

CSIRO
Minerals
Report

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**A Comparative Life Cycle
Assessment of Copper
Production Processes**

undertaken for

Intec Ltd

by T E Norgate

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by

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EXECUTIVE SUMMARY

A comparative Life Cycle Assessment (LCA) of both hydrometallurgical and pyrometallurgical processes for the production of refined copper metal has been carried out. The processes considered were:

- Intec
- CESL
- Dynatec
- Activox
- BacTech
- BioCOP
- Total pressure oxidation
- Flash smelting

The environmental impacts reported for each process were greenhouse gas emissions (ie. Global Warming Potential, GWP), acid rain gas emissions (ie. Acidification Potential, AP) along with the total (or full cycle) energy consumption.

The effect of a number of process parameters on the LCA results were also investigated, these being:

- concentrate grade
- black coal-based electricity, natural gas-based electricity and hydroelectricity

The LCA was based on publicly available data supplemented, in the case of the Intec process, by data provided by the company. All LCA inventory data, either published or provided, were accepted in good faith by CSIRO Minerals. Given that the various hydrometallurgical processes are at different stages of development, the amount of data available in the literature varied from process to process, and were not very detailed for any of the processes. This meant that a range of assumptions and approximations had to be made to establish the inventory inputs for each process, particularly for electricity and fuel. For this

reason it is emphasised that the LCA results presented in this report should not be considered as definitive results for the various processes considered, but rather as first estimates for the various processes at their reported stages of development. The relative rankings of the various processes may change as more detailed operating data become available or as the processes are developed further.

In an attempt to address the problem of product inequality between the different processes (copper dendrites for the Intec process, cathode copper sheets for the other processes), a product melting and casting step was also included in the LCA system boundary. The base case results for the various processes were compared both with and without this additional product melting and casting step included.

The LCA results showed that, as might be expected, the metal production and refining stage (whether it be pyrometallurgical or hydrometallurgical based) made the greatest contribution to the overall process environmental impacts, with the contributions of this stage towards the total energy consumption ranging from 56% (flash smelting) to 73% (Activox process) with the Intec process falling between these two at 61%. The contributions of this stage to the GWP and AP environmental impacts were of similar order to that for total energy.

For the metal production and refining stage, the Intec process had the lowest total energy consumption, GWP and AP of all the hydrometallurgical processes considered, being 51 MJ/kg Cu, 4.9 kg CO₂-e/kg Cu and 0.033 kg SO₂-e/kg Cu respectively, with the product melting and casting step included. Thus the Intec process was some 19-23% (total energy) and 22-27% (GWP) lower than the next best hydrometallurgical process (CESL), but some 4-8% (total energy) and 7-13% (GWP) greater than the corresponding values for the flash smelting process, depending on whether or not the product melting and casting step is included or not. The AP of the Intec process was the lowest of all the processes considered, including flash smelting. All hydrometallurgical processes are at a disadvantage in terms of total energy when compared on a life cycle basis with pyrometallurgical processes due to the relatively high electricity consumption of the electrowinning step and the related power plant generation inefficiencies.

While reducing the grade of concentrate produced in the mineral processing stage from 25% Cu (base case) to 20% and 15% Cu (accompanied by a corresponding increase in mineral processing copper recovery) resulted in a decrease in concentrator energy consumption (per tonne of copper) this was offset by an increase in the energy consumption of the metal production stage due to the extra amount of concentrate to be treated in this stage. The overall result was a slight increase in the total energy consumption of the Intec process (2% for a reduction in concentrate grade from 25% to 15% Cu), but for the other hydrometallurgical processes the increase in total energy ranged from about 7-17% while for flash smelting it was 23%, for the same change in concentrate

grade. The total energy consumption of the Intec process was about 11% lower than that for flash smelting at a concentrate grade of 15% Cu, and virtually the same as flash smelting at a 20% Cu concentrate grade. Thus the Intec process would appear to have an advantage over the other copper production processes at low concentrate grades due to its lower oxygen/air requirements relative to the other processes combined with the lower power requirement for the supply of air at atmospheric pressure rather than oxygen at pressure, as well as not requiring any grinding to ultrafine sizes. Furthermore, while flash smelting could in principle handle concentrates of such low grades, in practice this would probably be uneconomic.

Changing from black coal-based electricity to natural gas-based electricity at the same base case generation efficiency of 35% (Australian power grid average in 1995) reduced the GWP of all the processes considered, typically in the order of 30%. This is due to the lower greenhouse gas intensity of natural gas-based electricity compared to black coal-based electricity (0.57 t CO₂/MWh cf. 0.96 t CO₂/MWh). The APs of all the hydrometallurgical processes were reduced in the order of 70%, while for flash smelting it was reduced by about 30%. While improvements in power generation efficiency above the base case value of 35% were not considered for either black coal or natural gas-based electricity in the study, the use of hydroelectricity (with a generation efficiency of 80%) was included. As might be expected, the higher generation efficiency of hydroelectricity improved the standings of the hydrometallurgical processes relative to flash smelting in terms of total energy consumption. The extent of this improvement was such that the total energy consumption of the Intec process was less than that for flash smelting at the base case concentrate grade of 25% Cu.

In conclusion, based on data provided by Intec Ltd and that sourced from the literature, and bearing in mind the assumptions and approximations made in relation to these data, this LCA study has shown the Intec copper process to have the lowest environmental impact in terms of Global Warming Potential (GWP), Acidification Potential (AP) and total energy consumption of all the hydrometallurgical processes considered, but slightly higher than for flash smelting at the base case concentrate grade of 25% Cu. This advantage of the Intec process over other hydrometallurgical processes is extended further as the concentrate grade decreases and at very low concentrate grades (in the order of 15-20% Cu) even includes the flash smelting process.

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1 INTRODUCTION

The majority of copper sulphide concentrates produced worldwide are currently treated at smelter/refinery complexes. While the smelting process is well developed, it suffers from a number of disadvantages, including the high capital cost of new installations and environmental concerns with sulphur dioxide emissions. Despite these disadvantages, smelting is a relatively efficient process that has been made more so in recent years with improvements such as flash smelting and converting.

There have been many attempts to develop a hydrometallurgical process to replace copper smelting in the past, but with very little success. One of the main problems has been the refractory nature of the most common copper sulphide mineral, chalcopyrite. However in recent years a number of new hydrometallurgical processes for treating chalcopyrite with high extraction efficiencies have been developed. One of these is the Intec Copper Process developed by Intec Ltd. for the extraction of copper and associated precious metal by-products from copper sulphide concentrates, including chalcopyrite.

Development of the Intec Copper Process has proceeded from the pilot plant scale (0.05 tpd Cu) to the demonstration plant scale (1 tpd Cu). Prior to commercialising the process, Intec Ltd. has carried out both economic and environmental analyses of the process. Life Cycle Assessment (LCA) methodology was used for the latter analysis (CRESTA, 2000). The LCA was essentially "cradle-to-gate" in nature (ie from ore extraction through to metal production) with the environmental impact categories considered being:

- greenhouse effect
- acidification
- eco-toxicity
- human toxicity
- ozone layer depletion
- eutrophication
- smog

with the first four categories being subsequently found to have the most significant impact with regard to the Intec Copper process.

Intec Ltd. now seeks to have a comparative LCA carried out in which the Intec copper process is compared with a number of other competing process technologies. The initial intention was to include the first four of the above impact categories in the LCA. However concerns were raised about the ability to quantify the two toxicity categories for the competing processes. While CRESTA (2000) used the results from leachability tests on the solid residue from the Intec Copper process to give more realistic estimates of eco-toxicity and human toxicity for this process, it was anticipated that such data would not be forthcoming for any of the other processes. For this reason it was decided to exclude the two toxicity impact categories from the LCA at this point in time and focus on the greenhouse and acidification impact categories.

This report outlines the scope, data, assumptions used and results of the LCA of copper production processes carried out by CSIRO Minerals under the terms and conditions of CSIRO Minerals Quotation No. 32148 (2 August 2001). The contact person at Intec Ltd. for the project was Mr. David Sammut.

2 LIFE CYCLE ASSESSMENT

Life Cycle Assessment (LCA) is an analytical tool for quantifying the resource consumption and environmental impacts associated with a product, process or activity during its entire life cycle. LCA has also been referred to as “full fuel cycle analysis” or “cradle-to-grave” analysis. The objective of most LCA studies is to find the design option that minimizes the life cycle impact of the process.

The LCA methodology has four distinct stages:

- the goal definition and scoping stage, where the goals and scope of the proposed study are described and agreed upon with reference to the intended application;
- the inventory stage, where the material and energy inputs and outputs to and from the system are quantified;
- the impact assessment stage, where the results of the inventory analysis are interpreted in terms of the potential impacts they have on the environment
- the improvement assessment stage, where potential areas of improvement are identified

One of the most important issues at the goal definition and scoping stage is the definition of the system to be studied. This is a critical step in ensuring that the LCA is both manageable and meaningful. Attempts have been made to define “decision rules” which can be used to set system boundaries, although these still allow for considerable discretion. These rules are generally based on the contribution of the system components to total energy, total mass or “environmental relevance”. Decision rules based on mass contribution are probably the most commonly used for excluding ancillary materials; e.g., exclude materials that contribute less than 5% to the mass of a unit process or those that contribute less than 1% to the overall mass of the system. However, before excluding a material from the inventory it is advisable to repeat the exercise using energy as the criterion. It is also common practice in conducting LCAs to omit material and energy inputs associated with equipment manufacture and plant construction on the grounds that they are negligible. Another reason is that these inputs are quite difficult to calculate accurately and, furthermore, require an iterative approach; e.g., coal production requires inputs of steel and electricity but electricity production requires coal and steel. All inputs that are included in the LCA must be tracked back to naturally occurring materials.

Data availability is a major issue in conducting LCAs. Indeed it is the data collection associated with the inventory stage that is responsible for most of the time and cost of LCA. While the data preferably come from measurements on actual plants, such data are usually confidential to companies and very little is published in a form suitable for use in LCA. Alternatively, mass and energy balance outputs from process simulation models may be used as the source of inputs for LCAs. This is particularly the case for new process designs.

A problem that often arises at the inventory stage of an LCA is the use of allocation rules for estimating inventory data when there is more than one useful product. Simple allocation procedures may be based on mass, volume, energy content or economic value. The most common practice is to allocate on the basis of mass.

The inventory results are classified according to the kind of environmental problems to which they contribute during the impact assessment stage. Attempts are then made to quantify the contributions to each impact category. To do this equivalency factors are used. These factors indicate how much a substance contributes to an environmental impact compared to some reference substance. Some of the environmental impacts commonly considered are:

- global warming – measured relative to the effect of 1 kg of CO₂
- acidification - measured relative to the effect of 1 kg of SO₂
- photochemical oxidant formation - measured relative to the effect of 1 kg of ethylene
- nitrification - measured relative to the effect of 1 kg of phosphate
- resource depletion – measured relative to world reserves

Each inventory amount is multiplied by its corresponding equivalency factor and an aggregated score for each impact category is obtained. The results from this step are considered to be the environmental profile of the system. The aggregated scores for global warming and acidification are referred to as the Global Warming Potential (GWP) and Acidification Potential (AP) respectively. The GWP and AP per kg of metal produced were used to compare the various processes. The equivalency factors used in this study were:

Global Warming Potential	CO ₂	1
	CH ₄	21
	N ₂ O	310
	CF ₄	6500
	C ₂ F ₆	9200
Acidification Potential	SO ₂	1
	NO _x	0.7
	HCl	0.88
	HF	1.6

To assist with the data storage, retrieval and manipulation associated with LCA, a software program (LCA-PRO) has been developed by CSIRO Minerals using MS EXCEL™. This program allows the user to quickly build up an LCA workbook for a process, with separate worksheets for each process step. The emissions from one worksheet are carried over to the next until the final worksheet where the overall process emissions are calculated. The GWP and AP for the process is then calculated and the contributions of the various process steps to the overall result are reported. Program options include fuel type and efficiency for power generation, mode of transport and fuel calorific values. The LCA-PRO software was used in this study.

3 SCOPE OF STUDY

System boundary

The system boundary considered was from extraction of the ore from the ground through to the production of refined copper (ie. “cradle-to-gate”). The overall process was sub-divided into the processing stages listed below, with the first two stages being common to all of the copper production processes examined and the latter stage being split in two for the flash smelting process:

- mining (ie. ore production)
- mineral processing (ie. concentrate production)
- metal production and metal refining

However the refined metal product from the Intec process is in the form of copper dendrites while the products from the other processes are in the form of cathode copper sheets and hence the products are not all identical as required by LCA methodology.

While the Intec dendritic copper product could be sold without further treatment, it is more than likely that a melting and casting step would be required to give value-added product shapes. On the other hand, the cathode copper sheet product from the other processes (hydrometallurgical and pyrometallurgical) could be sold in unaltered form, but would still require melting at some stage. In an attempt to address this problem of product inequality, the LCA results for the various base cases are compared both with and without a product melting and casting step included.

Environmental impacts

The environmental impact categories reported for each process considered in the study were greenhouse and acidification gas emissions. These emissions are reported both on an individual gas basis and on an aggregated gas basis (Global Warming Potential (GWP) and Acidification Potential (AP) respectively). While not strictly an environmental impact, the total (or full-cycle) energy consumption was also reported for each process.

Functional unit

The functional unit for the study was 1 kg of refined copper metal, ie. GWP, AP and total energy are reported per kg of refined copper.

Processes considered

The eight process considered in the comparative LCA were:

- Intec process
- flash smelting
- CESL process
- Dynatec process
- BioCop process
- BacTech process
- Total pressure oxidation
- Activox process

A brief description of each process is given in the next section, and the various processes are referred to as listed above in the remainder of the report.

4 PROCESS DESCRIPTIONS

The hydrometallurgical treatment of copper concentrates can be accomplished in various media. The media of most interest are:

- chloride (or chloride/bromide) – Intec
- sulphate
 - Activox
 - Dynatec
 - Total Pressure Oxidation
 - BacTech) bioleaching
 - BioCOP)
- mixed sulphate/chloride - CESL

All of these processes involve sulphide being oxidised to sulphate and/or elemental sulphur. The formation of elemental sulphur instead of sulphate reduces the amount of oxygen consumed in the process. The amounts of oxygen consumed in the various hydrometallurgical processes were largely based on the results provided by Dreisinger (1998) supplemented by literature data where available.

These processes are all at different stages of development and the amount of data available in the literature varies from process to process, and any reported data were not very detailed for any of the processes. In the case of the Intec process, reasonably detailed operating data were provided by the company. Typical operating conditions for the seven hydrometallurgical processes considered in the study are compared in Table 1 and brief descriptions of the processes are given in the following sections together with

the flash smelting process. It was necessary to add a lime boil (except for the CESL process) and a cyanidation step (for precious metal recovery) to the competing hydrometallurgical processes to give products comparable to the Intec process as indicated in Table 1.

Table 1. Typical operating conditions for hydrometallurgical processes.

Process	Temperature (°C)	Pressure (kPa)	Air or oxygen	Autoclave leach	Lime boil	Cyanidation
Intec	85	101 (atm)	Air	No	No	No
CESL	150	1480	O ₂	Yes	No	Yes
Dynatec	150	1675	O ₂	Yes	Yes	Yes
Activox	100	1000	O ₂	Yes	Yes	Yes
Total press oxidation	220	3000	O ₂	Yes	Yes	Yes
BacTech	45-85	101 (atm)	Air	No	Yes	Yes
BioCop	45-85	101 (atm)	Air	No	Yes	Yes

4.1 Intec process

The Intec process essentially consists of the three sequential circuits of leaching, purification and electrowinning. A simplified flowsheet of the process is shown in Figure 1. The leach section is of three or four stage countercurrent configuration and operates at 85 °C and atmospheric pressure. Copper concentrate (after regrinding) is fed to stage one and lixiviant is fed to stage four. Gold is dissolved in leach stages three and four and is then recovered onto a carbon filter (cyanidation of the process residue is not required). The leach residue is washed in a filter press before being discharged from the circuit. Energy consumption data provided by Intec for this LCA study were based on a conservative assumption that four leach stages would be required.

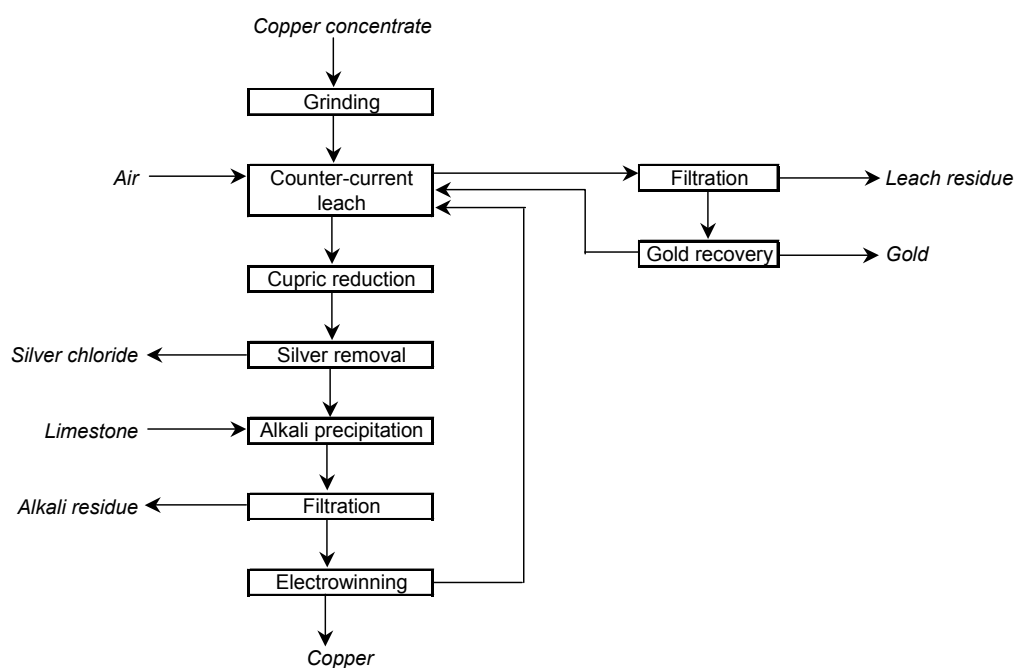


Figure 1. Schematic flowsheet of Intec process.

The pregnant electrolyte from the leach circuit is purified in a three stage system prior to electrowinning. The first stage involves the reduction of any residual cupric ions to cuprous while the second stage involves the addition of soluble mercury and copper dendrites (both recycled) to remove the silver as an amalgam which is subsequently treated to produce a silver product. In the third stage of purification, impurities such as iron, indium and bismuth are precipitated by the addition of limestone.

Electrowinning consists of the recovery of copper metal in the form of copper dendrites from an electrolyte containing cuprous (Cu^+) ions in a unique electrowinning cell. This is accompanied by the simultaneous regeneration of the leach liquor (lixiviant). The lixiviant contains the bromo-chloride ion (BrCl_2^-) and is referred to by Intec Ltd as Halex.TM The copper dendrites are washed and dried under inert atmosphere, then melted in a purged induction furnace and cast into ingots or directly to shaped products such as billet.

4.2 CESL process

This process is being developed and commercialised by Cominco Engineering Services (CESL), a wholly owned subsidiary of Cominco and has been tested at pilot plant (36 kg Cu/d) and demonstration plant (1-1.5 t Cu/d) scale. The process involves a two-stage leaching operation as shown in the schematic flowsheet in Figure 2.

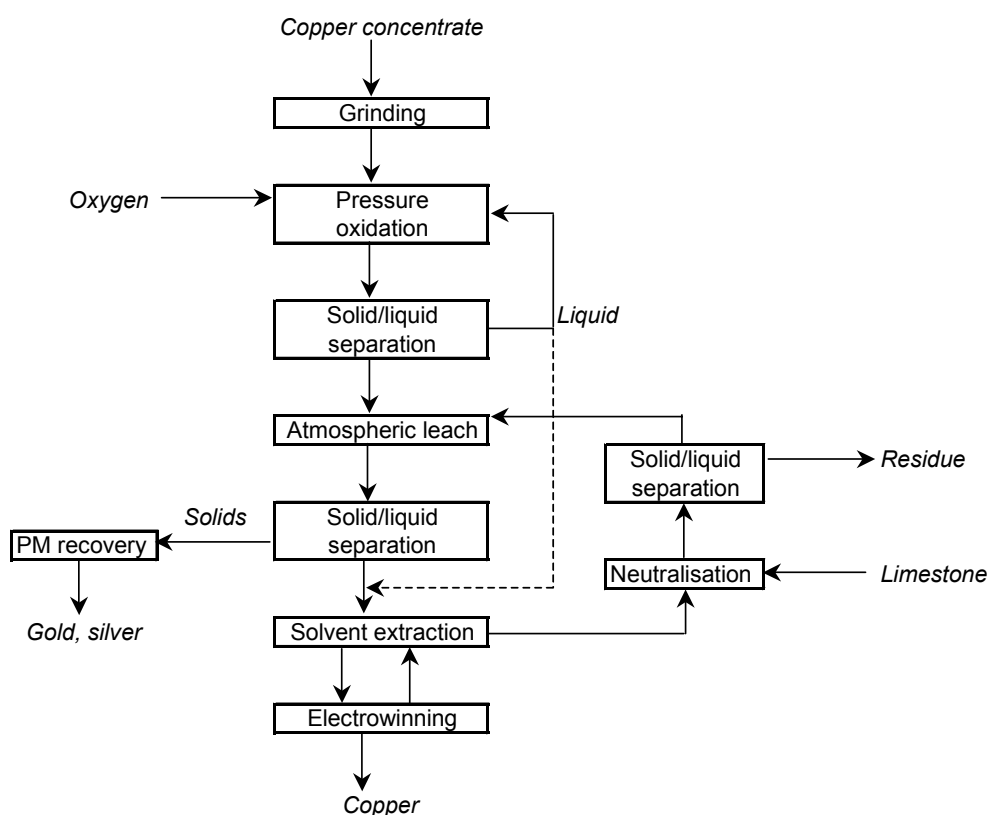


Figure 2. Schematic flowsheet of CESL process.

In the first stage reground copper concentrate is leached in an autoclave using high pressure oxygen and a mixed sulphate/chloride lixiviant. Leach conditions are typically 150 °C and 1480 kPa. The pressure oxidation converts all the copper sulphides to acid-soluble basic copper sulphate and this is followed by filtration.

The filter cake from pressure oxidation is leached at atmospheric pressure in dilute acid derived from the solvent extraction circuit to dissolve the basic copper sulphate. The atmospheric leach slurry then undergoes solid/liquid separation using a countercurrent decantation circuit (CCD). The separated solid (leach residue) goes to precious metal recovery, while the copper rich solution proceeds to solvent extraction followed by conventional electrowinning in sulphate media. The precious metals recovery flowsheet given by Jones (1999) does not include a lime boil pre-treatment prior to cyanidation so it was also omitted from the LCA study (see Table 1). Evaporation of some water takes place to maintain a water balance in the pressure oxidation.

4.3 Dynatec process

This process is being developed and commercialised by the Metallurgical Technologies Division of Dynatec Corporation and has been tested at pilot plant and miniplant (2-5 kg Cu/h) scale. A schematic flowsheet of this process is shown in Figure 3. Copper concentrate and recycled sulphide cake solids are reground and treated at elevated temperature and pressure (150 °C, 1675 kPa) in an oxidation pressure leach to solubilize the copper.

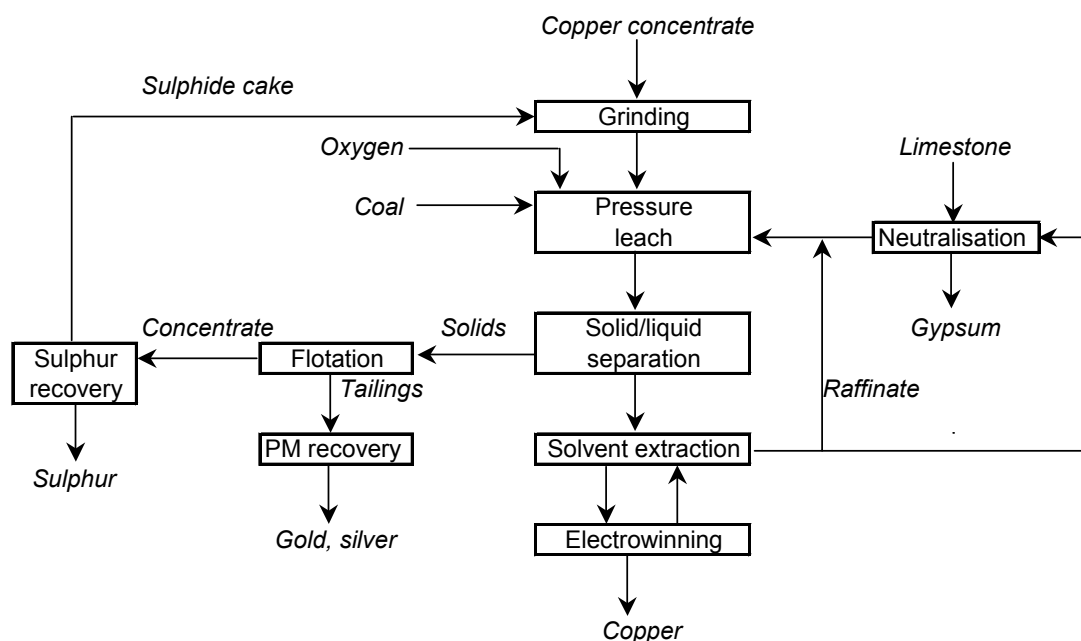


Figure 3. Schematic flowsheet of Dynatec process.

Coal is added as a surfactant to allow the removal of liquid elemental sulphur from the surfaces of unreacted sulphides. The pressure leach discharge slurry then undergoes solid/liquid separation and the pregnant solution proceeds to solvent extraction followed by electrowinning.

The separated leach residue undergoes flotation to produce a sulphur-sulphide concentrate and an oxidic tailings. The sulphur concentrate proceeds to sulphur recovery (melting and filtration) and the sulphide filter cake is recycled to the pressure leach. The tailings proceed to precious metal recovery where gold and silver are recovered by cyanidation preceded by a lime boil to improve silver recovery.

Buban and Collins (1997) indicate that it may be possible to recover copper from solution by direct electrowinning (ie. no solvent extraction) but also suggest that an alternative approach would include solvent extraction. Dreisinger (1998) is of the view that solvent extraction followed by electrowinning will be required for the Dynatec process, and was also assumed for this study. A large portion of the solvent extraction raffinate is recycled to the pressure leach. A raffinate bleed stream is treated with limestone to neutralize the acid.

4.4 Activox process

This process is being developed and commercialised by Western Minerals Technology Pty. Ltd. and has been tested up to demonstration plant (up to 50 kg concentrate/h) scale. A schematic flowsheet of this process is shown in Figure 4.

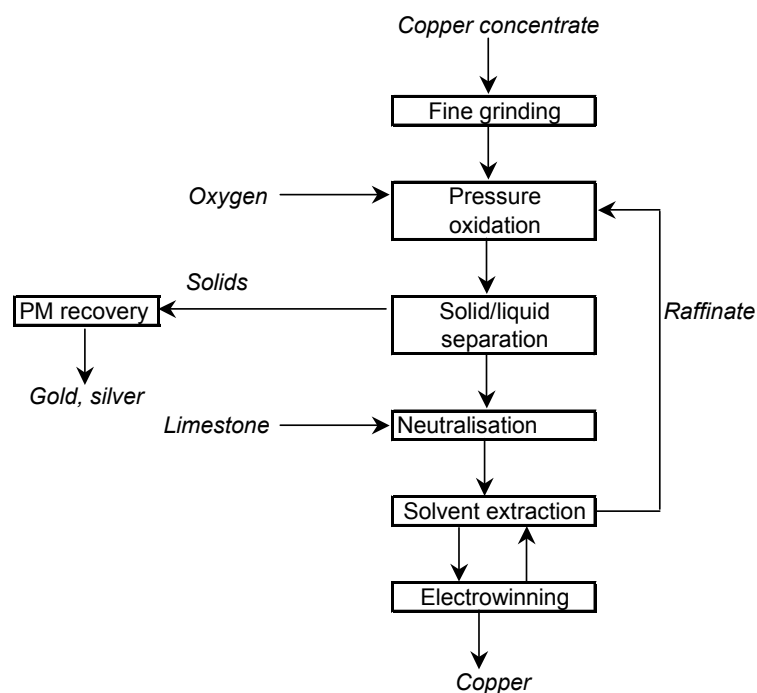


Figure 4. Schematic flowsheet of Activox process.

The first step in the Activox process is regrinding of the concentrate (typically in a vertical stirred mill) to a very fine size (P_{80} of 7-15 μm) to make it amenable to leaching at the temperatures used here (typically 100 °C). This is followed by autoclave leaching (at about 1000 kPa) with recycled raffinate as the leach medium and with oxygen addition.

The leach residue is separated from the slurry and washed. Gold and silver recovery takes place by cyanidation of the residue (with a lime boil pre-treatment to improve silver recovery). Excess free acid in solution is neutralized by limestone prior to solvent extraction, which is then followed by electrowinning.

4.5 Total pressure oxidation process

This process involves the complete (or total) oxidation of reground copper concentrate in an autoclave using oxygen at 220 °C and an oxygen overpressure of 690 kPa (which when added to the saturated steam pressure at 220 °C of 2320 kPa, corresponds to an absolute pressure in the vicinity of 3000 kPa) as shown in the schematic flowsheet in Figure 5. The autoclave slurry is “let down” to atmospheric pressure and then undergoes solid/liquid separation in a countercurrent decantation circuit (CCD) where the residue is washed to recover leached copper to the product solution. The recovery of copper from the weakly acidic solution is by conventional solvent extraction and electrowinning.

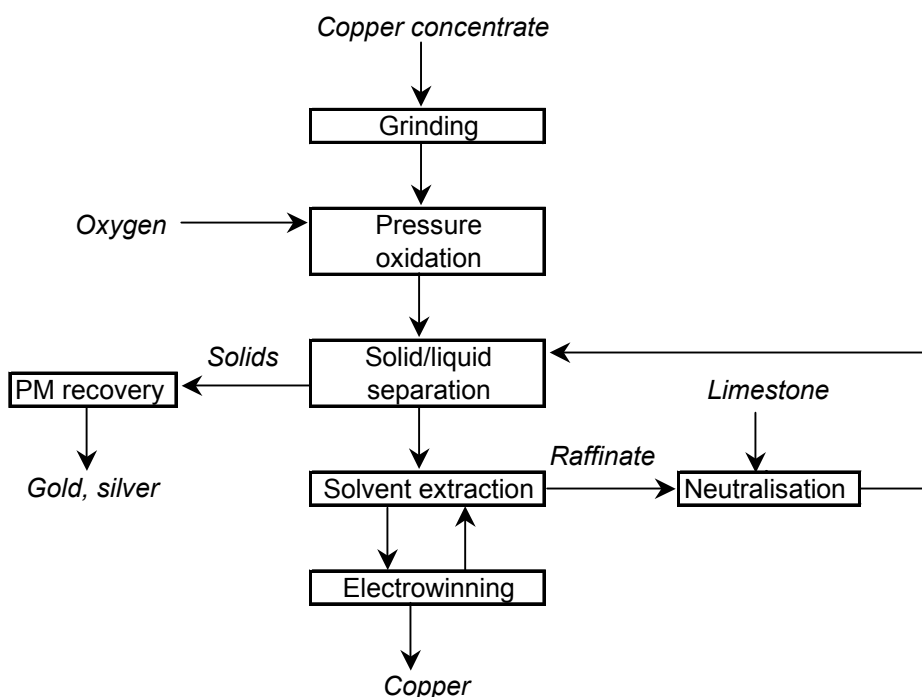


Figure 5. Schematic flowsheet of Total Pressure Oxidation process.

The washed leach residue goes to precious metals recovery (lime boil and cyanidation). The raffinate is neutralized with limestone prior to being recycled

to the CCD wash circuit. While the high temperatures and the use of oxygen for complete sulphur oxidation in this process is costly, Dreisinger (1998) suggests that this total pressure oxidation treatment may be desirable in selected circumstances.

4.6 BacTech process

This process is being developed and commercialised by BacTech and its partner Mintek (the South African national research organization) and is one of two bioleaching processes (the other being the BioCOP process – see next section) included in the study. Bioleaching of minerals (also known as bacterial oxidation or bio-oxidation) is carried out by naturally occurring micro-organisms. The micro-organisms which are employed in the BacTech process are moderate thermophiles (45-60 °C) or extreme thermophiles (60-85 °C). The thermophile cultures developed by BacTech were originally isolated from a mine in Western Australia. Commissioning of a 0.5 t concentrate/d demonstration plant is currently being completed at Industrias Penoles research facility in Monterrey, Mexico.

The copper concentrate first undergoes fine grinding (P_{80} of 5-15 μm) and then undergoes bioleaching in agitated aerated tanks as shown in the schematic flowsheet in Figure 6,

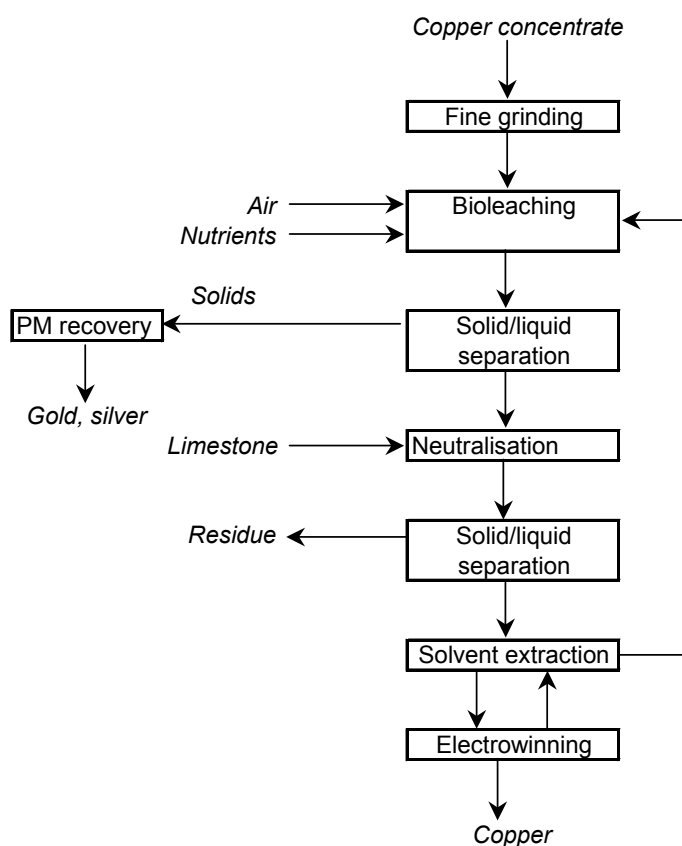


Figure 6. Schematic flowsheet of BacTech process.

This involves feeding a continuous stream of slurried concentrate into primary reactors containing a suspension of bacteria in a mildly acid environment. Most of the leaching occurs in these primary reactors. As concentrate is added to the primary reactors, partially oxidised material flows into secondary stage reactors where the final oxidation occurs.

The leached material then flows from the secondary reactors into thickening tanks for solid/liquid separation, which is usually carried out in a countercurrent decantation (CCD) circuit. The washed solid then goes to precious metal recovery. The bioleach solution is neutralized with limestone and the solids are removed by solid/liquid separation and the neutralized solution goes to solvent extraction and electrowinning. Raffinate is recycled to the bioleach reactors.

4.7 BioCOP process

The BioCOP process is currently being developed and commercialised by BHP/Billiton and had its origins in the tank bioleaching of refractory gold bearing sulphide concentrates (the BIOX process) pioneered by the South African company Gencor. A pilot plant (70 kg/d) has been operating as a joint development between Billiton and Codelco at the latter's Chuquicamata operation in Chile since the end of 1997.

The BioCOP process does essentially the same thing as the BacTech process but utilises a different strain of bacteria and therefore the BioCOP generic flowsheet shown in Figure 7 is the same as that for the BacTech process in Figure 6. However specific process flowsheets will vary, dependent on the concentrate mix and site circumstances. This is true of all the hydrometallurgical processes considered.

The BioCOP process generally involves the following steps:

- biological oxidation (using air) of the copper sulphide concentrate in stirred tank reactors at about 45 °C (mesophiles) or 65-85 °C (thermophiles) and 10-20% solids to produce a slurry containing soluble copper sulphate
- solid/liquid separation of the slurry to produce a copper rich solution
- solvent extraction of the copper from solution for subsequent electrowinning
- recycling at least part of the raffinate from solvent extraction to the bioleaching stage
- precious metal recovery from the washed bioleach solids

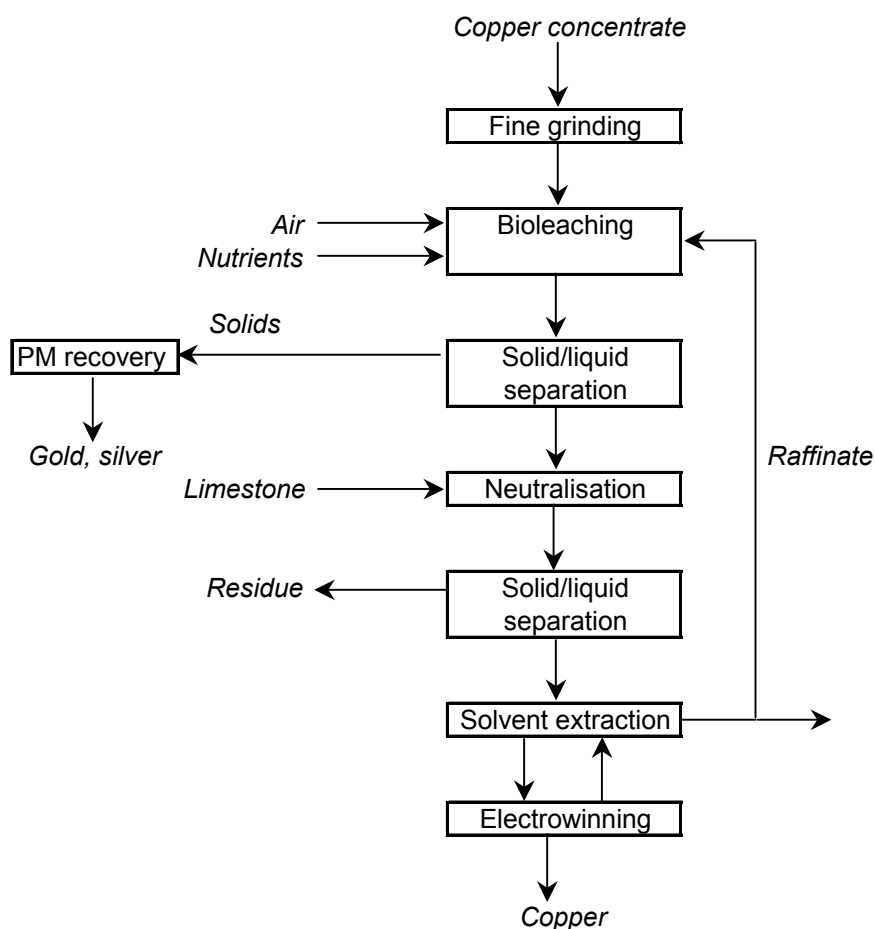


Figure 7. Schematic flowsheet of BioCOP process.

4.8 Flash smelting process

A schematic flowsheet of the flash smelting process is shown in Figure 8. Copper concentrate is first dried and then fed into the flash smelting furnace (FSF) together with recirculated slag concentrate and flux material. The offgas from the FSF goes to a waste heat boiler (WHB) where the temperature of the gas is reduced from about 1300 °C to 350-400 °C with the recuperated energy being used to generate steam (60-70 bar). After the WHB the gas is cleaned in electrostatic precipitators and a wet scrubbing section before going to the acid plant.

The matte produced in the FSF (typically containing 60-65% Cu) is ground and fed to the converting furnace along with flux material. Oxygen-enriched air is used in this furnace as well as the FSF. The offgas from the converting furnace is treated in a similar manner to that from the FSF. The slag from the converting furnace, together with that from the FSF, goes to the slag concentrator and the slag concentrate produced is recirculated to the FSF.

The blister copper from the converting furnace is fed into an anode furnace where the sulphur is oxidised in a short oxidation period and finally the oxygen

in the anode copper is reduced using natural gas before casting into anodes. The anode copper then undergoes electrorefining to produce cathode copper.

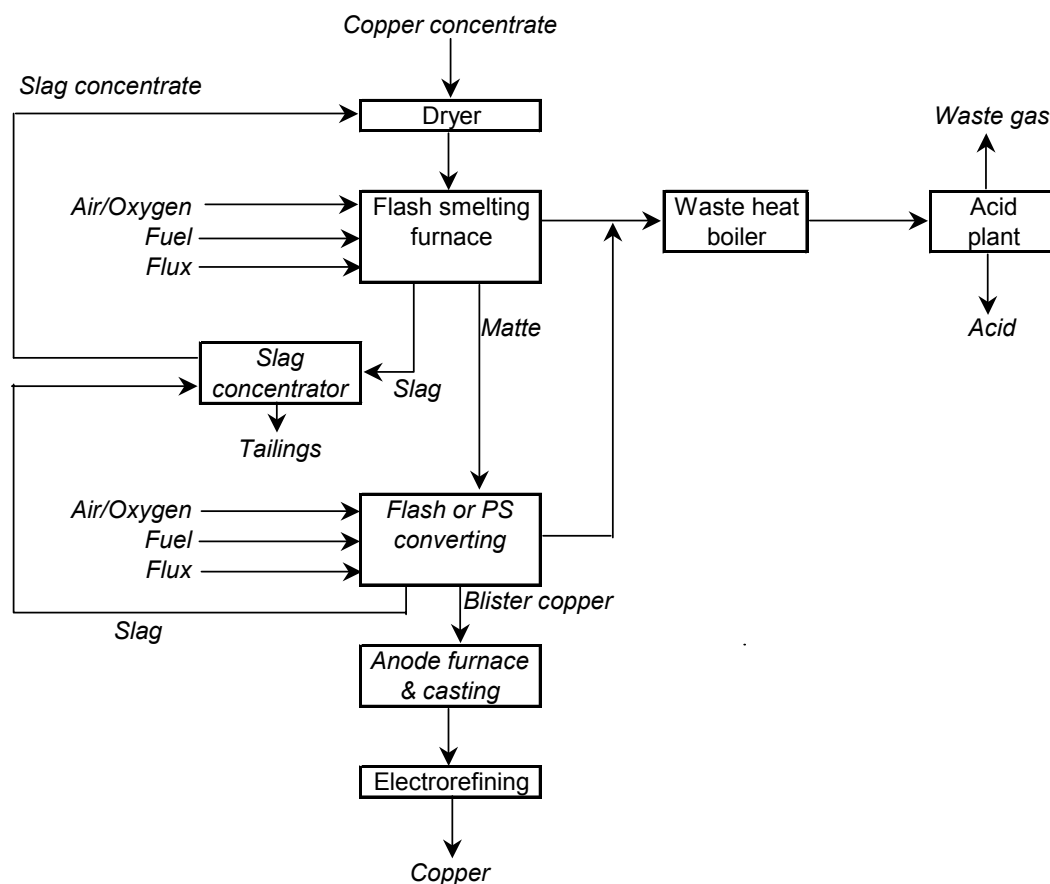


Figure 8. Schematic flowsheet of flash smelting process.

5. ASSUMPTIONS AND INVENTORY

The data used for input to each process model were cross-checked with more than one source where possible. These data and their sources are summarised in Table 2.

The following assumptions were made in carrying out the study:

- the mine and concentrator are in close proximity to each other and hence transport of ore between the two facilities can be ignored
- the hydrometallurgical processing plant is located at or near the minesite and hence transport of concentrate between the two facilities can be ignored
- given that roughly half the world's copper production occurs in integrated smelters (Brook Hunt, 2000) that presumably involves minimal transport of concentrate, while the other half is traded on the custom market and will involve some transport of concentrate (which will be initially most likely by

road or rail and then possibly by ship), a somewhat arbitrary but conservative compromise on concentrate transport distance of 500 km by rail was assumed for the flash smelting process – the sensitivity of the LCA results to this assumption is considered later in Section 7.1

- electricity is produced using black coal as the power plant fuel source
- power plant generation efficiency is 35%
- any neutralization of acid in the hydrometallurgical processes is done with limestone, not by any possible acid-consuming gangue material (ie. the acid is not used as a by-product reagent)
- the base case copper concentrate composition used in the study is:
25% Cu
30% S
0.01% Ag
0.001% Au
- recovery of copper in the concentrator of the mineral processing stage is 93.7% for the base case concentrate grade above (MIM (1998); Vonk (1993))
- while silver recovery may not be included in the preferred flowsheet by some of the companies developing hydrometallurgical processes, it has been included here for all processes to maintain comparability between the processes
- a lime boil pretreatment is necessary to achieve high silver recovery by cyanidation (Collins et. al. (2000))
- the metal recoveries assumed for the various processes for the base case concentrate are:

Table 3. Process metal recoveries.

Process	Recovery (%)			Reference
	Cu	Ag	Au	
Intec	98	93	90	Intec website
CESL	97	90	94	Jones (1999)
Dynatec	99	87	96	Barta et al (1999)
Activox	98	88 *	95	Evans (1999)
Total pressure oxidation	98	88 *	93	King & Dreisinger (1995)
BacTech	97	81	90	Miller (1999)
BioCOP	97	88 *	94	Dew et al (2000)
Flash smelting/converting	95	-	-	Rio Tinto (2000), Rich (1993)

* assumed equal to mean of reported values

- all bioleaching processes require fine grinding of the concentrate feed (comprised largely of chalcopyrite) in order to achieve satisfactory copper recovery (Dreisinger (1998))
- recovery of SO₂ in the smelter acid plant is 99%

Table 2. Process LCA inventory data used in study.

Stage	Process	Inventory	Reference	
Mining	Diesel fuel	0.002 t/ ore	Molinia (1993), Westcott & Hall (1993)	
	Electricity	13 kWh/t ore	Molinia (1993)	
	Mine drainage	0.78 t/ ore	Leahy (1993)	
Mineral Processing Metal Production	Waste rock	0.18 t/ ore	Leahy (1993)	
	Electricity	31 kWh/t ore	Various sources	
	Grinding media	0.001 t/ ore	Watters (1993)	
	Limestone	160 kg/t Cu	Houllis (2001)	
	Lime	133 kg/t S in conc	Houllis (2001)	
	Hydrochloric acid	20 kg/t Cu	Houllis (2001)	
	Sulphuric acid	10 kg/t Cu	Houllis (2001)	
	Nitrogen	10 kg/t Cu	Houllis (2001)	
	Carbon	20 kg/t Cu	Houllis (2001)	
	Sodium chloride	0.75 kg/t Cu	Houllis (2001)	
	Sodium bromide	20 kg/t Cu	Houllis (2001)	
	Air	2 kg/t Cu	Houllis (2001)	
	Natural gas	3340 kg/t S in conc	Dreisinger (1998)	
	Electricity	73 ¹ kg/t Cu	Houllis (2001)	
		535 kWh/t Cu	Houllis (2001)	
CESL process	Electricity	1650 kWh/t Cu	Houllis (2001)	
	Limestone	320 kg/t S in conc	Dreisinger (1998)	
	Oxygen	740 kg/t S in conc	Dreisinger (1998), Jones & Hestrin (1998), Jones (1999)	
	Sulphuric acid	33 kg/t conc	Dreisinger (1998), Jones (1996)	
	Hydrochloric acid	1 kg/t conc	Dreisinger (1998);	
	Natural gas	5 ¹ kg/t Cu	Jones (1999)	
	Electricity	900 kWh/t Cu	Jones (1999), Dreisinger (1998), Jones (1996)	
	PM recovery	Lime	61 kg/t conc	Mean of Dynatech and Total Pressure Oxidation processes
		Sodium cyanide	9 kg/t conc	Mean of Dynatech, Bactech, Activox and Total Pressure Oxidation processes
		Natural gas	42 ¹ kg/t Cu	Jones (1999), Houllis (2000)
		Electricity	200 kWh/t Cu	Jones (1999, 2001)
		Electricity	200 kWh/t Cu	Jones (2001)
	EW	Electricity	2100 kWh/t Cu	Jenkins et al (1999)

Activox process	Process	Limestone	240	kg/t S in conc kg/t S in conc kg/t conc kg/t Cu kWh/t Cu	Dreisinger (1998) Dreisinger (1998)
		Oxygen	670		
		Sulphuric acid	0		
		Natural gas	0 ¹		
		Electricity	1900		
		<i>PM recovery</i>			
SX	SX	Lime	61	kg/t conc kg/t conc kg/t Cu kWh/t Cu	Mean of Dynatech and Total Pressure Oxidation processes Corrans et. al. (1995), Dreisinger (1998) Jones (1999), Houllis (2000) Jones (1999, 2001)
		Sodium cyanide	15		
		Natural gas	93 ¹		
		Electricity	200		
		Electricity	200		
		Electricity	2100		
BacTech process	Process	Limestone	2390	kg/t S in conc kg/t S in conc kg/t conc kg/t Cu kWh/t Cu	Dreisinger (1998) Dreisinger (1998)
		Air	16360		
		Sulphuric acid	0		
		Natural gas	0 ¹		
		Electricity	1435		
		<i>PM recovery</i>			
SX	SX	Lime	61	kg/t conc kg/t conc kg/t Cu kWh/t Cu	Mean of Dynatech and Total Pressure Oxidation processes Miller (1999) Jones (1999), Houllis (2000) Jones (1999, 2001)
		Sodium cyanide	10		
		Natural gas	93 ¹		
		Electricity	200		
		Electricity	200		
		Electricity	2100		
Dynatech process	Process	Limestone	1180	kg/t S in conc kg/t S in conc kg/t conc kg/t conc kg/t Cu kWh/t Cu	Dreisinger (1998) Dreisinger (1998) Buban & Collins (1997)
		Oxygen	1230		
		Coal	25		
		Sulphuric acid	0		
		Natural gas	0 ¹		
		Electricity	1088		
<i>PM recovery</i>	<i>PM recovery</i>	Lime	45	kg/t conc kg/t conc kg/t Cu kWh/t Cu	Dreisinger (1998), Houllis (2000) Collins et. al. (2000), Barta et. al. (1999) Collins et. al. (2000), Barta et. al. (1999) Jones (1999), Houllis (2000) Jones (1999, 2001)
		Sodium cyanide	4		
		Natural gas	93 ¹		
		Electricity	200		
		Electricity	200		
		Electricity	2100		

Total pressure oxidation process	Process	Limestone Oxygen Sulphuric acid Natural gas Electricity	3420 2200 0 0 ¹ 1675	kg/t S in conc kg/t S in conc kg/t conc kg/t Cu kWh/t Cu	King & Dreisinger (1995) Dreisinger (1999), Dreisinger (1998), King & Dreisinger (1995) Dreisinger (1998), Houllis (2000)
	PM recovery	Lime Sodium cyanide Natural gas Electricity	77 6 93 ¹ 200	kg/t conc kg/t conc kg/t Cu kWh/t Cu	Dreisinger & Saito (1999), King & Dreisinger (1995) King et. al. (1993), King & Dreisinger (1995) Jones (1999), Houllis (2000) Jones (1999, 2001)
	SX EW	Electricity Electricity	200 2100	kWh/t Cu kWh/t Cu	Jones (2001) Jenkins et al (1999)
BioCOP process	Process	Limestone Air Sulphuric acid Natural gas Electricity	2390 16360 0 0 ¹ 1667	kg/t S in conc kg/t S in conc kg/t conc kg/t Cu kWh/t Cu	Dreisinger (1998) Dreisinger (1998) Dreisinger (1998)
	PM recovery	Lime Sodium cyanide Natural gas Electricity	61 31 93 ¹ 200	kg/t conc kg/t conc kg/t Cu kWh/t Cu	Mean of Dynatech and Total Pressure Oxidation processes Loayza et al (1996) Jones (1999), Houllis (2000) Jones (1999, 2001)
	SX EW	Electricity Electricity	200 2100	kWh/t Cu kWh/t Cu	Jones (2001) Jenkins et al (1999)
Flash smelting	Smelting & converting	Natural gas Coal Oil Electricity Oxygen Silica Limestone	24 17 7 1143 ² 834 130 21	kg/t conc kg/t conc kg/t conc kWh/t Cu kg/t Cu kg/t conc kg/t conc	Kojo & Hanniala (2001), Houllis (2001), Krag (1993) Ishikawa (1998), Lee et al (1999) Kojo & Hanniala (2001), Gonzalez & Ruiz (1999) Kojo & Hanniala (2001), Houllis (2001) Various sources Various sources
	Refining	Electricity Steam	323 230	kWh/t Cu kg/t Cu	Biswas & Davenport (1994), Houllis (2001) Houllis (2001), Norgate & Rankin (2000)

Notes: 1. Based on 53 MJ/kg for natural gas calorific value

2. Includes acid and oxygen plant

- melting and casting of refined copper products consumes 2000 MJ/t Cu of thermal energy (excluding the Intec process which uses electrical energy for melting) and 25 kWh/t electrical energy respectively (see Appendix 1)

6 RESULTS

6.1 Base case

The results for each of the eight process base cases are given in Tables 4 and 5, with the product melting and casting step excluded and included respectively. The results are also compared graphically in Figures 9-14.

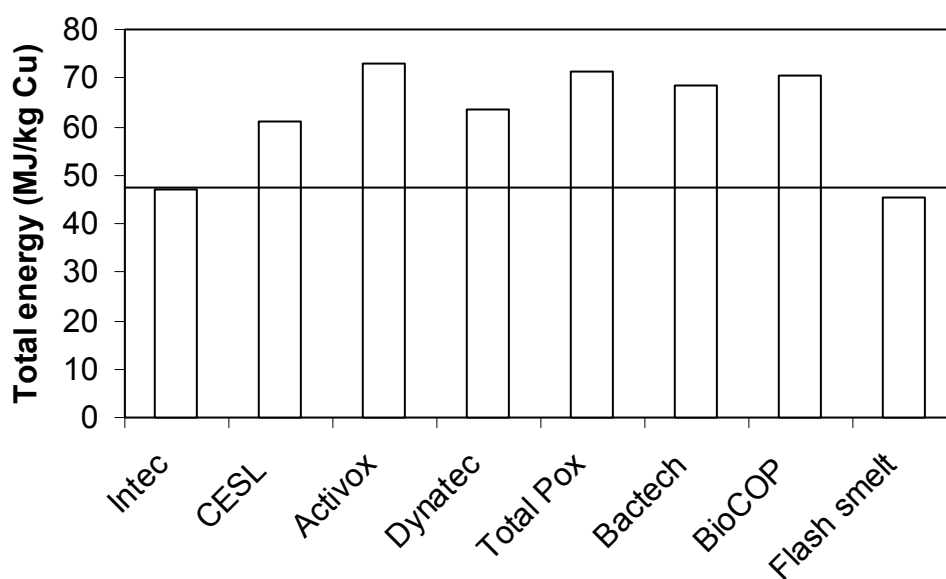


Figure 9. Total energy consumption (without melting and casting).

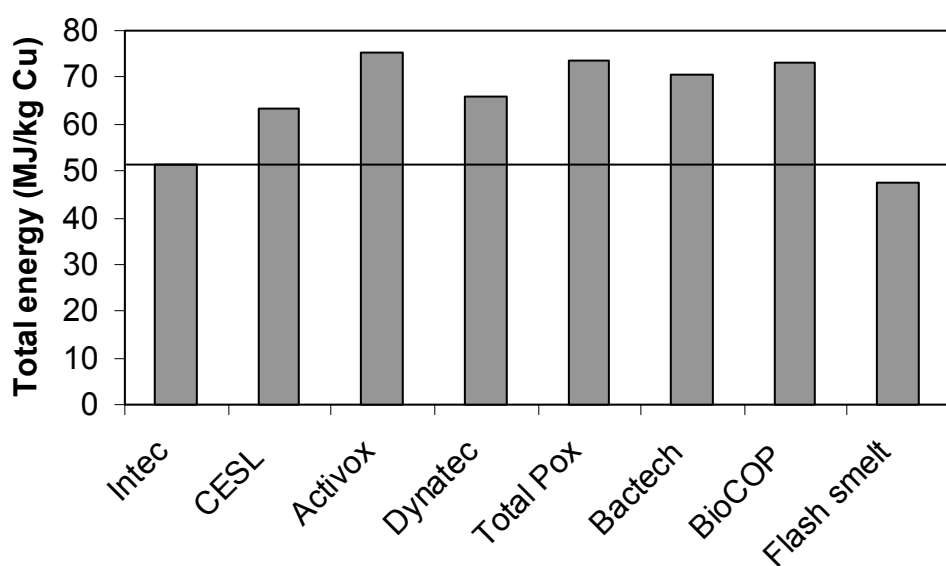


Figure 10. Total energy consumption (with melting and casting).

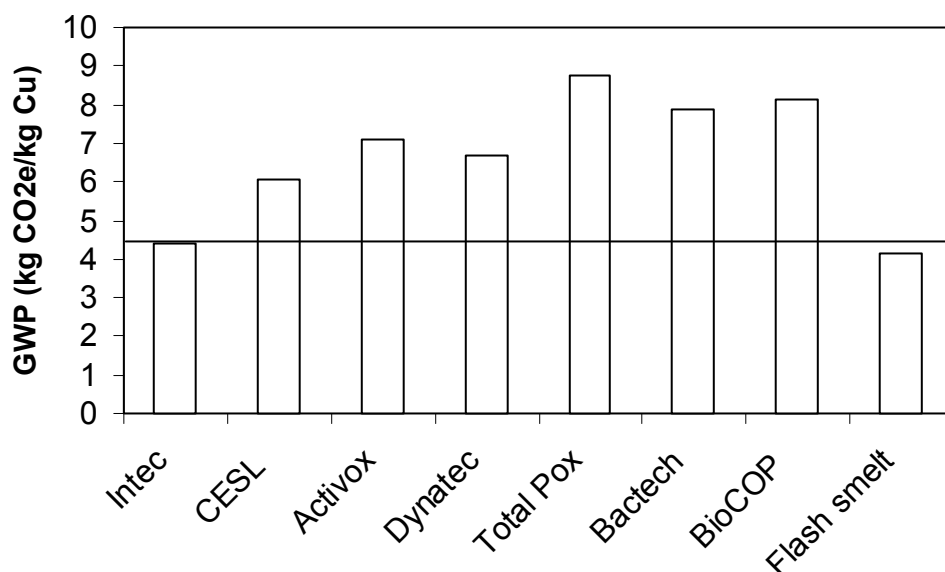


Figure 11. GWP (without melting and casting).

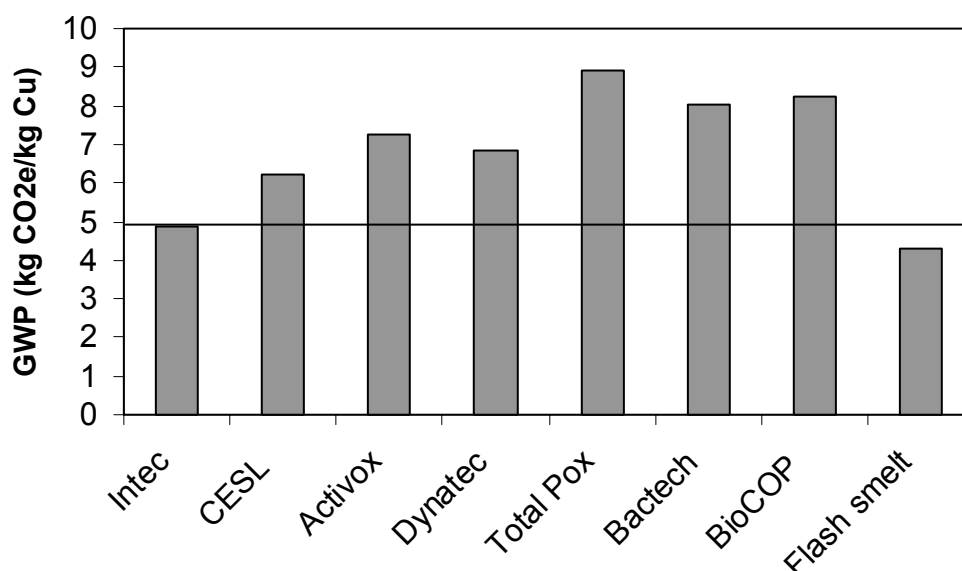


Figure 12. GWP (with melting and casting).

The Intec process had the lowest total energy and GWP of the seven hydrometallurgical processes considered, being in the order of 20-25% less than the next lowest process (CESL). However the total energy and GWP values for the Intec process were 4% and 7% greater than the corresponding values for the flash smelting process respectively without the product melting and casting step included. When the latter step is included these differences increase to 8% and 13% for total energy and GWP respectively. This increase results from the use of electrical energy (with its associated generation inefficiencies) for melting in the Intec process while thermal energy is used for melting in the other processes (eg. shaft furnaces – see Biswas and Davenport (1994), p. 436). The AP of the Intec process was the lowest of all the processes considered, including flash smelting. All subsequent process

comparisons in the remainder of the report include the melting and casting step.

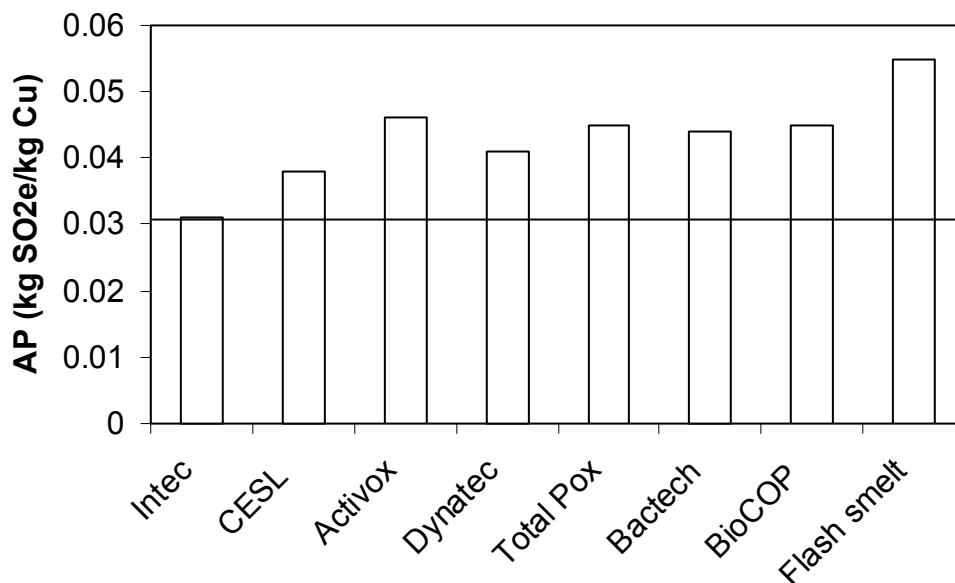


Figure 13. AP (without melting and casting).

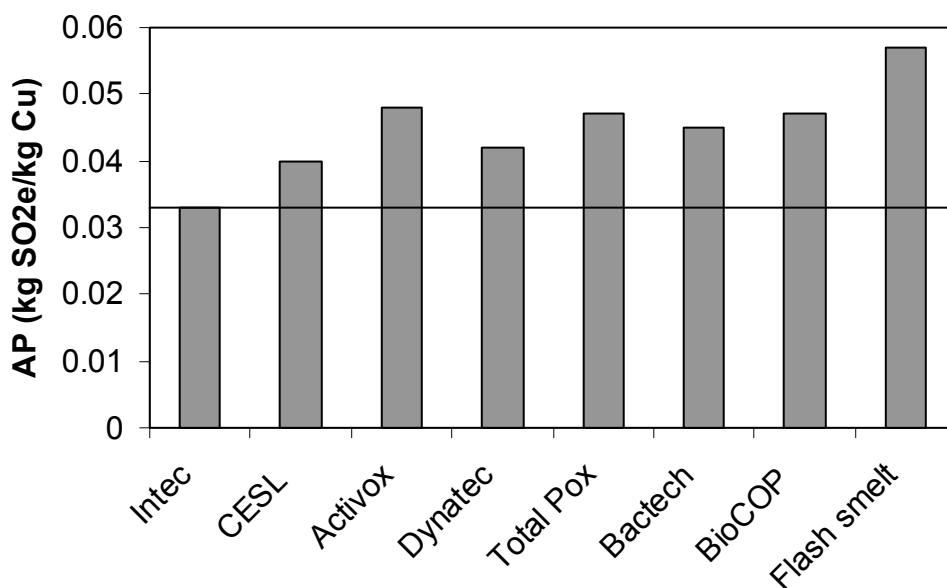


Figure 14. AP (with melting and casting).

The contributions of the various processing stages (mining, mineral processing, metal production and refining) to the total energy consumption, GWP and AP are shown in Figures 15-17 for the various processes, with the product melting and casting step included. It is obvious from these figures that the metal production and refining stage makes by far the greatest contribution to the overall impacts as this is by far the most energy-intensive stage in the overall metal production process. As the mining and mineral processing stages are common to all processes, the contributions of these stages are almost the same for all processes, with very minor variations (not

noticeable in Figures 15-17) due to differences in copper recovery between the various processes.

The metal production and refining step contributed about 61% to the total process energy consumption for the Intec process, while for the other hydrometallurgical processes the contribution was higher and in the order of 70% and for flash smelting it was 56%

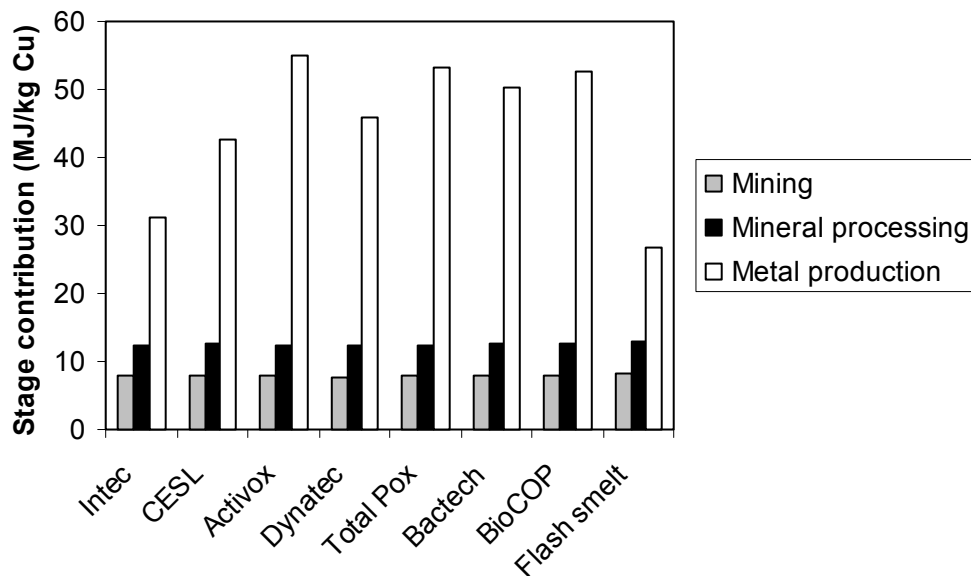


Figure 15. Process stage contributions to total energy (with melting and casting).

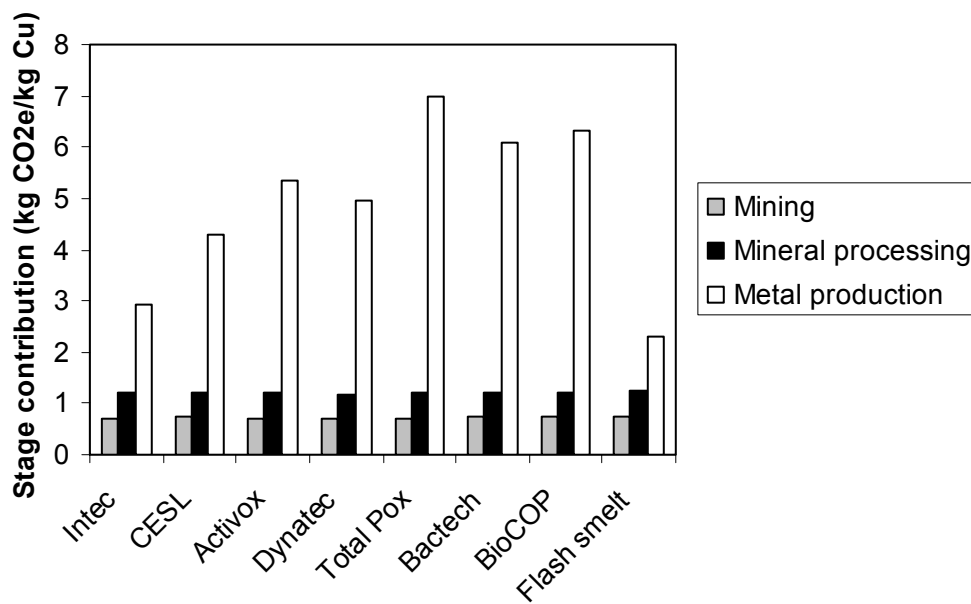


Figure 16. Process stage contributions to GWP (with melting and casting).

Table 4. Base case results (per kg of copper) – without melting and casting.

	Process							
	Intec	CESL	Activox	BacTech	Dynatec	Total PrOx	BioCOP	Flash smelting
<i>Total energy (MJ/kg)</i>	47	61	73	68	64	71	71	45
<i>Gaseous emissions</i>								
CO ₂ (kg/kg)	4.3	5.9	6.9	7.7	6.5	8.6	7.9	4.0
CO (g/kg)	1.2	1.2	1.5	1.5	1.4	1.6	1.5	2.0
N ₂ O (g/kg)	0.04	0.05	0.05	0.05	0.05	0.05	0.05	0.03
CH ₄ (g/kg)	6.1	7.3	9.1	8.6	8.1	8.9	8.9	6.2
NOx (g/kg)	23.2	27.6	34.3	32.6	30.6	33.7	33.6	24.1
NMVOC ** (g/kg)	0.5	0.5	0.6	0.6	0.6	0.6	0.6	0.7
SO ₂ (kg/kg)	0.014	0.019	0.022	0.021	0.019	0.022	0.022	0.039
GWP (kg CO ₂ e/kg)	4.4	6.1	7.1	7.9	6.7	8.8	8.1	4.2
AP (kg SO ₂ e/kg)	0.031	0.038	0.046	0.044	0.041	0.045	0.045	0.055

** Non Methane Volatile Organic Compounds.

Note: The above LCA results should be considered as first estimates for each process at their reported stages of development - the relative rankings of the respective processes may change as more detailed operating data become available or as the processes are developed further.

Table 5. Base case results (per kg of copper) – with melting and casting.

	Process							
	Intec	CESL	Activox	BacTech	Dynatec	Total PrOx	BioCOP	Flash smelting
<i>Total energy (MJ/kg)</i>	51	63	75	71	66	73	73	48
<i>Gaseous emissions</i>								
CO ₂ (kg/kg)	4.7	6.1	7.0	7.8	6.6	8.7	8.0	4.1
CO (g/kg)	1.2	1.4	1.7	1.7	1.6	1.7	1.7	2.2
N ₂ O (g/kg)	0.04	0.05	0.05	0.05	0.05	0.05	0.05	0.03
CH ₄ (g/kg)	6.6	7.9	9.7	9.2	8.7	9.5	9.5	6.7
NOx (g/kg)	25.1	29.7	36.5	34.7	32.8	35.8	35.8	26.2
NMVOC ** (g/kg)	0.5	0.5	0.6	0.7	0.6	0.7	0.6	0.8
SO ₂ (kg/kg)	0.016	0.019	0.022	0.021	0.019	0.022	0.022	0.039
GWP (kg CO ₂ e/kg)	4.9	6.2	7.2	8.0	6.8	8.9	8.3	4.3
AP (kg SO ₂ e/kg)	0.033	0.040	0.048	0.045	0.042	0.047	0.047	0.057

** Non Methane Volatile Organic Compounds.

Note: The above LCA results should be considered as first estimates for each process at their reported stages of development - the relative rankings of the respective processes may change as more detailed operating data become available or as the processes are developed further.

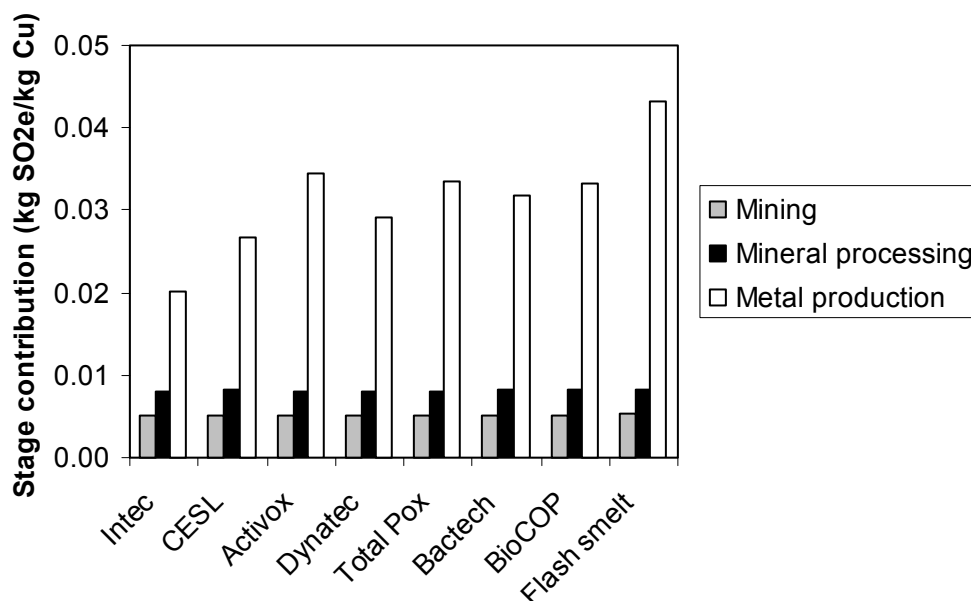


Figure 17. Process stage contributions to AP (with melting and casting).

6.2 Effect of selected process parameters

The effects of changing some selected process parameters on the LCA results for the various processes were also investigated. The two parameters concerned were:

- copper concentrate grade
- fuel used for electricity generation (ie. natural gas instead of black coal)

6.2.1 Copper concentrate grade

The concentrate grade assumed for the base case was 25% Cu (and 30% S – see Section 5) . Two other lower levels of copper grade were also considered, viz. 20% and 15% Cu, for each process. All the processes considered (including flash smelting) were expected to be capable, at least in principle, of treating these lower grade concentrates, although this would probably not be economic in the case of flash smelting.

The following issues will influence the LCA results when considering lower grade concentrates:

Mineral processing

- concentrator recovery
- concentrate composition, in particular sulphur
- concentrator energy consumption

Hydrometallurgical processing

- leach energy consumption
- leach oxygen consumption and neutralisation requirements
- leach recovery and grinding energy consumption

Pyrometallurgical processing

- smelting energy consumption
- smelting flux and oxygen consumption

Concentrator recovery

There is an approximately inverse relationship between recovery and grade of concentrate in all concentrating processes. This is shown in Figure 18 which has been derived from various data reported in the literature (eg. McKee et al (1976), Griffin et al (1993)). The base case recovery (93.7%) and grade (25%) are shown on this figure and concentrator copper recoveries of 96.4% and 97.8% were estimated from this figure for concentrate grades of 20% and 15% Cu respectively.

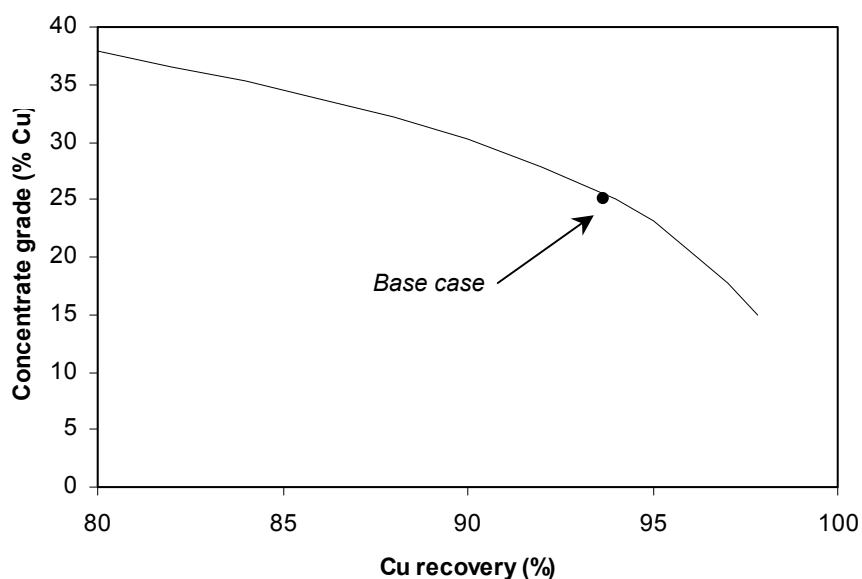


Figure 18. Concentrator recovery versus concentrate grade.

Concentrate composition

If the base case concentrate is assumed to be composed of chalcopyrite, pyrite and gangue then the amount of these components present is given in Table 6 (assuming the gangue does not contain any significant amounts of S and Cu). If it is further assumed that the reduction in concentrate grade with increasing concentrator recovery as shown in Figure 18 results from a lower recovery of chalcopyrite and a higher recovery of pyrite, with the gangue content remaining essentially unchanged, then the compositions of the two lower grade concentrates will be as shown in Table 6. The higher sulphur

contents in the lower grade concentrates will require additional oxygen in the leach step of the various hydrometallurgical processes.

Table 6. Concentrate compositions and concentrator recoveries.

	Concentrate grade (%Cu)		
	25	20	15
Chalcopyrite (%)	72.2	57.8	43.3
Pyrite (%)	8.9	23.3	37.8
Gangue (%)	18.9	18.9	18.9
Sulphur (%)	30.0	32.7	35.4
Concentrator recovery (%)	93.7	96.4	97.8

Concentrator energy consumption

A lower grade concentrate with higher recovery means a larger amount of concentrate will be produced from the same amount of ore. However the same amount of ore will be fed to the flotation circuit, and the same amount of material (concentrate plus tailings) will exit from the circuit. Therefore it is not expected that a reduction in concentrate grade will have any significant effect on the concentrator power consumption when expressed per tonne of ore (ie. 31 kWh/t ore - see Table 2). However there will be a reduction when expressed per tonne of concentrate or per tonne of copper in concentrate due to the greater amounts of these recovered, as shown in Table 7.

Table 7. Concentrator power consumption for different concentrate grades.

	Concentrate grade (% Cu)		
	25	20	15
kWh/t ore	31	31	31
kWh/t concentrate	276	214	158
kWh/t Cu in concentrate	1103	1072	1057

Leach energy consumption

Information provided by Houllis (2001) indicates that a fixed amount of copper per unit volume of liquor is added to the leach step of the Intec process. Thus with lower grade concentrates the amount of solids added per unit volume of liquor increases. The increased slurry volume will mean an increase in the power consumption of the leach step (eg. agitation, pumping, etc.) but not for subsequent downstream steps as the solids are separated from the liquor immediately after the leach. This scenario was also assumed to apply to the other hydrometallurgical processes considered.

Further information provided by Houllis (2001) indicated that a reduction in concentrate grade from 25 to 20% increases the slurry volume by about 1.5%. The power consumption for motors and mixing for the various processes was estimated to be in the order of 400-500 kWh/t Cu (see Appendix 1). Assuming a mean value of 450 kWh/t Cu and further assuming that the power consumption for agitation and pumping is roughly proportional to slurry volume, this increase in slurry volume will consume an additional 7 (ie. 450 x

1.5/100) kWh/t Cu. Extending this line of reasoning to a concentrate grade of 15% will see the leach power consumption increase by 14 kWh/t Cu over the base case.

Leach oxygen consumption and neutralisation requirements

As mentioned above, the additional sulphur in the lower grade concentrates will increase the oxygen consumption (as air or oxygen) in the leach step of the various hydrometallurgical processes, with either partial or total oxidation of the sulphur to sulphate. The additional sulphate will in turn require more limestone for neutralisation purposes (this increased consumption is easily calculated as the limestone consumptions in the LCA inventory (Table 2) are expressed per t of S in the concentrate feed).

For those processes using oxygen, the contributions of oxygen delivery to the process power consumptions are shown in Table 8 for the three concentrate grades considered. These contributions were calculated according to the following equation:

$$P = O \times S \times 400 / C$$

where

- P = power consumption for oxygen delivery (kWh/t Cu)
- O = leach oxygen consumption (t/t S in conc) – see Table 2
- S = % S in concentrate
- C = % Cu in concentrate
- 400 = oxygen plant power consumption (kWh/t oxygen)

For example, with the base case concentrate (25% Cu, 30% S) the oxygen delivery contribution to the CESL process power consumption is

$$\begin{aligned} P &= 0.74 \times 30 \times 400 / 25 \\ &= 355 \text{ kWh/t Cu} \end{aligned}$$

For those plants using air, the contribution of air delivery to the process power consumption is also shown in Table 8 for the three concentrate grades considered. This contribution was calculated according to the following equation:

$$P = A \times S \times 31 / C$$

where

- P = power consumption for air delivery (kWh/t Cu)
- A = leach air consumption (t/t S in conc) – see Table 2
- S = % S in concentrate
- C = % Cu in concentrate
- 31 = air delivery power consumption (kWh/t air - Dreisinger (1998))

For example, with the base case concentrate (25% Cu, 30% S) the air delivery contribution to the Intec process power consumption is

$$P = 3.34 \times 30 \times 31 / 25 \\ = 124 \text{ kWh/t Cu}$$

Table 8. Oxygen and air delivery power contributions (kWh/t Cu).

Process	Oxygen or air consumption (t/t S in concentrate)	Concentrate grade (% Cu)		
		25	20	15
Intec	3.34 (air)	124	169	244
CESL	0.74	355	484	699
Dynatec	1.23	590	804	1161
Activox	0.67	322	438	632
Total pressure oxid	2.20	1056	1439	2077
BacTech	16.36 (air)	609	830	1198
BioCOP	16.36 (air)	609	830	1198

Leach recovery and grinding energy consumption

Copper metal recovery for the various processes for the base case concentrate (25% Cu) were given earlier (Section 5) and were in the range 97-99% for the seven hydrometallurgical processes considered. With a lower grade concentrate it may be necessary to grind the concentrate to a finer size than for the base case if the same copper recovery is to be achieved. However there is insufficient data available at present on the effects of ore grade and grind size on copper recovery to quantify this effect. Therefore for the purposes of this study it was assumed that a reduction in concentrate grade from 25 to 15% Cu did not significantly alter the grinding energy consumption (expressed per t concentrate) for any of the hydrometallurgical processes. While this approach may underestimate the effect of concentrate grade on grinding energy consumption, it was considered that this conservative approach was more appropriate than making grinding energy estimates based on only a few data points.

The base case grinding energy consumption for the Intec, CESL, Dynatec and Total Pressure Oxidation processes was estimated to be 8.5 kWh/t concentrate while for the Activox, Bactech and BioCOP processes it was estimated to be 48.2 kWh/t concentrate. These values correspond to a ground concentrate P_{80} (80% passing size) of 44 μm and 10 μm respectively from a feed size P_{80} of 75 μm (Dreisinger, 1998). When expressed per t of copper product (ignoring minor differences in copper recovery between the processes) these values become 34 kWh/t Cu and 193 kWh/t Cu. Reducing the concentrate grade in turn from 25% Cu to 20% Cu and then to 15% Cu increases these grinding energy values to 43 kWh/t Cu and 242 kWh/t Cu (20% Cu concentrate) and to 57 kWh/t Cu and 321 kWh/t Cu (15% Cu concentrate) respectively.

The additional power requirements for the various hydrometallurgical processes for the two lower concentrate grades were estimated by adding the respective mixing, oxygen/air delivery and grinding contributions calculated above, as shown in Table 9.

Table 9. Process power consumption (kWh/t Cu) for different concentrate grades.

	Intec	CESL	Dynatec	Activox	Total POx	BacTech	BioCOP
<i>25% Cu</i>							
PROCESS	535	900	1088	1900	1675	1435	1667
<i>20% Cu</i>							
<i>Extra power</i>							
Mixing	7	7	7	7	7	7	7
O ₂ /air	45	129	214	116	383	221	221
Grinding	9	9	9	49	9	49	49
PROCESS	596	1045	1318	2072	2074	1712	1944
<i>15% Cu</i>							
<i>Extra power</i>							
Mixing	14	14	14	14	14	14	14
O ₂ /air	120	344	571	310	1021	589	589
Grinding	23	23	23	128	23	128	128
PROCESS	692	1281	1696	2352	2733	2166	2398

Smelting energy consumption

In the absence of a flash smelting simulation model to provide process mass and energy balances for different concentrate grades, it was not possible to make detailed estimates of the effect of concentrate grade on the smelting energy consumption. While a larger fuel input will be needed for the sensible heat (including drying) requirements of the increased feed, air and oxygen inputs at lower concentrate grades, this will more than likely be offset by a greater amount of heat from the smelting reactions due to the higher sulphur content in the concentrates. Therefore as a first approximation, it was assumed that concentrate grade did not have any significant effect on the smelting fuel consumption.

The smelting and converting electricity consumption for the base case concentrate was estimated at 1143 kWh/t Cu (see Table 2), including acid and oxygen plant contributions. A larger amount of lower grade concentrate with higher sulphur content will require more oxygen for the smelting reactions and will produce more sulphuric acid in the acid plant as well as more slag. Therefore the smelter electricity consumption will increase with lower concentrate grade due to the increased contributions of the acid and oxygen

Table 10. Electricity, flux and oxygen consumptions for smelting different concentrate grades.

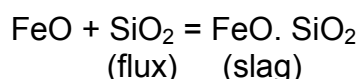
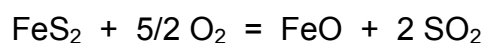
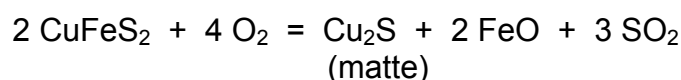
	Concentrate grade (% Cu)		
	25	20	15
Electricity (kWh/t Cu)	1143	1314	1562
Limestone (kg/t conc)	21	22	23
Silica (kg/t conc)	130	137	144
Oxygen (kg/t Cu)	834	1162	1702

plants, concentrate dryer and slag concentrator. Estimates of the smelter electricity consumption for different concentrate grades are given in Table 10.

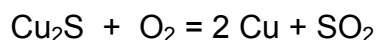
Smelting flux and oxygen consumption

The smelting flux requirements will increase at lower concentrate grades and more slag will be produced as indicated above. In order to estimate the new flux requirements it was assumed that the amount of flux added is proportional to the iron plus gangue content of the concentrate, and the base case flux additions given in Table 2 were scaled accordingly to give values for the lower concentrate grades and these are also given in Table 10.

The smelting of the chalcopyrite and pyrite concentrate can be represented by the following simplified equations (Biswas and Davenport (1994)):



while for the subsequent converting of the copper matte to blister copper:



The flash smelting oxygen requirements will increase at lower concentrate grades as was also indicated above. The stoichiometric oxygen consumption for the above smelting and converting reactions, using the compositions given earlier in Table 6) were used to scale the base case smelting oxygen consumption (see Table 2) to give values for the lower concentrate grades and these are also given in Table 10.

The various process LCA spreadsheet models were re-run with the revised concentrator recoveries, concentrate grades, electricity and other inputs used in place of the base case values given in the inventory table (Table 2). The results from these model re-runs are given in Table 11 in terms of total energy, GWP and AP and compared graphically in Figures 19 - 24 for total energy and GWP at the three concentrate grades.

These results show that while there was a reduction in concentrator energy consumption per tonne of copper at lower concentrate grades (Table 7), this was offset by the increase in the energy consumption of the metal production stage (expressed per tonne of copper) due to the extra amount of concentrate that had to be treated in this stage, giving rise to a net increase in the total energy of the overall process, together with a corresponding increase in GWP and AP. The extent of this effect varied between processes, ranging from a 2% increase in total energy for the Intec process up to 16% for the Total

Table 11. Effect of concentrate grade (with melting and casting).

	Concentrate grade (% Cu)								
	25			20			15		
	Energy	GWP	AP	Energy	GWP	AP	Energy	GWP	AP
Intec	51.3	4.9	0.033	51.4	4.9	0.033	52.1	5.0	0.034
CESL	63.2	6.2	0.040	64.9	6.5	0.041	68.3	7.0	0.042
Dynatec	65.9	6.8	0.042	68.2	7.3	0.044	72.5	8.2	0.046
Activox	75.1	7.2	0.048	77.0	7.5	0.049	80.6	8.0	0.050
Total POx	73.5	8.9	0.047	77.8	10.1	0.049	85.6	12.1	0.054
BacTech	70.6	8.0	0.045	73.5	8.8	0.047	78.9	10.2	0.050
BioCOP	73.0	8.3	0.047	75.9	9.1	0.048	81.3	10.5	0.051
Flash smelting	47.5	4.3	0.057	51.5	4.6	0.069	58.5	5.2	0.089

Note: Units are MJ/kg Cu for energy, kg CO₂-e/kg Cu for GWP and kg SO₂-e/kg Cu for AP.

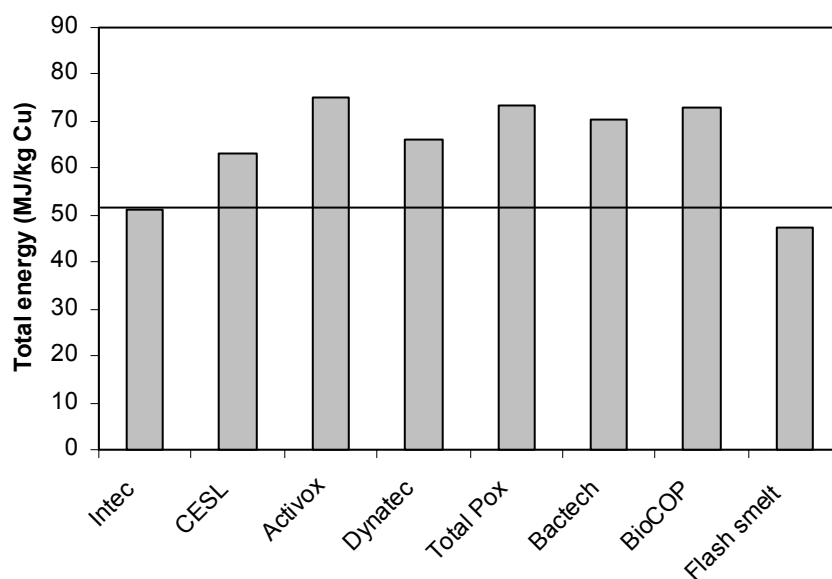


Figure 19. Total energy consumption for 25% Cu (with melting and casting).

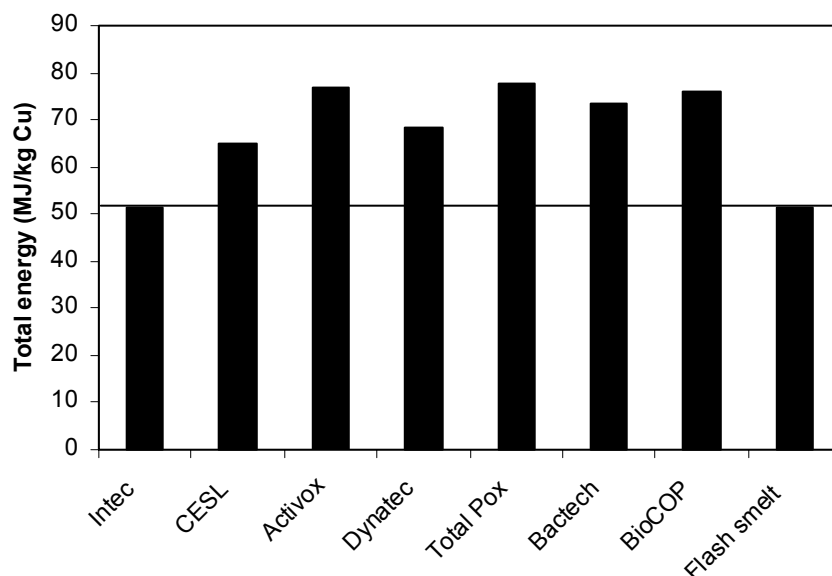


Figure 20. Total energy consumption for 20% Cu (with melting and casting).

Pressure Oxidation process and 23% for the Flash Smelting process for a decrease in concentrate grade from 25% Cu to 15% Cu.

At the lowest concentrate grade (15% Cu) the Intec process had the lowest total energy consumption of all processes considered, including flash smelting, while at a 20% Cu concentrate grade the total energy consumption of the Intec process was more or less the same (0.2% lower) as that for the flash smelting process.

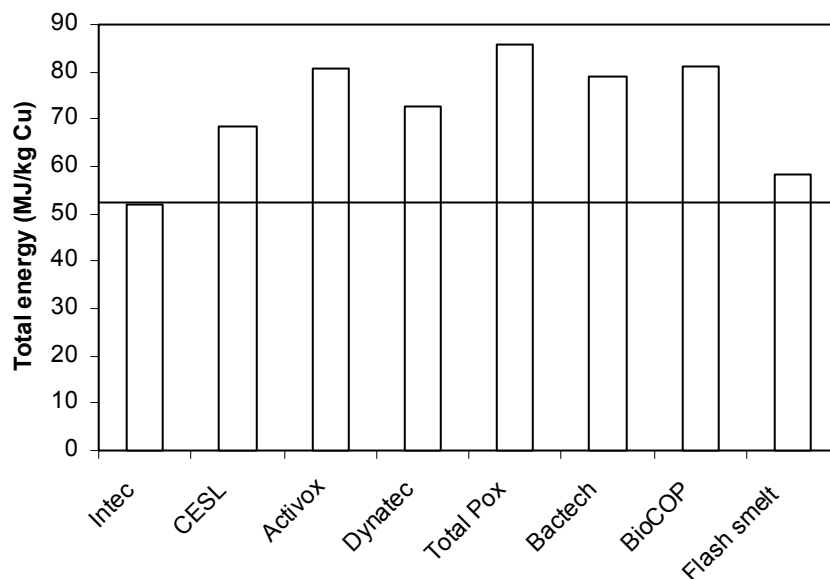


Figure 21. Total energy consumption for 15% Cu (with melting and casting).

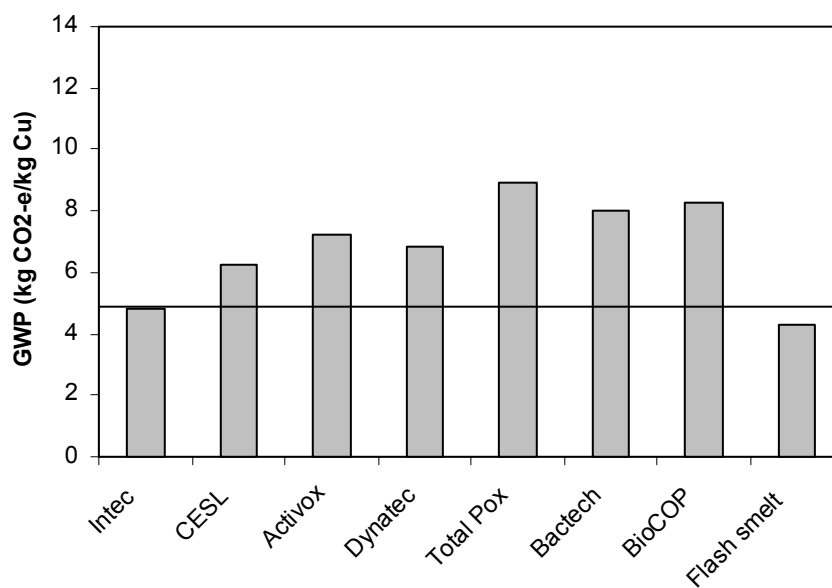


Figure 22. GWP for 25% Cu (with melting and casting).

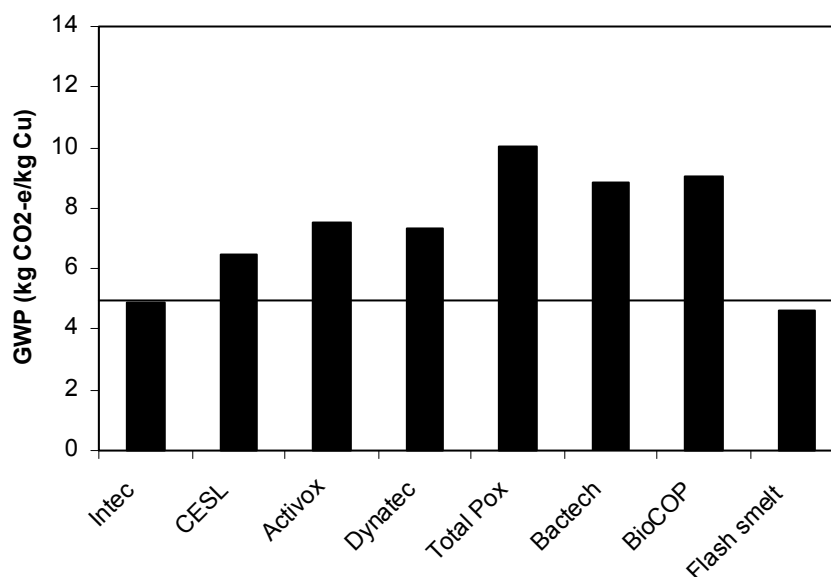


Figure 23. GWP for 20% Cu (with melting and casting).

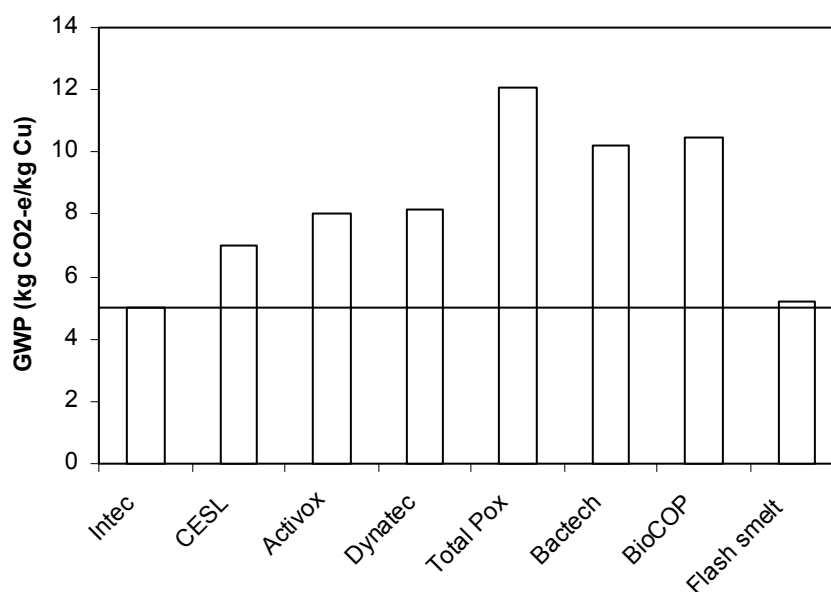


Figure 24. GWP for 15% Cu (with melting and casting).

6.2.2 Fuel type for electricity generation

Black coal was assumed to be the fuel source for electricity generation for the various process base cases (at 35% generation efficiency). The effects of changing to natural gas as the fuel source for electricity generation (at the same generation efficiency) on the GWP and AP for the various processes are given in Table 12 with the results for GWP shown graphically in Figure 25. Note that the total energy consumption is not affected by changing the fuel source for power generation at the same generation efficiency.

Table 12. Effect of electricity fuel type on GWP and AP (with melting and casting).

Process	Black coal		Natural gas	
	GWP (kg CO ₂ -e/kg)	AP (kg SO ₂ -e/kg)	GWP (kg CO ₂ -e/kg)	AP (kg SO ₂ -e/kg)
Intec	4.9	0.033	3.4	0.010
CESL	6.2	0.040	4.4	0.012
Dynatec	6.8	0.042	5.0	0.013
Activox	7.2	0.048	5.1	0.014
Total pressure oxidation	8.9	0.047	6.8	0.014
BacTech	8.0	0.045	6.0	0.014
BioCOP	8.3	0.047	6.2	0.014
Flash smelting/converting	4.3	0.057	3.2	0.039

As natural gas-based electricity has a lower greenhouse gas intensity than black coal-based electricity (0.57 t CO₂/MWh cf. 0.96 t CO₂/MWh), it is not surprising that changing the fuel source for electricity generation from black coal to natural gas reduced the GWP for all processes considered, typically in the order of 25-30%. The APs of the hydrometallurgical processes were reduced in the order of 70% in changing to natural gas-based electricity, while for the flash smelting process it was reduced by about 30%.

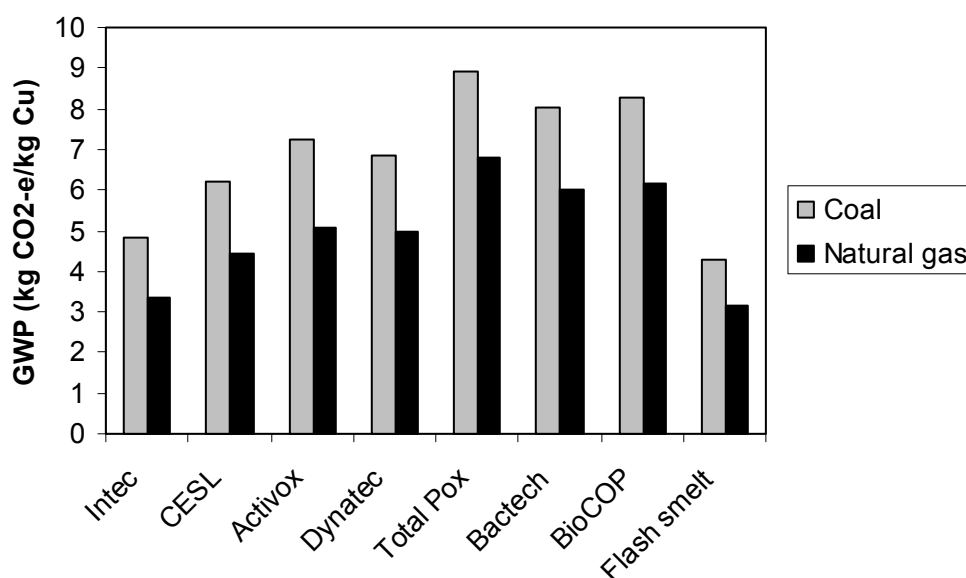


Figure 25. Effect of electricity fuel type on GWP (with melting and casting).

7 DISCUSSION

The various hydrometallurgical processes considered in this comparative LCA study are at different stages of development. As such, the amount of data available in the literature varied from process to process and were not very detailed for any of the processes. This meant that a range of assumptions and approximations had to be made to establish an LCA inventory table for each process, in particular for electricity and fuel inputs. Furthermore, it is to be expected that the performance data for the various processes will be site and application specific and strongly dependent on concentrate mineralogy.

For this reason it is emphasised that the LCA results presented in this report should not be considered as definitive results for the various processes examined, but rather as first estimates for the various processes at their reported stages of development. Therefore the relative rankings of the various processes may change as more detailed operating data become available or as the processes are developed further. It is also worth noting that all operating data published or provided by the various companies for their respective processes were accepted in good faith by CSIRO Minerals.

All hydrometallurgical processes had total energy consumptions (and associated GWPs) greater than that for flash smelting for the base case concentrate. This is not entirely unexpected, as all the hydrometallurgical processes include an electrowinning step which consumes a considerable amount of electrical power and therefore these processes are at a disadvantage when assessed on a life cycle basis due to power plant generation inefficiencies. Of all the hydrometallurgical processes examined, the Intec process had the lowest electrowinning power consumption and had a base case total energy consumption only about 4% greater than that for flash smelting without the product melting and casting step included. Including the melting and casting step increased this difference to 8% as electrical heating, with its associated generation inefficiencies, is used for product melting purposes in the Intec process while direct thermal energy is used for melting purposes in the other processes.

Electrical power generation at efficiencies greater than the 35% (Australian power grid average in 1995) assumed for this study would improve the standings of the hydrometallurgical processes relative to flash smelting in terms of total energy consumption, but this aspect was not addressed in any real detail in the present study with respect to black coal or natural gas-based electricity. However, as hydroelectricity has been reported (Briem et. al. (2000)) to have a best present generation efficiency of 80%, it was of interest to see how this source of electricity affected the results. While hydroelectricity is often assumed not to be associated with any greenhouse gas emissions, recent studies (ACARP (2000), Carvalho and Bizzo (2000)) have suggested that decaying vegetation submerged by flooding may give off appreciable quantities of greenhouse gases. The amount of greenhouse gases emitted from hydroelectric dams varies greatly, depending on climatic factors and the nature of the land that is flooded to create the dam. Based on data provided by these authors, a best estimate greenhouse gas emission value of 190 kg CO₂-e/MWh was assumed for hydroelectricity. The results for hydroelectricity are given in Table 13 and shown graphically in Figures 26 and 27 for total energy and GWP respectively. As suggested above, the increase in generation efficiency from 35% (black coal) to 80% (hydroelectricity) improved the energy standings of the hydrometallurgical processes (mean 46% reduction in total energy) over the flash smelting process (38% reduction in total energy), with the total energy of the Intec process being some 8% less than for flash smelting at the base case concentrate grade of 25% Cu. The GWP results basically mirrored the total energy results although the magnitude of the reductions (on a % basis) were somewhat higher.

Table 13. Effect of hydroelectricity on total energy and GWP.

Process	Black coal		Hydroelectricity	
	Total energy (MJ/kg)	GWP (kg CO ₂ -e/kg)	Total energy (MJ/kg)	GWP (kg CO ₂ -e/kg)
Intec	51.3	4.9	27.0	1.5
CESL	63.2	6.2	34.0	2.2
Dynatec	65.9	6.8	35.9	2.7
Activox	75.1	7.2	40.3	2.4
Total pressure oxidation	73.5	8.9	39.9	4.2
BacTech	70.6	8.0	38.3	3.5
BioCOP	73.0	8.3	39.4	3.6
Flash smelting/converting	47.5	4.3	29.4	1.8

Note : 35% generation efficiency for black coal-based electricity, 80% for hydroelectricity

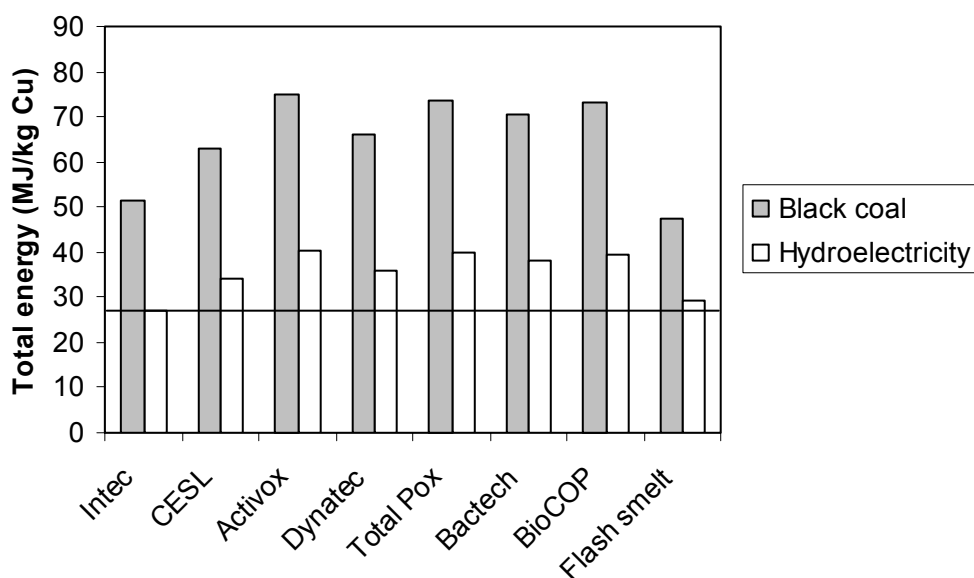


Figure 26. Effect of hydroelectricity on total energy.

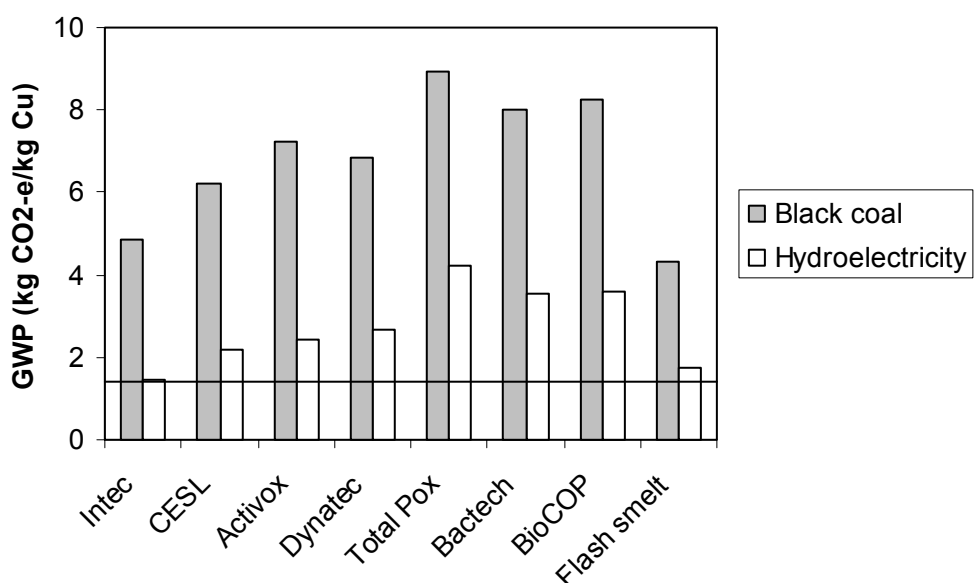


Figure 27. Effect of hydroelectricity on GWP.

While operating at concentrate grades lower than the base case does give rise to a slight increase (2% for a reduction in concentrate grade from 25 to 15% Cu) in the total energy consumption of the Intec process (due to the larger amount of concentrate to be handled per t of copper product), it does appear to offer advantages over the other hydrometallurgical processes at these lower grades, where the total energy increases ranged from about 7-17% for the same change in concentrate grade. This is due primarily to the lower oxygen/air requirements of the Intec process (3.34 t air/t S or 0.77 t O₂/t S – see Table 8) relative to the other processes combined with the lower power requirement to supply oxygen as air rather than as oxygen directly (ie. 31/0.23 = 135 kWh/t O₂ cf. 400 kWh/t O₂). Not needing to grind to ultrafine sizes also contributes to this result (see Table 9). The total energy consumption of the Intec process was about 11% lower than that for flash smelting at a concentrate grade of 15% Cu and virtually the same as flash smelting at a 20% Cu concentrate grade. However, while flash smelting could in principle handle concentrates of such low grades, it would probably not in practice as this would more than likely be uneconomic. It should also be borne in mind that it was necessary to make two important assumptions in this study to examine the effect of concentrate grade, namely that a reduction in concentrate grade does not have any significant effect on the hydrometallurgical leach grinding energy consumption or on the flash smelting fuel consumption. These assumptions could have an influence on the results.

The total energy results presented in this report for the various hydrometallurgical processes, ranging from 47 MJ/kg Cu for the Intec process up to 73 MJ/kg Cu for the Activox process (without product melting and casting), compare with a value of 64 MJ/kg Cu reported by Norgate and Rankin (2000) for heap leaching (plus SX/EW) of copper sulphide ore (also without product melting and casting). Similarly, the GWP and AP results ranging from 4.4 kg CO₂-e/kg Cu and 0.031 kg SO₂-e/kg Cu for the Intec process up to 8.8 kg CO₂-e/kg Cu and 0.046 kg SO₂-e/kg Cu for the Total Pressure Oxidation and Activox processes respectively, compare with values of 6.2 kg CO₂-e/kg Cu and 0.05 kg SO₂-e/kg Cu reported by the same authors.

7.1 Sensitivity analysis

Given the major contribution that the power consumption of the various hydrometallurgical processes makes to the total energy consumption for each process, and the difficulties encountered in establishing the various base case values, a sensitivity analysis was carried out by varying the process power consumptions by 30% above and below the base case values given in Table 2 and Appendix 1. The results of this analysis are shown in Figure 28 which indicates that even with a 30% reduction in the assumed base case process power consumptions, none of the other hydrometallurgical processes had a lower total energy consumption than the Intec base case.

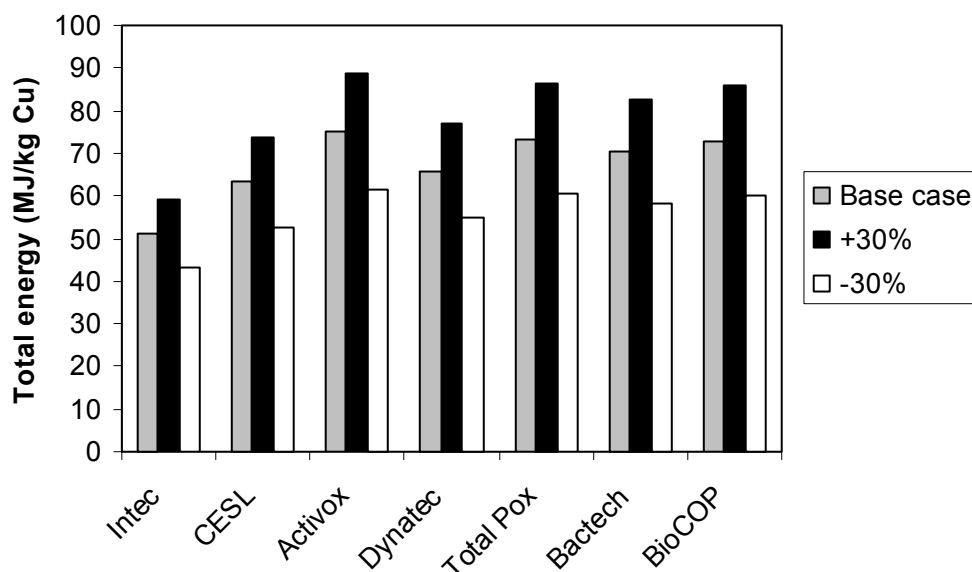


Figure 28. Sensitivity of hydrometallurgical processes to power consumption.

It was also of interest to determine the sensitivity of the flash smelting LCA results to the concentrate transport mode and distance between mine and smelter. This was previously assumed to be 500 km by rail (see Section 5). The flash smelting LCA spreadsheet model was rerun with a combination of different concentrate transport modes and distances and the results are given in Table 14. This table shows that while concentrate transport mode and distance does have an observable effect on the flash smelting results, they were not overly sensitive to these parameters (viz. 0.5 MJ/kg Cu/500 km for rail and 0.2 MJ/kg Cu/500 km for ship).

Table 14. Effect of concentrate transport mode and distance on flash smelting results.

Impact parameter	Transport mode and distance (km)					
	Rail	0	500 *	500	500	500
	Ship	0	0 *	500	1000	5000
Total energy (MJ/kg)		47.01	47.54	47.75	47.96	49.65
GWP (kg CO ₂ -e/kg)		4.46	4.30	4.31	4.33	4.45
AP (kg SO ₂ -e/kg)		0.056	0.057	0.057	0.058	0.060

* Base case

8 ACKNOWLEDGEMENTS

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APPENDIX 1

Hydrometallurgical process energy calculations

Precious metals recovery

CYANIDATION

Source: Jones (1999)

Power operating cost	0.5 c/lb Cu
Power cost	3.5 c/kWh
Electricity consumption	= (0.5/3.5) x 2204.6 = 315 kWh/t Cu

However subsequent information from Jones (2001) indicated that PM recovery (primarily cyanidation) power consumption is about 200 kWh/t Cu, so therefore choose to use the latter value here rather than the back-calculated value.

Fuel operating cost	0.5 c/lb Cu
Fuel cost	\$5/GJ
Fuel consumption	= (0.5/500) x 2204.6 x 1000 = <u>2205 MJ/t Cu</u>

Note: Jones (1999, 1996) indicates some residue pre-treatment is required before cyanidation but it does not appear to be a lime boil, therefore assume the above fuel input is for carbon regeneration

LIME BOIL

Lime boil simulated using METSIM flowsheeting package based on information provided by Dreisinger (1999). Assume 20% w/w residue slurry at 25 C heated to 95 C and held for 10 hours (residue solids assumed to be all hematite).

Fuel consumption estimate = 2718 MJ/t Cu

Oxygen plant

Electricity consumption = 400 kWh/t O₂ (Jones (1999) and other sources)

Solvent extraction and electrowinning

SOLVENT EXTRACTION

Electricity consumption = 550 kWh/t Cu (Jenkins et. al. (1999))

However subsequent information from Jones (2001) indicated that the power consumed for SX is quite a bit less than this, and that part of the the reason for this is associated with how the power is allocated between SX and EW. The SX power consumption back-calculated from the data provided by Jones (2001) was in the order of 200 kWh/t Cu and this value was then used for all processes included in the study.

ELECTROWINNING

Electricity consumption = 2100 kWh/t Cu (Jenkins et. al. (1999))
 = 1650 kWh/t Cu for INTEC (Houllis, (2000))

Melting and casting**MELTING**

Source: Biswas and Davenport (1994) p. 436

Mean thermal energy consumption for 2 sites = 2000 MJ/t Cu
 = 38 kg natural gas/ t Cu
 cf. electric furnace input for melting (Intec) above = 400 kWh/t Cu
 = 1440 MJ/t Cu

CASTING

Source: Various

25 kWh/t metal mean of reported values for continuous casting of steel –
 assume similar figure applies to casting of copper

Note: Summing the total energy above for melting and casting gives 2090
 MJ/t Cu which falls within the range reported by Paschen et al (1991) of 2000-
 3000 MJ/t Cu for melting and casting of copper.

Processes**INTEC***Electricity*

Grinding	35 kWh/t Cu	(Houllis, 2000)
Motors	400 kWh/t Cu	(Houllis, 2000)
Furnace	400 kWh/t Cu	(Houllis, 2000)
Auxiliaries	<u>100 kWh/t Cu</u>	(Houllis, 2000)
	<u>935 kWh/t Cu</u>	

Fuel

Heating	3860 MJ/t Cu	(Houllis, 2000)
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CESL*Electricity*

Total process 3400 kWh/t Cu (Jones (2001))
 Less SX/EW and PM recovery = 3400 – 2500 = 900 kWh/t Cu

Fuel

Cyanidation 2205 MJ/t Cu

DYNATEC*Electricity*

Grinding	34 kWh/t Cu	(Dreisinger (1998))
Autoclave	364 kWh/t Cu	(Dreisinger (1998))
Oxygen supply	590 kWh/t Cu	(Dreisinger (1998))
Auxiliaries	<u>100 kWh/t Cu</u>	(assume same as for Intec)
	<u>1088 kWh/t Cu</u>	

Fuel

Cyanidation 2205 MJ/t Cu
 Lime boil 2718 MJ/t Cu

ACTIVOX*Electricity*

Power operating cost 8 c/lb Cu (Evans (1999))
 Power cost 4 c/kWh (Evans (1999))
 Electricity consumption = (8/4) x 2204.6 = 4409 – say 4400 kWh/t Cu
 The latter figure was subsequently confirmed by Evans (2001).
 Less SX/EW and PM recovery = 4400 – 2500 = 1900 kWh/t Cu

Fuel

Cyanidation 2205 MJ/t Cu
 Lime boil 2718 MJ/t Cu

TOTAL PRESSURE OXIDATION*Electricity*

Grinding	34 kWh/t Cu	(Dreisinger (1998))
Autoclave	485 kWh/t Cu	(Dreisinger (1998))
Oxygen supply	1056 kWh/t Cu	(Dreisinger (1998))
Auxiliaries	<u>100 kWh/t Cu</u>	(assume same as for Intec)
	<u>1675 kWh/t Cu</u>	

Fuel

Cyanidation	2205 MJ/t Cu
Lime boil	2718 MJ/t Cu

BACTECH*Electricity*

Grinding	201 kWh/t Cu	(Dreisinger (1998))
Mixing	500 kWh/t Cu	(Dreisinger (1998))
Air delivery	634 kWh/t Cu	(Dreisinger (1998))
Auxiliaries	<u>100 kWh/t Cu</u>	(assume same as for Intec)
	<u>1435 kWh/t Cu</u>	

Fuel

Cyanidation	2205 MJ/t Cu
Lime boil	2718 MJ/t Cu

BIOCOP*Electricity*

The electricity consumption of the BioCOP process was initially considered likely to be similar to that of the Bactech process as both are bioleaching processes. The overall power consumption for the Bactech process (including SX/EW and PM recovery) was estimated to be 3935 kWh/t Cu as shown below. Subsequent information indicated that overall power consumption for bioleaching processes may be slightly higher and in the order of 4400 kWh/t Cu. It was decided that until further data become available, the mean of these two values, viz. 4167 kWh/t Cu would be used for the BioCOP process. The power consumption of the hydrometallurgical step of this process was back-calculated accordingly, as shown below.

Fuel

Cyanidation	2205 MJ/t Cu
Lime boil	2718 MJ/t Cu

Summary

Electricity (kWh/t Cu)

	Process	Lime boil	Cyanidation	SX	EW	Melting & casting	Total without melting & casting	Total with melting & casting
Intec	535	-	-	-	1650	425 **	2185	2610
CESL	900	-	200	200	2100	25	3400	3425
Dynatec	1088	-	200	200	2100	25	3588	3613
Activox	1900	-	200	200	2100	25	4400	4425
Total Pox	1675	-	200	200	2100	25	4175	4200
BacTech	1435	-	200	200	2100	25	3935	3960
BioCOP	1667	-	200	200	2100	25	4167	4192

** includes 400 kwh/t for melting; 25 kwh/t for casting

Fuel (MJ/t Cu)

	Process	Lime boil	Cyanidation	Melting	Total without melting & casting	Total with melting & casting
Intec	3860	-	-	-	3860	3860
CESL	-	-	2205	2000	2205	4205
Dynatec	-	2718	2205	2000	4923	6923
Activox	-	2718	2205	2000	4923	6923
Total Pox	-	2718	2205	2000	4923	6923
BacTech	-	2718	2205	2000	4923	6923
BioCOP	-	2718	2205	2000	4923	6923

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