications, and in fact was described in the first patent for the so-called SiroSmelt submerged lance (Floyd 1981). The lance and furnace technology is currently supplied by IsaSmelt (owned by Glencore) and Ausmelt (owned by Outotec).

In 1995, the Bindura (Zimbabwe) nickel-copper smelter commissioned an Ausmelt-supplied TSL conversion furnace (shut down in 2000) to process high copper-nickel leach residue in a three-stage residue smelt, matte convert, and slag reduction operation. The first large-scale Ausmelt conversion furnace treating primary sourced material was commissioned in 1999 at the Houma smelter in China (Mounsey et al. 1999). This facility initially operated on a batch basis, with a separate slag blow and a subsequent copper blow.

In 2002, the Rustenberg (South Africa) PGM smelter commissioned a large TSL converter distinguished by matte fed down the lance and described below (Viviers and Hines 2005). Ausmelt converters were commissioned at the Birla (India) smelter in 2003 (shut down shortly after because of design and operational problems) and the Gejiu (China) smelter in 2012. Matusewicz et al. (2007) describe Ausmelt's "continuous copper conversion (C3)" process and discuss the advantages and disadvantages of silica, calcia/silica, and calcia-based slags. Development work conducted at the Houma smelter led to the selection of ferrous calcia silicate slag for the Gejiu converter. Table 10 shows key physical parameters for exemplar Ausmelt-supplied converters.

In 1997, the Hoboken (Belgium) secondary smelter commissioned an IsaSmelt-supplied TSL converter to process complex blast furnace solid matte (Edwards and Alvear 2007). The operation is performed on a batch basis using a high-lead, silica-based slag in the slag blow with a copper blow or conversion stage to produce blister copper, which is granulated and subjected to sulfate leach/electrowinning to recover the copper and precious metals. IsaSmelt also supplied a copper alloy–producing furnace to the Lünen (Germany) secondary smelter, which was commissioned in 2002. This operation processes copper residues and copper alloy scrap in a smelting step to produce a "black" copper (high tin and lead content) and high-zinc slag. IsaSmelt has described concepts (its so-called IsaConvert process) for large-scale continuous TSL conversion using calcia-based slags for copper (Edwards and Alvear 2007) and nickel-PGM smelters (Bakker et al. 2011).

Flow sheet and process description. The Rustenburg PGM flow sheet (Figure 11) involves the granulation of nickel matte and drying/conveying in a spray dryer via heated air to a bin ahead of the conversion furnaces (one duty, one cold standby). Additionally, matte received via road from other PGM smelters is pneumatically conveyed to dedicated bins. The blended matte is pneumatically conveyed into the converter via the lance. The product matte is slow cooled in molds for subsequent processing to refined precious metals and nickel. Converter slag is granulated and dried for subsequent cleaning to recover contained metal.

The furnace off-gas leaves the converter by means of a Foster Wheeler high-pressure water membrane freeboard and uptake and is then de-dusted and cooled in an evaporative cooler and sent to an acid plant. In the integrated flow sheet at the Gejiu smelter (Figure 12), matte produced from the smelter is separated from the slag in a settling furnace and transferred to the TSL converter either directly via launder (as at Houma) or granulated and subsequently fed through a bin.

Containment principles. The Houma and Gejiu conversion furnaces of Ausmelt design use a chrome magnesite refractory and water-cooled furnace shell. Refractory life of 18 months has been reported with this arrangement. Viviers and Hines (2005) describe the Rustenberg smelter rationale for adopting high-intensity Hatch copper-cooler technology and a water-cooled freeboard for its Ausmelt technology-based converter (Figure 13). Service life has been very good with the changes adopted.

Operational elements. The Houma smelter is reported to operate their converter in batch mode with two stages. The first stage involves laundering molten matte from the settling furnace with concurrent continuous conversion to white metal via lance/bath action. Upon maximum operating level being achieved, feed is interrupted and the slag tapped. The second stage involves conversion of the white metal to blister copper as a batch. It is reported that to accommodate this batch operation, the lance turndown ratio is designed to be 3:1 (Mounsey et al. 1999). Prior to 2007, the slag used was a silica-based, or fayalite, slag. After joint development with Ausmelt, a ferrous calcia silicate slag was adopted, the C3 process (Abbott 2007).

The Gejiu smelter based their converter design and initial operation on the Houma experience. Following commissioning it was decided to change from the two-stage batch process to an almost fully continuous blister-making process, whereby the granulated matte feed is interrupted during slag tapping to allow settling of any entrained copper (Outotec, personal communication).

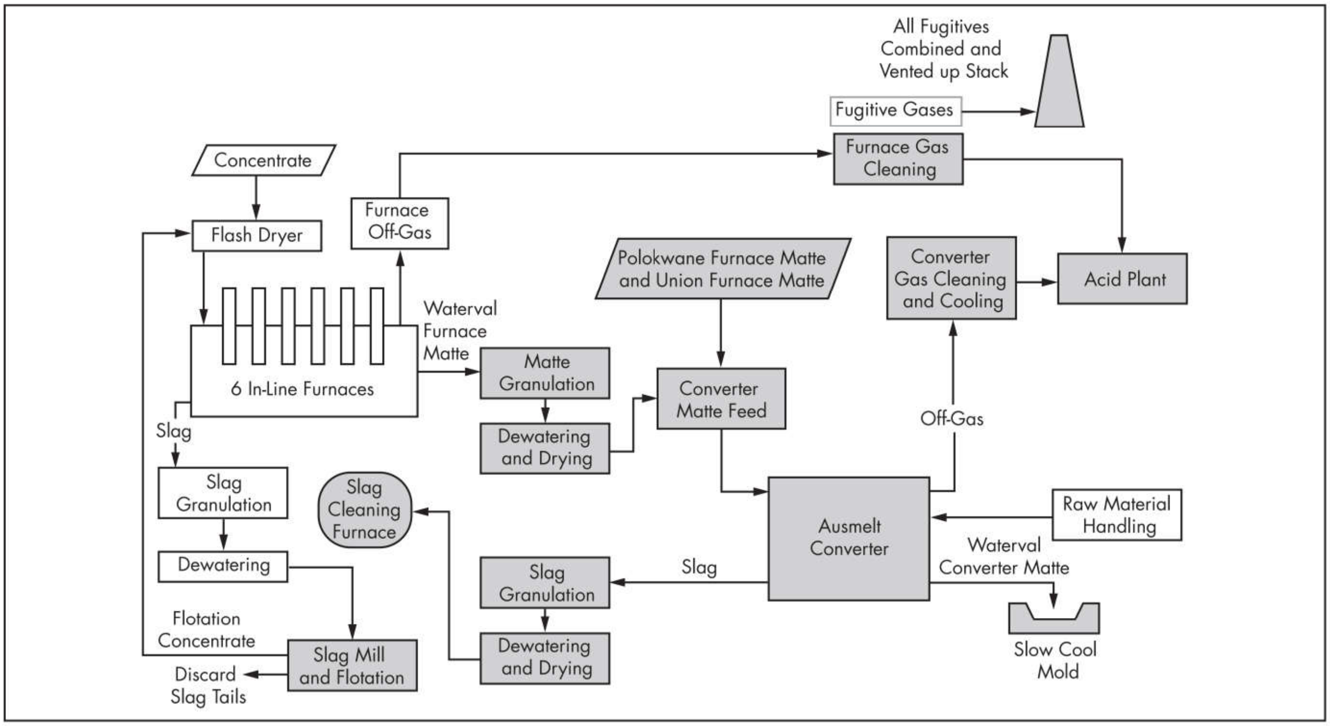
The Rustenberg PGM smelter reports that its ability to satisfy a very tight tolerance of 3.0 ± 0.5% Fe in final nickel matte was significantly enhanced through a continuous conversion (Viviers and Hines 2005). Operation of the converter was initially intended to be in batch mode with two stages. The first stage involved concurrent feeding and conversion to 13% Fe, the intention being to minimize oxidic metal losses in the initial slag. The second stage involved interruption of feed and subsequent batch conversion to the final nickel matte target composition. This mode of operation proved operationally difficult because of melt stratification, making it difficult to sample while in operation and also resulting in frequent, and violent, bath foaming. Additionally, variable starting conditions for the second stage made it difficult to achieve final target composition. Continuous operation of the converter was adopted because of these problems and involves concurrent feeding and conversion to the final target of 3% Fe with nickel in slag used to control air/oxygen flow rates via the lance. Matte is tapped before the slag, with matte removed until slag is detected to ensure the matte level does not approach the slag taphole. This mode of operation has proven to be a success because of easier lance level control, elimination of matte carryover–induced granulator explosions, product quality that is not impacted by feed rates changes, achievement of constant gas flow to the acid plant, and reduced cycle time through feeding without interruption. However, oxidic losses to the slag are significantly higher, requiring a change in operation of the cleaning furnace.

Table 10 Physical parameters of Ausmelt top submerged lance conversion furnaces

Item	Houma, China	Rustenberg, South Africa	Gejiu, China
Year started	1999	2002	2012
Height (inside dimensions), m	12.6	14.8	~16
Diameter (inside dimensions), m	4.4	4.4	4.4
Lance diameter (inside dimensions), m	0.35	0.4	0.4
Metal/matte produced	~230 t/d metal	~650 t/d matte	~300 t/d metal

UNIT OPERATIONS FOR FERROUS AND NONFERROUS REFINING

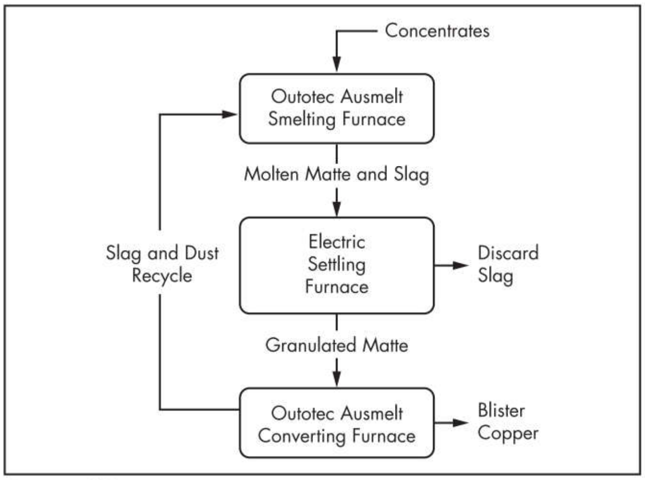
A massive increase in primary steel production occurred during the 20th century. Pig iron production in 2014 was 1,190 Mt (USGS 2015), up from about 30 Mt in 1900, a 40-times increase. Most of the pig iron produced in 2014 was refined to various grades of steel using a combination of top lancing and submerged injection of oxygen and inert gases such as a basic oxygen furnace and derivative technologies. Ferronickel



Adapted from Viviers and Hines 2005

Figure 11 Flow sheet for the Rustenberg, South Africa, PGM smelter

was not a production commodity in 1900. The first large-scale electric furnace smelting of calcined laterite ore to produce ferronickel commenced in the 1950s (Thurneyssen et al. 1960). Steel-grade consistency and range also advanced with the adoption of secondary steel refining ladle and degassing technologies. The reader is referred to Barker et al. (2012), Kor and Glaws (2012), and Miller et al. (2012) for a comprehensive treatise on ferrous unit operations.



Courtesy of Outotec

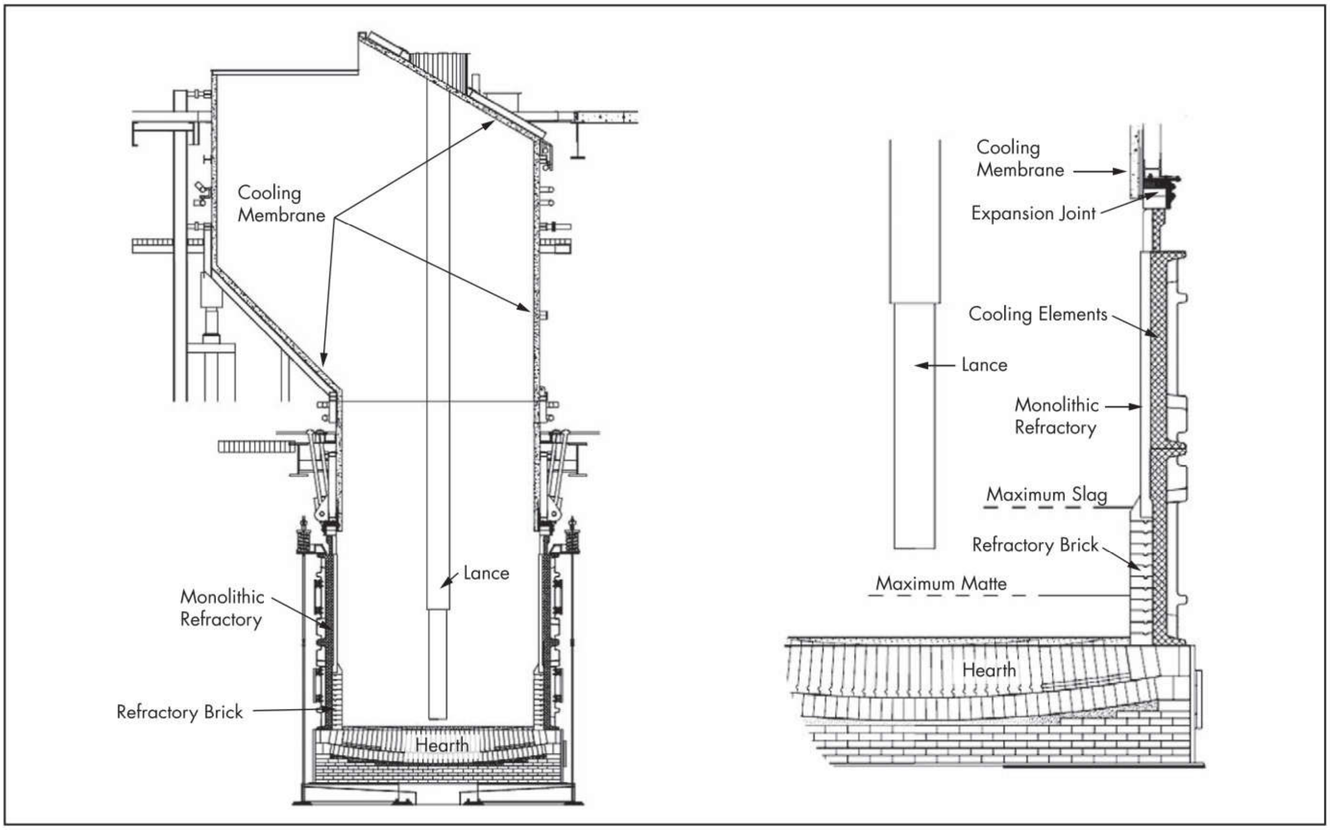
Figure 12 General integrated flow sheet used at the Gejiu, China, smelter

Primary copper production in 2014 was 18.7 Mt, up from about 0.6 Mt in 1900, about a 30-times increase. More than 80% of copper produced in 2014 was electrolytically refined from anode copper. Current submerged tuyere anode refining technology was adopted around the middle of the 20th century and has remained dominant given its relative simplicity along with improvements—some in part as a result of steel industry technology influences such as porous plugs and oxygen/fuel burners as now used in copper operations.

Primary lead and tin in 2014 was 5.5 Mt/yr and 0.3 Mt/yr, respectively, mostly derived from batch drossing processes using large kettle technologies. High-productivity continuous drossing technologies have also been employed for lead refining. The reader is referred to Fern and Shaw (1983), Knight and Reader (1996), and Moor (2000) for descriptions of lead unit operations.

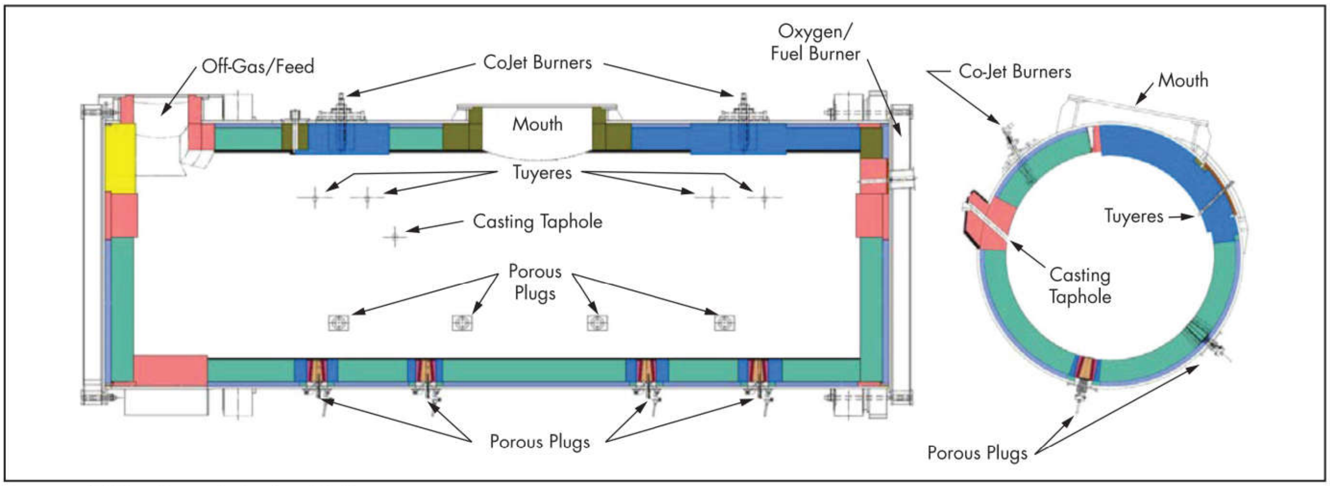
Copper Anode Furnace

At the start of the 20th century, demand for high-quality electrolytically refined copper for electrical applications was rapidly expanding, placing greater demand on the throughput and quality of anode copper production. Although rotary furnaces had been in use for some time, one of the first large-size rotary anode furnaces was the 3.9 m diameter by 9.1 m long unit installed in 1949 at the New Cornelia smelter at Ajo, Arizona, United States (Byrkit 1953), but installation and operation of the reverberatory furnace remained common up to the 1950s–1960s. Of interest is the conventional 130-t reverberatory furnace installed at the Gaspe smelter in Canada as late as 1955 when the smelter started operations; however, it was



Courtesy of Hatch, with permission from Anglo American Platinum

Figure 13 Anglo conversion process furnace elevation section general arrangement from the Rustenberg smelter



Courtesy of RHI, with permission from Rio Tinto

Figure 14 Anode refining furnace elevation longitudinal and lateral sections from the Salt Lake City smelter

later replaced with a rotary anode furnace (the Gaspe smelter closed in 2002).

The initial operating practices for rotary anode furnaces reflected the then-established practice of using green birchwood poles thrust into the melt for deoxidation (referred to as "poling"). The first use of gaseous reductant occurred in 1961 at the Douglas smelter in Arizona (McKerrow and Pannell 1972). Today, rotary anode furnace technology is basically the only choice available for refining blister copper, regardless of the source.

Configuration and Layout

The electrolytic copper refining process requires a flat, relatively smooth anode surface to be cast for best results. To achieve this, casting operations require copper having low sulfur (<0.003% S) and low oxygen (<0.12% O) contents.

The first step in blister copper refining is blowing with air using a submerged tuyere to remove the last traces of sulfur, followed by a deoxidation or reduction step involving the injection of natural gas or fuel oil, sometimes along with superheated steam, to remove the oxygen picked up by the oxidation step. The finished melt is cast as anodes typically on a circular casting wheel. Anodes today may be up to 950 mm long by 900 m wide by 50–55 mm thick and can weigh ~400 kg.

The submerged-tuyere anode furnace consists of a refractory-lined rotary furnace equipped with a fuel (oil or gas) burner and port, furnace mouth for charging blister copper from Peirce–Smith converters via crane as well as to act as an off-gas port, a copper tapping spout, one to four submerged tuyeres and ancillaries, porous plugs for melt agitation, and the furnace driving mechanism. For furnaces receiving laundered copper from converter alternatives to the batchwise Peirce–Smith converter, a separate off-gas port, also acting as a feed port, is usually used (Figure 14). Slag removal from the mouth is usually more frequent, and porous plug agitation more critical.

Generally, scrap charging through the mouth must also be practiced in these facilities. Some furnaces, such as those deployed at the Salt Lake City smelter, employ Praxair CoJet burners to assist with efficient scrap melting and temperature control.

Tuyere operation involves high-pressure air injection (typically 400–500 kPa) for oxidation and introduction of reductant, typically high-pressure natural gas or fuel oil, for de-oxidation. The tuyere consists of a 25-to-30-mm-diameter stainless-steel pipe (often concentrically set within a larger outer pipe).

Submerged-tuyere anode furnaces in use today are commonly sized about 4.3 m diameter and about 10 m long and hold a 270-to-300-t copper charge. The airflow for oxidation is typically 360–400 Nm^3/h, and the oxidation cycle can range from 0.5 to 0.75 hours. When natural gas is used for deoxidation, the typical flow rate is about 600 Nm^3/h per tuyere, with the deoxidation cycle lasting 3 hours (with two tuyeres operational).

As smelters have increased in size, the size of the anode furnace has also increased. Recent large FC plants have installed anode furnaces up to 4.9 m diameter by 14.2 m long, with a copper charge capacity of more than 550 t.

CONCLUDING REMARKS

In this chapter, the authors have attempted to balance description of the current major technologies employed for conversion and refining for the ferrous and nonferrous industries in the context of a historical narrative. In doing so, they hope to have demonstrated that changes in practice have been, and in all likelihood will continue to be, directed by end-use market demands, raw materials and energy availability and efficiency, production cost, and environmental and workplace hygiene imperatives. Historically, there are many examples where these influences have been so strong that change has been quite rapid (e.g., post–Second World War steel markets worldwide and the adoption of basic-oxygen-furnace technology). However, it is far more frequent in conversion and refining that these forces result in a steady direction away from less-efficient established practice to better alternatives—for example, the vastly greater productivity of horizontal rotary vessels (Peirce–Smith converters and anode refining furnaces) over reverberatory furnaces, development of better capture of fugitive gases from Peirce–Smith converters, adoption of porous plug agitation and CoJet steelmaking technologies by the copper-nickel industry in anode refining, large-scale continuous copper conversion in an FC furnace, and so on.

As practices improve, the ability of the broader conversion and refining industry to adopt these improvements depends on the technical risk, complexity, and the incremental capital cost involved balanced against gains in the cost of production. These factors evolve much more rapidly in an industry that is actively expanding. This is why Europe was the center of change in the 1800s, both North America and Europe were centers of activity in the 20th century, and China has been, and will probably continue to be, the most active in the 21st century.

It is instructive to look at the development of SKS/BBC and SLCR technologies as a 21st-century example of what can result from the "churn" that comes from rapid industry expansion. These technologies are an integration of largely existing ferrous and nonferrous western technologies (e.g., the Savard–Lee shrouded tuyere; Kapusta et al. 2015) specifically adapted in China to meet a strategic need for an alternative for retrofitting the country's numerous modestly sized copper smelters. The apparently successful commercial application in smelting (SKS/bottom-blowing smelting) clearly reduced the perceived technical risk involved in adopting the same approach for copper conversion (BBC) in several new, and large, facilities. And along the same trajectory, this has recently been extended to integrating conversion and refining in the same vessel, known as the SLCR process developed in China (Cui et al. 2016).

History reveals that there is significant value in maintaining cross-industry awareness of developments. The scale of pig iron–derived steel refining is 80 times larger than that of copper conversion and refining. Operational practices and vendor development technologies of the larger industry can probably continue to benefit an astute operator in the smaller industry. The final matte produced in the smelting/conversion of nickel ores/concentrates is now treated hydrometallurgically, and zinc is now almost exclusively won and refined hydrometallurgically after roasting the sulfide concentrates. Can the same trend be expected for copper concentrate treatment?

At the current scale of copper concentrate smelting, with its substantial by-production of sulfuric acid, relatively benign slag, and excellent environmental performance, it seems unlikely that the large tonnage processes of smelting and conversion will be replaced in the foreseeable future. However, given that the majority of concentrates are now shipped long distances to third-party custom smelters, could these facilities be motivated to adopt alternative hydrometallurgical treatment routes for blister copper using processes that lower energy requirements and reduce working inventory costs?

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PART 12

Processing of Selected Metals, Minerals, and Materials