Energy Consumption in Copper Sulphide Smelting

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Abstract

Using thermochemical modeling and industrial data, energy consumption in copper sulphide concentrate smelting was calculated for the following processing routes: a) Flash smelting + flash converting; b) Isasmelt smelting + Peirce-Smith (PS) converting; c) Mitsubishi continuous copper smelting and converting; and d) Noranda/Teniente continuous bath smelting + PS converting. For all routes, the battery limits of the exercise included value metals recovery from slag, heat recovery from process gas streams, and abatement of SO_2 from both process gas streams and secondary streams. A unit operation approach was adopted in this study, thus permitting rapid evaluation of other flowsheets such as “flash smelting + PS converting”. Although natural gas was the main fuel used in the computations, the model can be readily used to explore the impact of alternative fuels. Possible further energy savings and reduction of greenhouse gas emissions are examined.

1 Introduction

It is more than 30 years since H. H. Kellogg and J. M. Henderson published their seminal paper on “Energy Use in Sulfide Smelting of Copper” [1]. These authors not only calculated the direct fuel and electrical energy consumption for nine different pyrometallurgical routes, but also evaluated the energy required to produce main process supplies, thus pioneering copper life cycle assessment. Since the publication of this classic study, the practice of anode copper production from sulphide concentrates has experienced many changes. The reverb furnace, that was the dominant copper smelting technology in the 1970s, has been replaced by far more energy efficient and environmentally sound flash and bath smelting technologies. Top submerged lance smelting has become an important processing route in recent years. Continuous converting is currently challenging the one-hundred-year old Peirce-Smith converter. Tonnage oxygen consumption in both smelting and converting has substantially increased, thus fulfilling Paul Queneau’s dream of “lifting the dead hands of nitrogen from oxidation reactions which utilize the oxygen in air” [2]. Throughout the world, increasingly more stringent environmental regulations are being imposed on the industry, and emissions of greenhouse gases are being examined in depth as part of the worldwide strategy to slow down climate change.

The present authors considered, then, that it was time to revisit the subject by examining some of the more energy efficient and environmentally acceptable routes currently used for the production of anode copper from sulphide concentrates. Toward this end, they selected the following four flowsheets: (a) Flash smelting + flash converting; (b) Isasmelt^{TM} smelting + Peirce-Smith converting; (c) Mitsubishi continuous copper smelting and converting; and (d) Noranda/Teniente continuous bath smelting + Peirce-Smith converting. In this study, the authors adopted the same unit operation approach originally used by Kellogg and Henderson. This permits rapid evaluation of other flowsheets such as “flash smelting + PS converting”.

The objective of the exercise was to provide answers to questions such as: What has been the net energy payout by substituting new technologies for reverb smelting? What has been the corresponding decrease in CO_2 emissions? How big a price is industry paying in its quest for sustainability? Which are the unit operations and secondary materials management techniques that offer further significant energy savings?

Due to time constraints, the present authors have postponed undertaking the more ambitious goal of determining the copper life cycle assessment from concentrate to anode copper. They will deal with this matter in a future paper.

2 Methodology

The methodology adopted to develop smelting energy data in this study was based in part on the approach used by Kellogg and Henderson [1]. A standardized flowsheet was developed for each of the leading smelting process routes that were selected. A Metsim model previously developed by Tripathi et al. [3] was used as a basis for designing four new models for this study. Heat and mass balance, and energy consumption data were then computed for each process for subsequent analysis and comparison. Process “boundary limits” were: Inputs - wet concentrate, flux and other consumables delivered to the smelter day bins; and Outputs - copper anodes, sulphuric acid, acid plant tail gas and cleaned fugitive gas released to atmosphere, and cleaned slag. Waste heat recovery from process gas streams was also included as part of the study. As well, a literature review for previous relevant information was conducted.

The four processing routes examined in this study are listed below. The respective flowsheets are presented in the next section.

1. Outokumpu Flash Smelting + Kennecott-Outokumpu Flash Converting
2. Isasmelt Smelting/RHF Matte/Slag Settler + PS Converting
3. Mitsubishi Continuous Smelting Process
4. Noranda/Teniente Continuous Bath Smelting + PS Converting

Energy consumption in auxiliary unit operations and energy equivalent for process supplies were computed from a set of unit energy consumption factors. These are presented in Table 1 below.

Table 1: Unit energy parameters used in this study

Item	Unit Energy	Ref.
Steam dryer	2 tonne steam/tonne water evaporated	Authors
Conversion of steam to electricity at smelter	6.25 kg of steam/kWh*	Authors
Tonnage oxygen (production + gaseous surge capacity)	285 kWh/tonne of oxygen	[14]
Compress tuyere air (~110 kPa)	0.05 kWh/Nm^3	[1]
Compress lance air (~ 60 kPa)	0.03 kWh/Nm^3	Authors
Process off-gases handling	0.0085 kWh/Nm^3	Authors
Fan-secondary gases	0.002 kWh/Nm^3	[23]
Furnace cooling water	3 kWh/tonne Cu	Authors
Matte granulation and handling	9 kWh/tonne of matte	Authors
Slag granulation and handling	3 kWh/tonne of slag	Authors
Matte comminution and handling	10 kWh/tonne of matte	Authors
Lighting and misc power (allowance)	30 kWh/tonne of Cu	[1]
Acid plant operation (double contact)	[(646.8/%SO_2) + 63.7] kWh/tonne of acid	[1]
Energy - Flux	(90 MJ + 3kWh)/tonne of flux	[1]
Energy - Limestone calcination (CaO flux)	7000 MJ/tonne of CaO	Authors
Energy - Wear steel in slag milling	20.7 MJ/Kg of steel	[21]
Energy - Pig iron	15.5 MJ/Kg of pig iron	Authors

Notes: * Based on a rate of 5 kg steam/kWh and an operational efficiency of 80 % to account for potential losses on start-up/standby etc.

Appropriate data was taken from Kellogg and Henderson [1], and updated information was used in other cases. Examples of auxiliary unit operations are: producing tonnage oxygen, compressing PSC injection air, delivering low-pressure air to burners, moving process off-gas, drying concentrate and other materials, transporting and injecting fine solids suspended in a stream of gas (dense phase transportation of solid particulates).

3 Brief Description of Processes

Flowsheets of the four selected processes are presented in Figures 1 to 4. Specific references are given in each brief process description. Data from the 2003 worldwide copper smelter survey [4] was used to fill gaps in these flowsheets. Each process route also included the following “standard” unit operations: (a) Complete secondary gas collection and cleaning; (b) Anode refining and casting; and (c) Process gas treatment in a double-contact acid plant with acid delivery to storage tanks. As noted above, heat recovery from process off-gases was also considered. Concentrate and other solid process streams requiring drying were treated in steam dryers using waste heat steam. Surplus steam was assumed to generate electricity.

3.1 Outokumpu Flash Smelting + Kennecott-Outokumpu Flash Converting

The Utah copper smelter flowsheet [5] was used for developing the model for this process route (see Figure 1 below).

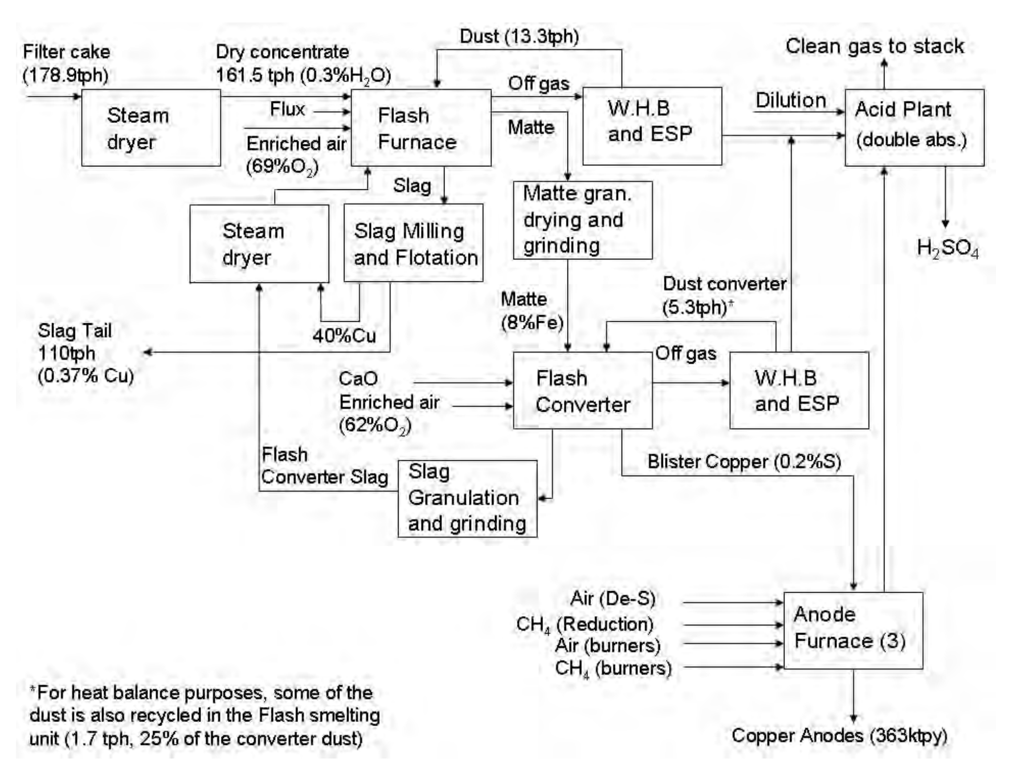

Figure 1: Outokumpu Flash Smelting + Kennecott-Outokumpu Flash Converting

Dry concentrate is fed to the flash furnace. The product matte (~70 % Cu) is granulated, ground and dried. It is, then, fed to the flash converter to produce ~0.2 % S blister. A lime-ferrite slag, compatible with the high oxygen potential required to produce relatively low-sulfur copper, is used in the converter. This slag is granulated and returned to the flash furnace. In turn, the slag from the flash furnace is controlled cooled and milled for copper recovery. The slag concentrate is also returned to the flash furnace. Except for minor proportions, dust is recycled to the respective generating furnace.

3.2 Isasmelt Smelting + PS Converting

Isasmelt is one of the two “top submerged Lance” (TSL) technologies currently practiced for copper concentrate smelting. It was chosen for this exercise, because it has demonstrated annual throughputs over one million tonnes of concentrate in various locations. The authors’ model was built based on published flowsheet information on the modernization of the Ilo Smelter [6-8] (Figure 2 below).

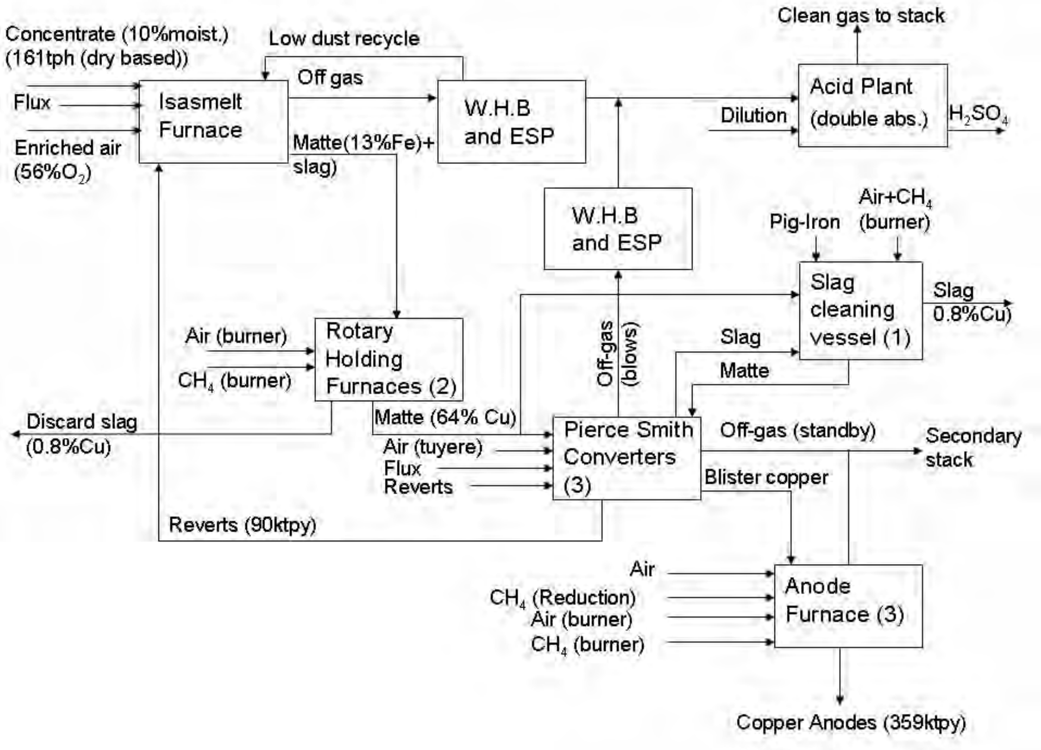

Figure 2: Isasmelt Smelting + Peirce-Smith Converting

An agglomerated mixture of concentrate, flux, recycled dust and coal is fed to the vertical stationary reactor, and oxygen-enriched air and supplementary gaseous or liquid fuel are injected through the vertical lance. The product matte (60-65 % Cu) and slag are jointly transferred to a rotary holding furnace settler. An electric furnace is used in some Isasmelt smelters for matte/slag separation. The slag from the settler, usually containing <1 % Cu, is discarded. The matte is converted to blister in PS converters. Copper is recovered as matte from the converter slag in a cylindrical-horizontal tilting vessel, using pig iron as reductant. In the authors' view, converter slag milling could be considered as an alternative.

3.3 Mitsubishi Continuous Smelting Process

The authors' model is based on the Mitsubishi three-furnace operation as described by M. Goto et al. [9] (see Figure 3 below).

In the smelting furnace, dry concentrate and other feed materials, including dust from the smelting and converting furnaces, are injected directly into the matte in a stream of oxygen-enriched air via vertical lances. Matte (68 % Cu) and slag flow jointly and continuously into an electric furnace settler. Molten matte is, in turn, continuously transferred from the settler to the converting furnace, while the slag (~0.7 % Cu) is discarded. In the converting furnace, limestone flux is injected into the molten bath in a stream of oxygen-enriched air also via vertical lances. Blister copper (0.75 % S) and lime-ferrite slag, each at constant composition, coexist in the converting furnace. The two melts are continuously and separately discharged from the converter. Granulated converter slag is recycled to the smelting furnace.

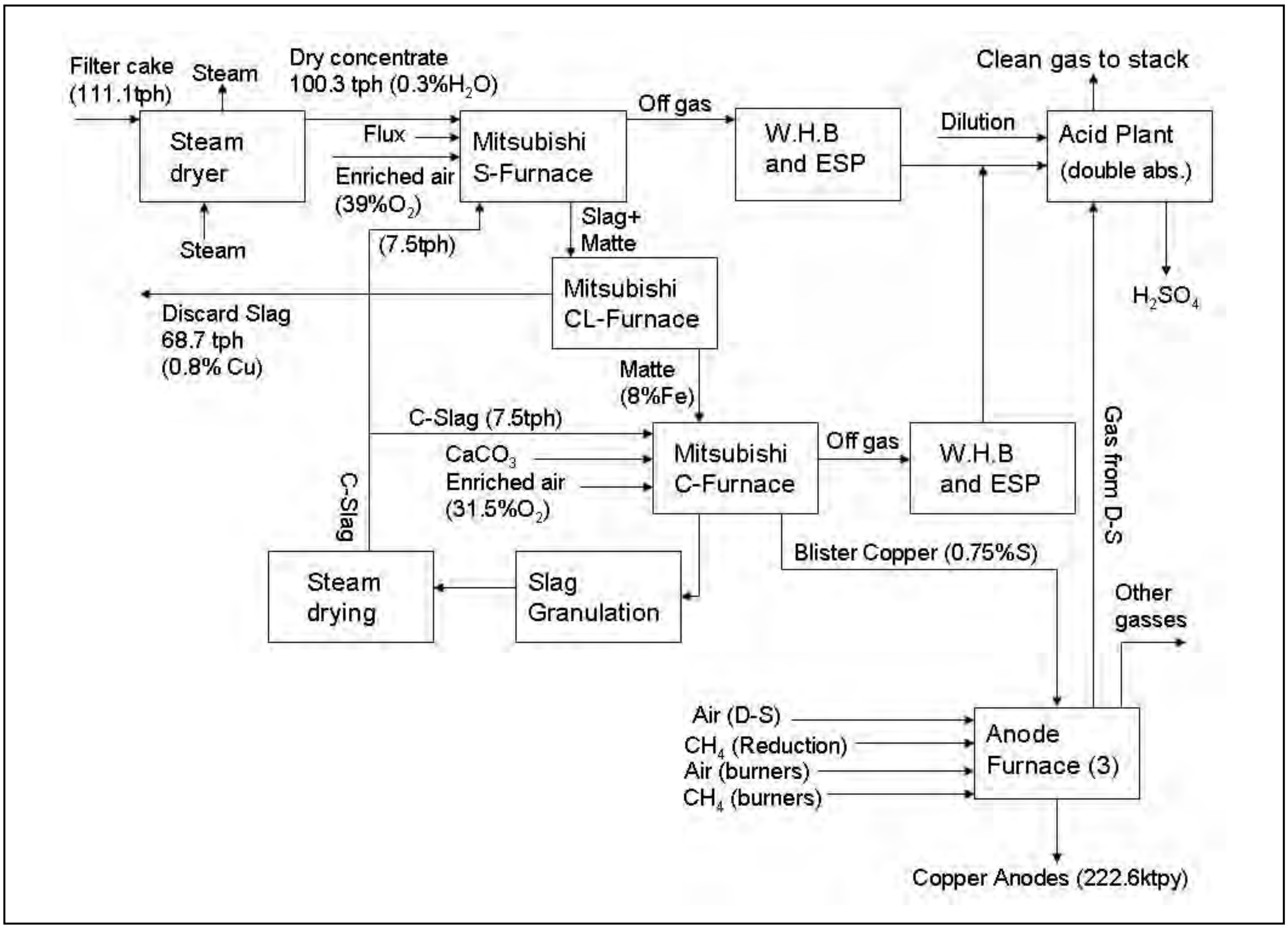

Figure 3: Mitsubishi Copper Continuous Smelting

3.4 Noranda/Teniente Continuous Bath Smelting + PS Converting

The Altonorte Smelter flowsheet [10] was used as a basis for developing the model for this processing route (see Figure 4 below).

The smelting unit is a Noranda Reactor-Teniente Converter hybrid. Most of the concentrate is injected dry through tuyeres into the molten bath. Wet concentrate, flux, coke and coarse internal smelter reverts are fed to the reactor via a Garr-Gun. Dust is recycled to the Reactor. The product matte (~74 % Cu) is converted in PS converters. The reactor slag is controlled cooled and milled for copper recovery, while solid converter slag is recycled to the reactor in the present case.

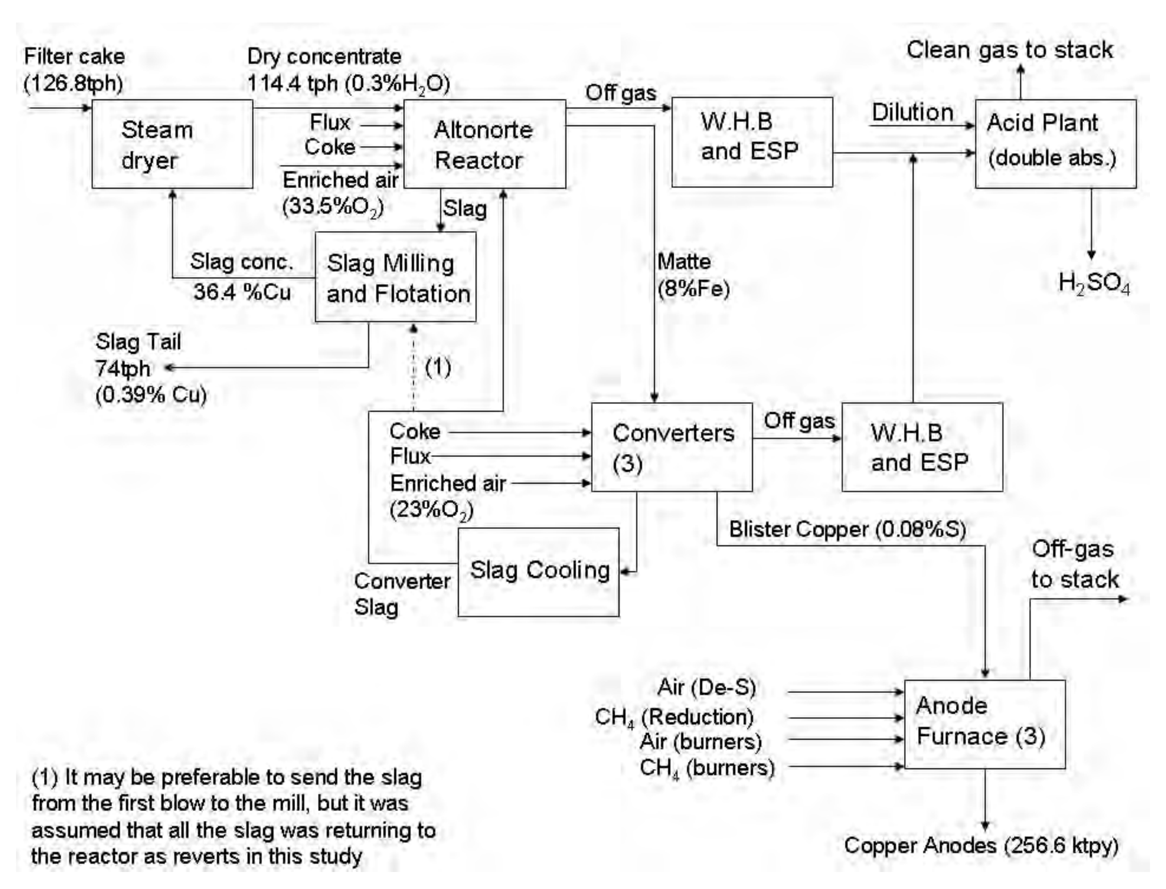

Figure 4: Noranda/Teniente Continuous Bath Smelting + Peirce-Smith Converting

4 Calculation of Process Energy Requirements

4.1 Smelter Throughput

As noted above, the four models developed in the present work used the initial model by Tripathi et al. [3] as a base. Consistent with the trend towards higher smelter capacity [4], high tonnage plants were considered in the present work as shown in Table 2 overleaf.

Table 2: Annual copper concentrate throughputs

Process Route	Technology	Copper concentrate throughput, million tonnes/year
1	Outokumpu	1.2
2	Isasmelt	1.2
3	Mitsubishi	0.75
4	Noranda/Teniente	0.85

Note: These tonnages are based on current proven industrial performance and adapted to handle this study "standard" concentrate (see Table 3); 85 % smelter on-stream time or 7446 h/year was also assumed. Specific furnace dimensions are found in the appropriate references.

4.2 Standard Conditions

The standard conditions that were used for the calculation of energy requirements for each of the chosen processes are presented in Table 3 below. These include the assay of a "standard" copper concentrate, and data on fluxes and fuels.

Table 3: Standard conditions used in the modified Metsim model

Item and data
Concentrate analysis (dry basis): 30.5% Cu, 28.5% Fe, 31.5% S, 5% SiO_2, 2% Al_2O_3
Concentrate moisture content: 10% H_2O (Bone-dry feed charged to Outokumpu, Mitsubishi and Noranda/Teniente furnaces; wet feed charged to Isasmelt furnace)
Flux analysis (dry basis): 88% SiO_2, 2% CaO, 6% Al_2O_3, 2% MgO and 2% Fe_3O_4
Flux moisture: 3% H_2O
Natural gas: 37.3 MJ/Nm^3
Coal: 28.4 MJ/Kg
Ambient conditions: 0^0C, 760 mm Hg

4.3 Peirce-Smith Converting

In this study, Peirce-Smith converting is used in two of the four flowsheets. It should be noted that in most smelters that practice flash smelting, the product matte is also treated in PS converters. The Metsim converter model developed here was based on standard converter practice, and for the tonnages used in the study a typical 4.6 m diameter by 13.4 m long vessel was considered. Unit energy factors used for converting Isasmelt matte are presented in Table 4 below. Data for the Noranda/Teniente matte were developed in a similar way.

Table 4: Energy Required for PS Converting (Isasmelt matte)

Item	Amount	Unit energy	Electrical power kWh/tonne Cu	Fossil fuel MJ/tonne Cu
Compressed air	1,419 Nm^3/t Cu	0.05 kWh/Nm^3	71.0	
Process off gas handling	3,520 Nm^3/t Cu	0.0085 kWh/Nm^3	29.9	
Fuel for pre-heating and standby	Estimated by authors			385.2
Flux preparation	0.19 t flux/t Cu	(90 MJ + 3 kWh)/t flux	0.6	17.1
Total before steam credit			101.5	402.3
Steam production	12.5 t/hr. It is noted that smelter and converter steam was used for drying, surplus smelter steam was used to produce electricity			

The above energy numbers compare well with those reported by Kellogg and Henderson in their paper [1].

4.4 Anode Refining and Casting

This is a common unit operation for the four process routes covered in this study. A nominal 250 tonne capacity anode furnace was used in this study. A complete furnace cycle consists of loading blister, oxidation, reduction and casting of anodes. The length of the oxidation time per tonne of blister produced by PS converting is shorter than for the higher sulphur-containing blister produced in both the Mitsubishi and the Kennecott Outokumpu converters. The length of the reduction time per tonne of copper was assumed to be the same for the four process routes. The energy requirements for refining low-sulfur blister and anode casting per tonne of anode are presented in Table 5 overleaf.

Table 5: Energy requirements for anode refining and casting

Item	Amount	Unit energy	Electric energy kWh/tonne of Cu	Fossil fuel MJ/tonne Cu
Furnace heating fuel (natural gas)	23.8 Nm^3/tonne of Cu	37.3 MJ/Nm^3		888
Natural gas for poling	6 Nm^3/tonne of Cu	37.3 MJ/Nm^3		224
Compressed air and misc energy	Estimated by authors		14.7	
Process off gas handling	1494 Nm^3/t Cu	0.0085 kWh/Nm^3	12.7	
Total			27.4	1112

4.5 Slag Cleaning

Milling is used for copper recovery from slags in two cases (Outokumpu flash smelting and Noranda/El Teniente bath smelting). An electric furnace (settler) is used for separating matte from slag in Mitsubishi continuous smelting, while a rotary holding furnace (RHF) is used in the Isasmelt flowsheet for similar purposes. Finally, a cylindrical-horizontal-tilting furnace is used for converter slag reduction with pig iron and copper recovery as matte in the latter process route. Unit energy factors for each of these operations are summarized in Table 6 below.

Table 6: Equivalent energy requirements for slag treatment per tonne of slag

Item	Slag milling	Converter Slag Cleaning (Isa)	Slag/Matte Settler	
			El. Fce. (Mitsubishi)	RHF (Isasmelt)
Electric energy	66 kWh		45 kWh	
Grinding media	0.78 kg steel			
Electrode carbon			0.5 kg	
Coke breeze			8 kg	
Natural gas (heating)		28.3 Nm^3		15.8 Nm^3
Pig iron		23 kg		
Miscellaneous power		6 kWh		6 kWh
Off gas handling		1197 Nm^3		666 Nm^3

Note: Data based on present model and additional information from Kellogg & Henderson [1]

4.6 Secondary and Fugitive Gases Handling

A modern copper smelter includes facilities for collecting and handling secondary and fugitive gases. Examples of recent installations can be found in papers by Toyo authors [11] and by Boliden authors [12]. Based in part on these references, the present authors estimated volumes of unit secondary and fugitive gases that are approximately linked to the number of furnaces employed in a process routes. These unit volumes are estimated at 23,000 Nm^3/tonne of copper for process routes (1) and (3), and 38,000 Nm^3/tonne of copper for process routes (2) and (4). The unit electrical power required to handle these gases is given in Table 1.

4.7 Process Operating Parameters

As noted, concentrate throughput and model operating parameters for each of the four processing routes were based on published information [4-10]. Private communication with industry was used to confirm or modify data. The values of these parameters were adapted for the standard concentrate used in this study. Due to space constraints, only key process parameters will be given in succeeding paragraphs. The reader is encouraged to check this paper references for further detail.

4.7.1Outokumpu Flash Smelting + Kennecott-Outokumpu Flash Converting

A flowsheet of this process is presented in Figure 1. Operating parameters are based on data reported in the 2003 Copper Smelter Survey [4] and D. George et al’s paper on the modernization of the Utah Smelter [5]. Following are the values of key parameters:

- Smelting: (a) Feed rate - 161 tonnes dry concentrate/h; (b) Matte - Grade 70% Cu, furnace temp 1324 ºC; (c) Slag - Fe/SiO_2 1.28, 2.5% Cu.
- Converting: (a) Blister - 0.2% S, furnace temp. 1270ºC; (b) Slag - 15% CaO, 22% Cu.

4.7.2 Isasmelt Smelting + RHF Matte/Slag Settler + PS Converting

The Isasmelt process flowsheet presented in Figure 2, as well as dimensions and capacities of the various furnaces, are based on published information on the modernization of the Ilo Smelter in Peru [6-8]. Private communication with smelter personnel assisted in further developing operating parameters [13]. Following are the values of key parameters:

- Smelting: (a) Feed rate - 161 tonnes dry concentrate/h; (b) Matte - Grade: 64% Cu, furnace temp. 1185 ºC; (c) Slag - Fe/SiO_2 wt ratio 1.4, 0.8% Cu (discharged from RHF).
- Converting: (a) Slag - Fe/SiO_2 wt ratio 1.5, 4.4% Cu.

4.7.3 Mitsubishi Continuous Smelting Process

The flowsheet of this process route is presented in Figure 3. Operating data for the four smelters currently practicing the Mitsubishi process are found in the 2003 Copper Smelter Survey [4]. In addition, the metallurgy and commercial practice of this process are fully discussed in a Mitsubishi monograph first published in 1987, and successively revised in 1998 and 2002 [9]. Following are the values of key parameters:

- Smelting: (a) Feed rate – 100.7 tonnes dry concentrate/h; (b) Matte - Grade 70.1% Cu, temp. 1200 ºC; (c) Slag - Fe/SiO_2 wt ratio 1.28, 0.8% Cu (discard).
- Converting: (a) Blister 0.75 %S, temp. 1250ºC; (b) Slag - 15%CaO, 15% Cu.

4.7.4 Noranda/Teniente Continuous Bath Smelting + PS Converting

As earlier discussed, the flowsheet presented in Figure 4 is based on the operation of the Altonorte Smelter in Chile. Features and dimensions of the reactor, a hybrid Noranda Reactor/Teniente Converter, and other key components of the smelter flowsheet are discussed elsewhere [10]. Operating data are presented in the 2003 Copper Smelter Survey [4]. In addition, two of the present authors have had opportunity to familiarize with the operation of the Altonorte Smelter. Following are the values of key parameters:

- Smelting: (a) Feed rate 114.2 tonnes dry concentrate/h; (b) Matte - Grade 70 %Cu, temp. 1240 ºC; (c) Slag - Fe/SiO_2 wt ratio 1.4, 4.5 % Cu.
- Converting: (a) Slag - Fe/SiO_2 wt ratio 1.5, 6.8% Cu.

5 Results

The results of model calculations for process routes 1 to 4 are presented respectively in Tables 7 to 10. Separate entrances for electric energy, expressed in kWh/tonne of anode copper, and fuel, expressed in MJ/tonne of anode copper, are provided in these tables. The fuel equivalent of electric energy has been calculated using a power plant efficiency of 38% [15, 16]. In the tables, numbers for items such as fuel, oxygen, compressed air, secondary and fugitive gases correspond to respectively overall smelter consumption or production.

Table 7: Energy requirements for producing anode copper from concentrate using Outokumpu Flash Smelting + Kennecott-Outokumpu Flash Converting (Process 1)

Item	Electric Energy/tonne anode		Fuel	Total
	KWh	Equivalent MJ	MJ/tonne anode	MJ/tonne anode
Fossil fuel			1,479	1,479
Oxygen	312	2,957		2,957
High pressure air	10	95		95
Low pressure blowing air	16.3	154		154
Burner air & misc.	3.7	35		35
Process gas handling	27	256		256
Sec/Fugitive gas handling	46	436		436
Matte/Slag granulation & hand.	53	504		504
Supplies (flux, etc.)	2.1	20	39	59
Slag milling	159	1,503		1,503
Acid production	340	3,216		3,216
Lighting & miscellaneous	30	284		284
Steam (power) credit	-20.5	-194		-194
TOTAL	979	9,266	1,518	10,784

Table 8: Energy requirements for producing anode copper from concentrate using Isasmelt Smelting/RHF Matte/Slag Settler + PS Converting (Process 2)

Item	Electric Energy/tonne anode		Fuel	Total
	KWh	Equivalent MJ	MJ/tonne anode	MJ/tonne anode
Fossil fuel			3,962	3,962
Oxygen	227	2,152		2,152
High pressure air	71	674		674
Low pressure blowing air	33.4	317		317
Burner air & miscellaneous	18.7	177		177
Process gas handling	74	702		702

Sec/Fugitive gas handling	76	720		720
Supplies (flux, steel, etc.)	1.8	17	213	230
Acid production	350	3,316		3,316
Lighting & miscellaneous	30	284		284
Steam credit	-154	-1,456		-1,456
TOTAL	729	6,903	4,175	11,078

Table 9: Energy requirements for producing anode copper from concentrate using Mitsubishi Continuous Smelting and Converting (Process 3)

Item	Electric Energy/tonne anode		Fuel	Total
	KWh	Equivalent MJ	MJ/tonne anode	MJ/tonne anode
Fossil fuel			2,431	2,431
Oxygen	220	2,081		2,081
High pressure air	10	95		95
Low pressure blowing air	77	727		727
Burner air & miscellaneous	7.9	75		75
Process gas handling	52	497		497
Sec/Fugitive gas handling	46	436		436
Slag granulation & handling	18.3	173		173
Supplies (flux, steel, electric furnace electrode, etc.)	22	208	67	276
Electric fce. matte/slag settler	103	972		972
Acid production	341	3,231		3,231
Lighting & miscellaneous	30	284		284
Steam (power) credit	-28.7	-272		-272
TOTAL	898	8,508	2,498	11,006

Table 10: Energy requirements for producing anode copper from concentrate using Noranda/El Teniente Bath Smelting + PS Converting (Process 4)

Item	Electric Energy/tonne anode		Fuel	Total
	KWh	Equivalent MJ	MJ/tonne anode	MJ/tonne anode
Fossil fuel			2,577	2,577
Oxygen	167	1,578		1,578

High pressure air	165	1,567		1,567
Burner air & misc.	6.7	64		64
Process gas handling	78	739		739
Sec/Fug gas handling	76	720		720
Supplies (flux, steel, etc.)	1.7	16	81	97
Slag milling	160	1,519		1,519
Acid production	414	3,922		3,922
Lighting & miscellaneous	30	284		284
Steam credit	-33.9	-321		-321
TOTAL	1,065	10,088	2,658	12,746

6 DISCUSSION

6.1 Energy Consumption in Copper Smelting

6.1.1 This Study vs. Kellogg and Henderson

The results of the calculated smelting energy requirements by process are summarized in Table 11 overleaf. Also shown in this table are energy requirements for selected processing routes covered in the Kellogg and Henderson paper [1]. Separate entrances for electric energy and fossil fuel are provided in Table 11 to facilitate the discussion.

Interestingly, the numbers in Table 11 show that three of the four processes investigated in this study have similar energy requirements. In fact, the difference in energy consumption between processes is within the margin of error of assumptions plus model calculations. There is, though, quite a variation in relative amounts of electric energy and fossil fuel requirements, with process 2 having the lowest electric energy consumption, but a fossil fuel requirement almost 3 times higher than process 1. The energy requirement of process 4, in particular the electric energy portion, is significantly higher than for any of the other three processes; process 4 has the largest consumption of high-pressure injection air and the highest acid making energy requirement. The data in tables 7-10 help to explain other differences between processes.

Table 11: Energy requirements for producing anode copper from concentrate (MJ/tonne of

anode) Processes 1-4 and selected Kellogg and Henderson’s processes [1]

Processing Route	Electric Energy	Fossil Fuel	Total
Process 1 (Flash-Flash)	9,266	1,518	10,784
Process 2 (Isasmelt)	6,903	4,175	11,078
Process 3 (Mitsubishi)	8,508	2,498	11,006
Process 4 (Noranda-Teniente)	10,088	2,657	12,746
KH-Hot Calcine Reverb	2,173	15,935	18,108
KH-Outokumpu Flash	7,477	6,760	14,237
KH-Mitsubishi	6,904	9,306	16,210
KH-Noranda	9,045	5,220	14,265

Notes on KH’s numbers:

- Power plant efficiency of 32.5% used to calculate the fuel equivalent of electric energy.
- WH recovered from all process gases. Steam raised in the WH boilers mostly used to generate power that it is then credited to the electric energy required.
- SO_2 recovered as acid from all process gases, except for reverb gas.
- Flash furnace operated with 30% O_2 enriched air. Matte grade 61.4% Cu.
- Noranda Reactor operated with 36% O_2 enriched air. Matte grade 75% Cu.

A comparison of the energy requirements of modern processes with those studied by Kellogg and Henderson [1] indicates that advances in technology and increased oxygen consumption have more than compensated for increased energy requirements to meet more stringent environmental standards. For instance, while in the Kellog and Henderson exercise process gases were treated in single-contact acid plant, double-contact technology was used in the present study. The handling of secondary and fugitive gases, not an issue in the late 1970s, was also incorporated in this investigation. On the other hand, a substantially lower energy requirement for producing tonnage oxygen (285 vs. 397 kWh/tonne) and a significantly higher power plant efficiency (38 vs. 32.5%) were used in this study, a reflection that in addition to copper smelting other relevant technologies have experienced substantial advances since the Kellogg and Henderson study was published. The latter factors explain that, despite the environmentally driven demands on modern smelter operation and a substantial increase in oxygen consumption, their average electric energy requirement (processes 1-3) is not substantially higher than for the three Kellogg and Henderson processes listed at the bottom in Table 14. What it is truly different is the much lower consumption of fossil fuel in modern smelters. Factors such as improved furnace design and operating practices, computerized process control, higher furnace feed throughput have certainly contributed to these achievements.

6.1.2Greenhouse Gas Emissions

As shown in Table 12 below, direct (heating fuel) and indirect (burning fuel for power generation) CO_2 emissions are highly dependant on the type of fuel used for each purpose [15, 16].

This study has shown that most of the energy requirement in modern copper smelting processes consists of electric energy. It is, then, the particular fuel mix used for the generation of power in regions with a large concentration of copper smelters that is currently determining the carbon footprint of copper sulphide smelting.

Table 12: Direct (heating fuel) and indirect (power generation) CO_2 emissions for various types of fuel.

Type of fuel	Direct CO_2 emission (tonne/MWh)	Indirect CO_2 emission* (tonne/MWh)
Natural gas	0.20	0.53
Light oil	0.26	0.68
Heavy oil	0.28	0.74
Anthracite	0.33	0.87
Lignite	0.40	1.05

- Assuming power plant generation efficiency of 38%.

Using these numbers and data presented in Tables 7-10, and assuming natural gas for both direct smelter heating and for electricity generation, the calculated average total CO_2 emissions for the four smelting processes studied here, is 0.63 tonnes of CO_2/tonne of anode copper. Direct heating accounts for 24% of these emissions and 76% is due to power generation

Detailed data on greenhouse gases emitted by the seven Chilean smelters have been recently published [17, 18]. The numbers are reported as CO_2 equivalent. Two of these smelters are connected to the Northern Interconnected Power Generating System (SING), and the other five to the Central Interconnected System (SIC). In 2007, 99.6% of SING energy was generated from fossil fuel (57.6% coal, 22.6% NG, 16.4%, 3% fuel oil), while SIC energy mix consisted of 47.1% thermal (less than one third of this was coal generated) and 52.9% hydro. Not surprisingly, the CO_2 equivalent emissions of SIC per MWh were about one third of the emissions of SING. Based on operating data for SING and SIC, Cochilco estimated that, in 2007, the average CO_2 equivalent per tonne of anode copper emitted by the Chilean smelters was 0.86 tonnes.

The Cochilco data illustrate the impact of the particular fuel mix used for power generation on greenhouse gas emissions per MWh.

6.1.3Other Data

Other data on copper smelting energy requirements have been published in recent years. These are presented in Table 14 below.

Table 14: Energy consumption in copper smelting (MJ/tonne ER cathode)

Reference	Electric Energy	Fossil Fuel	Total
Cochilco [17, 18]	5,090 (13,395)	3,990	9,080 (17,385)
Piret [15] (Smelting)	8,411	5,170	13,581
Piret [15] (Oxygen)	1,989		1,989
Marsden [21]	5,046 (13,279)	6,296	11,342 (19,575)

Note: Numbers in brackets correspond to the equivalent fuel of electric energy numbers reported in the respective references, using a 38% efficiency conversion factor.

Cochilco’s (Comisión Chilena del Cobre-Chilean Copper Commission) smelting energy data are taken from their comprehensive study on energy consumption in Chile’s primary copper mining industry [17, 18]. These are average numbers for the seven smelters currently operating in Chile. Four of them are equipped with Teniente Converters; one smelter operates a hybrid Noranda/Teniente Reactor; the flowsheet of another one combines an Outokumpu flash furnace and Teniente Converters; and one operates an Outokumpu flash furnace as the sole primary smelting unit. All of them use PSCs for processing high-grade matte to blister copper, but different slag cleaning technologies. The Cochilco numbers presented in Table 14 were slightly adjusted to account for copper losses in smelting. It should be noted that waste heat recovery from process gases is not normal practice in Chilean smelters. It should also be noted that five of these smelters get their electric energy from an interconnected power generation system with an important hydro component [17, 18].

Piret’s data [15] are based on a flash smelting operation with an annual copper production capacity of 150,000 tonnes. The electric energy number in Table 14 corresponds to his calculated fuel equivalent, also using 38% conversion efficiency. As shown in the table, Piret reports a separate energy consumption number for oxygen generation. Assuming that all of this oxygen is consumed in smelting, his total energy requirement for smelting increases to 15,570 MJ/tonne of copper.

Marsden investigated energy consumption in Freeport-McMoran primary copper operations in the USA, Chile and Peru [19]. These operations include various hydrometallurgical processing routes. His smelting number was based on the Miami Isasmelt Smelter operation. Apparently, in his paper, the electric energy number corresponds to direct conversion of kWh to MJ. On this assumption, a fuel equivalent, shown in brackets, was also included in Table 14.

No information on smelting various unit operations is given in the papers and reports published by the authors whose numbers are presented in Table 14. It is not possible, then, to comment on differences between their numbers and those calculated by the present authors. However, it should be noted that the energy consumption numbers developed in this study most probably correspond to ideal operations, and do not account for almost unavoidable real life inefficiencies. Factors such as heating of launders and ladles, short partial or total smelter shut downs that require maintaining furnaces hot, generation of solid reverts, etc. contribute to increasing heating fuel requirements above model calculations. In addition, neither Kellogg and Henderson nor the present authors consider smelter maintenance in their respective smelter models, an item that it is most probably included in industrial data. Nevertheless, the writers believe that results of studies such as the one they conducted contribute to better understanding processes and may assist in identifying unit operations that offer potential for improvement.

There are still some additional published data on energy consumption in copper smelting, such as those reported in a paper by Norgate and Rankin [20] on life cycle assessment of copper and nickel. However, their basic assumptions are much different from those used by the present authors.

7 Concluding Comment

Is there room for further energy savings in copper concentrate smelting? Recovering heat from a variety of combustion gases (anode furnaces, melt holding furnaces, etc.), a rare practice in smelters today, is an option. The corresponding decrease of greenhouse gas emissions may become an additional incentive for implementing these practices. New technological advances, such as the continuous copper fire refining process under investigation in Chile [21], also offer opportunities for energy savings. Possible new reductions in energy requirement for producing oxygen have been announced [22]. In summary, it would not be surprising to see even more energy efficient smelters in the near future.

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