Low grade ores – Smelt, leach or concentrate?

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A R T I C L E   I N F O

Article history:
Received 14 July 2009
Accepted 3 October 2009
Available online 7 November 2009

Keywords:
Hydrometallurgy
Leaching
Pyrometallurgy
Environmental
Extractive metallurgy

A B S T R A C T

Metallic ore grades are falling globally as the higher grade reserves are exploited first and are progressively depleted. At the same time, the demand for primary metals extracted from these ores is expected to increase, despite increased levels of dematerialisation and recycling. Sustainability concerns have highlighted the need to meet these demands while at the same time minimising resource consumption and environmental emissions. A study was therefore undertaken using life cycle assessment methodology to examine various alternative processing routes for extracting metal from low grade ores (down to 0.1% metal), particularly those of copper and nickel, in terms of their life cycle-based energy consumption (embodied energy) and greenhouse gas emissions. The processing routes examined included conventional concentrating and smelting, direct ore smelting, heap leaching, pressure leaching and in situ leaching. This paper presents the results of this study.

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1. Introduction

Present supplies of metals can be primary, secondary or both. Primary or "new" metals are extracted from newly mined ores then added to the existing inventory of metals (i.e., "metals in use") for the first time. Secondary (recycled) metals having once entered the metal inventory as primary metals experience service lives of varying duration, at the end of which they are either wasted, recycled or re-allocated. Wasted metals are lost from the inventory, whereas recycled and re-allocated metals combine with the supply of primary metal to enlarge the inventory. Despite increased levels of dematerialisation and recycling, it can be expected that there will be an on-going need for primary metals well into the future as the world population increases and more people strive to achieve a higher standard of living. Laznicka (2006) suggests that in addition to the classical ore deposits, numerous non-traditional geological metal sources (e.g., seabed nodules, earth crust rocks) will likely contribute to the future supply of metals as shown in Fig. 1. Steen and Borg (2002) estimated the cost of producing metal concentrates by heap leaching of the earth's crust containing 0.002% copper and 0.001% nickel (amongst other metals) and found it to be orders of magnitude greater than current costs of producing concentrates from metallic ores. However, the ore grade (i.e., metal content) of the earth's crust is below the so-called 'mineralogical barrier' (Skinner, 1976), where metals are trapped by atomic substitution rather than being present as separate minerals as shown in Fig. 2. The work described in this paper relates to ore grades above this 'mineralogical barrier'.

Table 1 shows the reserves of some common metals and the years of supply remaining at the current rate of consumption based on data reported by the US Geological Survey, along with typical economic processing ore grades. Sohn (2006) and Tilton and Lagos (2007) have reported similar results for years of supply of copper and nickel to those shown in Table 1. In practice, metal reserves change constantly, reflecting changes in metal prices, relative currency values and technological developments, and are more appropriately thought of as working inventories. The world reserves of metals from existing resources could increase significantly if metal prices increase and/or new processing technologies are developed that lower the economically recoverable ore grade from a resource. Alternatively, metal reserves may also increase through new resources being identified. However, the grades of metallic ores are falling globally as the higher grade reserves are exploited first and are progressively depleted. For example, Fig. 3 shows the general decline in base and precious metal ore grades in Australia over time (Mudd, 2007). Furthermore, many of these newer reserves are fine-grained, requiring finer grind sizes in order to achieve mineral liberation. Both of these issues, either together or in isolation, have significant implications for primary metal production in the future, although based on current reserves, this may not occur for some decades.

In an earlier paper (Norgate and Jahanshahi, 2006) the authors used life cycle assessment (LCA) methodology to examine the effect of ore grades down to 0.25% and grind sizes down to 5 μm on the life cycle-based energy consumption and greenhouse gas emissions of copper and nickel metal production by conventional pyrometallurgical processing (i.e., ore concentration followed by smelting and refining). The results showed that the effect of declining ore grades on these environmental impacts is significant...
at grades below about 1% due to the additional energy consumed (and greenhouse gases emitted) in the mining and mineral processing stages to move and treat the additional gangue (waste) material. In light of these findings, a further study was carried out to examine some alternative processing routes for low grade ores, in particular those of copper and nickel, in terms of their embodied energy and greenhouse gas performance. This work is presented in this paper.

2. Energy required for metal extraction

As the focus of this paper is primarily energy consumption and associated greenhouse gas emissions for primary metal production, it is of interest to compare the theoretical and actual energy consumptions for metal extraction, as this comparison gives an indication of the likely scope for reducing energy consumption in practice. Metals are generally extracted from either oxide or sulphide ores. The oxides and sulphides of the important industrial metals are chemically stable and significant energy is required to break the chemical bonds to produce metal. Gibbs free energy is the ultimate measure of chemical stability, but the heat of formation normally dictates the minimum energy requirements for metal extraction processes (Brooks and Subagyo, 2002). The theoretical and actual energy consumptions for the production of various metals (extraction stage only) reported in the literature are compared in Table 2, while Fig. 4 compares the ratios of actual to theoretical energy calculated by the authors from the mean values in Table 2, to those reported by Gupta (2003) and Ray et al. (2005).

Actual energy values are influenced by process technology route (e.g. pyrometallurgical versus hydrometallurgical – see later), although attempts have been made to compare the same processing route for any particular metal in Table 2 and Fig. 4. However,

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**Table 1**

<table>
<thead>
<tr>
<th>Metal</th>
<th>Economic ore grade (%w/w)</th>
<th>Reserves (Mt of metal)</th>
<th>Production in 2006 (Mt/y)*</th>
<th>Years of supplyb,c</th>
</tr>
</thead>
<tbody>
<tr>
<td>Iron/steel</td>
<td>30–60</td>
<td>79,000</td>
<td>858</td>
<td>92</td>
</tr>
<tr>
<td>Aluminium</td>
<td>27–29</td>
<td>4675</td>
<td>33</td>
<td>142</td>
</tr>
<tr>
<td>Copper</td>
<td>0.5–2</td>
<td>480</td>
<td>15.3</td>
<td>31</td>
</tr>
<tr>
<td>Lead</td>
<td>5–10</td>
<td>67</td>
<td>3.4</td>
<td>20</td>
</tr>
<tr>
<td>Zinc</td>
<td>10–30</td>
<td>220</td>
<td>10</td>
<td>22</td>
</tr>
<tr>
<td>Nickel</td>
<td>1.5–3</td>
<td>64</td>
<td>1.6</td>
<td>40</td>
</tr>
</tbody>
</table>

*a* Includes primary and secondary metal.

*b* Assumes consumption rate closely balanced to total production rate.

*c* Assumes no recycling.
the processing route is not always clear for some of the energy data reported in the literature. The reported actual energy values are also influenced by whether or not electrical energy is converted to primary energy, and if so, what thermal efficiency is used. A common value of 35% has been used by the authors above, but once again, these two points are not always clear in the reported data. Nevertheless, despite these uncertainties, the results can be used to make broad comparisons of the relative energy efficiencies of the various metal production processes. It can be seen from Fig. 4 that iron and steel production is the most energy efficient process, while nickel is the least. The remainder of the paper focusses on alternative processing routes for reducing the energy consumption and associated greenhouse gas emissions for nickel and copper production, particularly at low ore grades.

### 3. Assessment of alternative processing routes

Metals are generally extracted from ores by either pyrometallurgical or hydrometallurgical processing routes. The former involves smelting of the ore or concentrate at high temperatures, while the latter involves leaching the ore or concentrate at relatively low temperatures, including ambient. These two broad processing paths may be broken down further into the following routes, as shown in Fig. 5, for copper and nickel ores:

#### Pyrometallurgical processing
- concentration and smelting;
- direct smelting of ore;

#### Hydrometallurgical processing
- in situ leaching of ore;
- heap or pressure leaching of ore or concentrate.

Unlike the other processing routes shown in Fig. 5, the direct ore smelting route is not used commercially for the production of copper or nickel metal (although ferronickel is produced by smelting nickel laterite ores without any prior concentration). These various processing routes are discussed in more detail later.

### 3.1. Copper and nickel ores

About 80% of the world’s copper comes from copper sulphide and copper–iron sulphide ores, with chalcopyrite \((\text{CuFeS}_2)\) being the most common mineral in these ores. \text{Cu–Fe–S} minerals are not easily dissolved by aqueous solutions, so the vast majority of copper extraction from these minerals is pyrometallurgical. The remaining 20% of the world’s copper is produced hydrometallurgically.
cally from oxidised copper minerals (e.g. oxides and carbonates), as most of these ores are not amenable to present-day flotation techniques. About 70% of the world's land-based nickel resources are contained in laterites and the about 30% as sulphides. However, laterites currently only account for about 40% of world nickel production compared to 60% for sulphides, although laterite production is expected to overtake sulphide production by the end of the decade. Unlike sulphide ores, laterite ores are mineralogically and chemically complex which makes processing more complicated as they are difficult to concentrate. The sulphide ores and about 35% of the laterite ores are processed pyrometallurgically, with the remainder of the laterite ores processed hydrometallurgically. The main nickel sulphide mineral is pentlandite ((Fe,Ni)9S8), which occurs with pyrrhotite (Fe1−xC15S8), pyrite (FeS2) and chalcopyrite. Nickel laterites include such minerals as garnierite ((Ni,Mg)12SiO32(OH)2) and nickeliferous limonite (nickel mixed with hydrated iron oxide).

3.2. Life cycle assessment

Life cycle assessment has become a widely accepted methodology for assessing the environmental impacts of the production or provision of products or services, and essentially involves the compilation of an inventory of relevant environmental exchanges during the life cycle of the product or service and evaluating the potential environmental impacts associated with those exchanges. The full product life cycle is usually divided into the following stages:

- cradle to entry gate (raw material extraction and refining);
- entry gate to exit gate (product manufacture);
- exit gate to grave (product use, recycling and disposal).

Based on impact assessment, two types of LCA can be distinguished; problem-oriented (mid points) or damage-oriented (end points). In the problem-oriented approach followed here to assess the various ore processing routes, the environmental loads quantified in the inventory analysis were classified into the environmental impacts to which they contribute using appropriate equivalency factors.

Individual “cradle-to-gate” LCA spreadsheet models of each ore processing route for copper and nickel production were set up, with each flowsheet being constructed at a level of detail consistent with processing data available in the literature. The inventory data used for each processing route were typical process data averaged over a number of sources where possible. This generally resulted in a flowsheet of 3–5 process steps, e.g. mining, mineral processing, smelting and refining. The main environmental impact category considered was greenhouse gas emissions (i.e. global warming potential (GWP)), with the intergovernmental panel on climate change (IPCC) characterisation model being used to calculate this impact category. The gross energy requirement (GER), also referred to as embodied energy or cumulative energy demand, which is the cumulative amount of primary energy consumed in all stages of a metal’s production life cycle was also included in the LCAs. It was assumed that electric power was generated from black coal in all cases, at a generation efficiency of 35%. The inventory data and underlying assumptions used in setting up these spreadsheet models have been reported previously for most of the processing routes considered (Norgate and Rankin, 2000). The study used the international standards framework for conducting life cycle assessments contained in the ISO 14040 series. A functional unit of 1 kg of refined metal was used in the LCAs and no allocation of environmental impacts was made to any sulphuric acid produced. The effect of declining ore grades on the two impacts referred to above for each processing route was obtained by reducing the input grades into the respective LCA spreadsheet models. The upper limit of ore grade used for a particular processing route was limited to that usually considered applicable to that route, particularly as practised in Australia, while the lower limit was 0.1% in all cases. The effect of grind size was obtained by using the Bond equation for grinding energy consumption (Norgate and Jahanshahi, 2006).

3.3. Process modelling

While data, however brief in some cases, have been reported in the literature for most of the processing routes considered, this was not the case for the direct smelting route, as this process is not used commercially for the production of copper or nickel metal from their ores as pointed out earlier. Therefore it was necessary to conduct process modelling studies of the direct ore smelting route in order to derive mass and energy balances that could be used to provide LCA inventory data for this processing route. Thermody-
namic modelling of the process was initially performed using CSIRO's MPE (multi-phase equilibrium) model (Jahanshahi et al., 2004) to give the processing conditions necessary to produce smelting products (matte and slag) of the required composition based on the initial ore compositions assumed. For this paper, only the direct smelting of copper ore has been considered. The composition assumed for the copper ore was chalcocite (2.5%), pyrite (48.3%) and quartz (49.2%). The processing conditions obtained from the MPE model were then incorporated into a simulation model of the process set up using the commercial process flow-sheeting package METSIM. This model was subsequently used to derive mass and energy balances for the process.

4. Pyrometallurgical processing

4.1. Concentration and smelting

The conventional pyrometallurgical processing route of copper and nickel sulphide ores involves the following stages:

- beneficiation of the ore by grinding and froth flotation to produce a concentrate;
- smelting of the concentrate to produce a high-copper or nickel matte (copper or nickel-enriched molten sulphide phase);
- converting the molten matte to produce blister copper or an upgraded nickel matte;
- refining of the blister copper and nickel matte to produce high purity copper and nickel metal.

As mentioned earlier, the effects of declining ore grades and finer grind sizes on the life cycle-based energy consumption and greenhouse gas emissions of copper and nickel metal production by this route were reported by Norgate and Jahanshahi (2006), and these earlier embodied energy results extended down to 0.1% ore grade are shown in Figs. 6 and 7 and Table 3. The trends for greenhouse gases (in terms of CO₂ equivalents) with declining ore grades are given in Table 4 and are similar to those for embodied energy, as most of the energy inputs are fossil fuel-based.

4.2. Direct smelting

Rather than beneficiating the ore by grinding and flotation to produce a concentrate for smelting, an alternative is to smelt the whole ore directly. While this processing route eliminates the beneficiation stage and its associated impacts (including energy consumption and greenhouse gas emissions), more solid material has to be handled and heated up to smelting temperatures (typically 1250°C for copper) requiring additional energy input (and associated greenhouse gas emissions). For the purposes of this study, only a conceptual version of this processing route for copper ores was considered, with no particular smelting technology specified, although some of the emerging technologies for direct iron-making could be amenable to this type of processing. This processing route was not considered for nickel ores as MPE modelling indicated that the relatively high MgO content typically present in many Australian nickel ores would result in high slag volumes relative to matte, with significant nickel losses to the slag.

Many copper mines have waste sulphide dumps containing significant amounts of iron sulphides (pyrite and pyrrhotite). Using this material as a flux and fuel source (the oxidation of sulphides is exothermic) in the direct smelting route has the potential to reduce the fossil fuel-based energy requirements of the process as

Fig. 6. Effect of ore grade and grind size on the embodied energy of copper production (concentrating and smelting).

Fig. 7. Effect of ore grade and grind size on the embodied energy of nickel production (concentrating and smelting).

Table 3

<table>
<thead>
<tr>
<th>Processing route</th>
<th>Copper Ore grade (% Cu)</th>
<th>Nickel Ore grade (% Ni)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.1 0.25 0.5 1.0 2.0 3.0</td>
<td>0.1 0.25 0.5 1.0 2.0 2.3</td>
</tr>
<tr>
<td>Concentrating and smelting</td>
<td>662 249 131 72 43 33</td>
<td>791 366 224 154 118 114</td>
</tr>
<tr>
<td>– 75 µm (grind size)</td>
<td>1165 474 244 129 71 52</td>
<td>1421 618 350 216 150 141</td>
</tr>
<tr>
<td>– 5 µm (grind size)</td>
<td>954 385 195 100 – – –</td>
<td>– – – – – – – – – – –</td>
</tr>
<tr>
<td>Direct smelting</td>
<td>791 332 179 103 64 52</td>
<td>– – – – – – – – – – –</td>
</tr>
<tr>
<td>Heap leaching</td>
<td>1182 469 267 153 96 77</td>
<td>1497 624 333 187 115 105</td>
</tr>
<tr>
<td>Pressure leaching</td>
<td>1345 554 290 158 – – –</td>
<td>1262 530 286 164 – – –</td>
</tr>
<tr>
<td>In situ leaching</td>
<td>1345 554 290 158 – – –</td>
<td>1262 530 286 164 – – –</td>
</tr>
</tbody>
</table>
well as disposing of a waste that can give rise to acid mine drainage (the outflow of acidic water from mine sites). This option was included in the process modelling outlined earlier and the results indicated that no external fossil fuel was required for the smelting and converting stages. However, this was offset by additional electrical power requirements for the smelting stage due to the additional amount of feed material to be treated, as well as for the acid plant to convert the additional amount of sulphur in the furnace off-gas into sulphuric acid. The effects of declining ore grades on embodied energy and greenhouse gas emissions from the direct smelting of copper ore are shown in Tables 3 and 4 and Fig. 8.

5. Hydrometallurgical processing

The conventional hydrometallurgical processing route of copper oxide and sulphide ores and nickel laterite ores involves the following stages:

- leaching (heap or pressure) of copper or nickel from crushed and/or ground ore or concentrate to produce an impure copper or nickel-bearing aqueous solution;
- transfer of copper or nickel from this impure solution to pure, high-copper or nickel electrolyte using solvent extraction;
- electrowinning pure copper or nickel from this electrolyte.

5.1. Heap leaching

Heap leaching was originally developed to process copper oxide ores, but was subsequently modified to accomodate an increased contribution from sulphide minerals in the ore (Watling, 2006). Bacterially-assisted heap leaching of low-grade copper sulphides is a developing technology that has been applied successfully to the extraction of copper from sulphide minerals such as chalcocite at ore grades down to 0.15% (Watling, 2006). The ore is typically crushed to about 6–13 mm prior to stacking into heaps. The effect of declining ore grades on embodied energy for heap leaching of copper sulphide ore is shown in Fig. 9 and Tables 3 and 4.

5.2. Pressure leaching

Nickel can be solubilised from laterites using sulphuric acid at atmospheric pressure, however this requires a large amount of acid (Whittington and Muir, 2000), making the process uneconomic. The main reason for this is that for limonitic ores (low nickel, high iron) in particular, the very high iron content must be dissolved in order to solubilise the nickel. High pressures and temperatures significantly reduce the acid consumption.

Pressure leaching of nickel laterite ores using sulphuric acid in autoclaves at around 250°C followed by solvent extraction and electrowinning is now an established processing route for nickel metal production. Laterite ores do not generally require grinding for metal liberation, and the leaching process is fairly insensitive to particle size (Kyle, 1996), although it has been reported that grinding improves the extraction of cobalt in the laterite ore (Whittington and Muir, 2000). The ‘grinding circuit’ is therefore used to slurry the ore, and may include a SAG mill or ball mill. Ores are typically ground to 100–250 μm prior to the whole ore being leached. The effect of declining ore grades on embodied energy for pressure acid leaching of nickel laterite ore is shown in Fig. 10 and Tables 3 and 4.
Pressure leaching has also been applied in recent years to the extraction of copper from chalcopyrite ores. Direct leaching of these ores has limited application, mainly because the kinetics of chalcopyrite dissolution is extremely slow even in very strong oxidising leaching media as pointed out earlier. A number of processes are in varying stages of development, from pilot plant through to commercial facility. One such process is the total pressure oxidation process (Padilla et al., 2007; Dreisinger, 2006; Peacey et al., 2003) in which ore is first concentrated then leached in an autoclave at a temperature of about 200–230 °C in a sulphate medium using oxygen as the oxidant. The grind size for concentration is typically about 40 μm (Dreisinger, 2006). Padilla et al. (2007) showed that the rate of dissolution during the pressure oxidation leaching of chalcopyrite increased as the grind size decreased.

The effects of declining ore grades on embodied energy and greenhouse gas emissions from the pressure acid leaching of chalcopyrite are also shown in Tables 3 and 4 and Fig. 10. While reducing the grind size of the concentrate may improve leaching performance as mentioned above, this does not greatly increase the embodied energy and greenhouse gas emissions as only the concentrate is reground and not the whole ore as was the case in Figs. 6 and 7. For example, reducing the concentrate grind size from the base case value of 40 μm indicated above to 5 μm, increases the embodied energy and greenhouse gas emissions of refined copper metal from 77 MJ/kg and 9.8 kg CO₂e/kg respectively (see Tables 3 and 4) to 79 MJ/kg and 10.0 kg CO₂e/kg. Furthermore, as a relatively constant grade of concentrate (e.g. 27% Cu) is typically produced irrespective of initial ore grade, any increase in embodied energy and greenhouse gas emissions with decreasing concentrate grind size is relatively unaffected by any accompanying changes in ore grade.

5.3. In situ leaching

In situ leaching, also known as solution mining, involves leaving the ore where it is in the ground, and using a lixiviant (liquid medium) which is pumped through it to recover the minerals out of the ore by leaching. Consequently there is little surface disturbance and no tailings or waste rock generated. However, the orebody needs to be permeable to the liquids used, and located so that they do not contaminate ground water away from the orebody. In situ leaching occurs prolifically in nature, but usually over a long time span. Commercial applications of in situ leaching seek to greatly accelerate and improve the recovery of metals over these natural forms of in situ leaching. In situ leaching has been used commercially for the extraction of uranium, copper and gold, with around 16% of the world’s uranium production being by this method (O’Gorman et al., 2004). Copper has been recovered by in situ leaching at a number of copper deposits (including some where chalcrocite is the major copper mineral) in Arizona using sulphuric acid. Copper recoveries of 50–60% have been reported, although with some degree of uncertainty (O’Gorman et al., 2004).

The energy requirement for in situ leaching varies over a wide range because geochemical conditions, depth of ore body, number of wells, operational life of each well and ore properties vary widely. Mudd and Diesendorf (2007) reported a value of 172 GJ/t U₃O₈ (mainly used for pumping purposes) for the Beverley uranium in situ leach mine in Australia, which, for a reported ore grade of 0.18% U₃O₈ and an assumed recovery of 80% (Australian Uranium Association, 2006), corresponds to 69 kW h/t ore. However, data on other in situ leaching operations are lacking in the literature, and therefore it was assumed that a similar amount of pumping energy is required for in situ leaching of copper and nickel ores. Recoveries of 55% were also assumed for in situ leaching of copper and nickel. The effects of declining ore grades on embodied energy and greenhouse gas emissions for the in situ leaching of copper and nickel ores using sulphuric acid are shown in Tables 3 and 4 and Fig. 11.

6. Discussion

The embodied energy and GWP results for all processing routes are compared in Figs. 12 and 13 for copper and Figs. 14 and 15 for nickel. Only broad comparisons can be made between the pyrometallurgical and hydrometallurgical results in these figures as these two routes generally treat different types of ores, as outlined earlier. More stricter comparisons can be made within each of these routes for processes that are amenable to the same ore type. Marsden (2008) reported total energy consumption data (but not life cycle-based) for a number of alternative processing routes for copper sulphide ores. These data were converted by the authors to an approximate life cycle basis by assuming that the reported electricity consumption was generated from black coal at 35% efficiency, giving embodied energy for the concentration and smelting and heap leaching routes of 131 MJ/kg Cu and 70 MJ/kg Cu respectively, both at 0.5% Cu ore grade. These values compare with values...
of 114 MJ/kg Cu and 179 MJ/kg Cu respectively from the present study shown in Fig. 12 at the same ore grade. The difference in the case of heap leaching can be explained by the higher electricity consumption assumed for this process in the present study, which is similar to the electricity consumption data reported by Giurco et al. (2001) for this processing route.

In the case of copper, if no additional grinding of the ore is required as the ore grade falls then concentrating and smelting is the preferred route where pyrometallurgical processing can be applied. However, if fine grinding is required (down to 5 \( \mu \)m) then the preferred routes are heap leaching and direct smelting, in that order. In situ leaching is the least preferred processing route followed by pressure leaching, although it must be recalled that the in situ leaching results are based on limited data extrapolated from uranium mining. Concentrating and smelting is preferred to direct smelting until the grind size falls below about 10 \( \mu \)m, at which point direct smelting becomes the preferred route at all grades between 0.1\% and 1.0\% copper. In the case of nickel, concentrating and smelting is also the preferred route provided no additional grinding of the ore is required as the ore grade falls. If fine grinding is required (down to 5 \( \mu \)m) then in situ leaching becomes the preferred route, slightly ahead of pressure acid leaching.

It should be noted that the above results are dependent on the metal recoveries assumed for each processing route, as the lower the recovery assumed, the higher the embodied energy and greenhouse gas emissions per unit mass of metal produced. The metal recoveries used in the study were taken from the literature and are given in Table 5. While falling ore grades and finer grind sizes

<table>
<thead>
<tr>
<th>Table 5</th>
<th>Metal recoveries (%) for copper and nickel processing routes.</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Concentrating and smelting</td>
</tr>
<tr>
<td>Copper</td>
<td>91</td>
</tr>
<tr>
<td>Nickel</td>
<td>78</td>
</tr>
</tbody>
</table>
may influence these recoveries, the extent of this influence is difficult to quantify at the present time and was not considered in the study. Nevertheless, the results of this study suggest that no definitive answer can be given to the question posed in the title of this paper – the most appropriate route for processing low grade ores in terms of embodied energy and greenhouse gas emissions will largely depend on the mineralogy of the ore deposit concerned. Furthermore, no economic considerations have been included in identifying the most appropriate route for processing low grade ores, and such issues will strongly influence the route eventually chosen. For example, economic considerations would suggest that the direct smelting route would require the smelting plant to be located in close proximity to the mine site in order to avoid excessive transportation costs in moving the ore from the mine to the smelter. The potential impacts of a number of emerging technologies which could also influence the route chosen have not been considered in the paper. These include:

- ore sorting – which could be used to reduce the amount of ore (and hence energy) required for direct smelting;
- waste heat recovery – e.g. from smelting slags (Norgate and Jahanshahi, 2007), matts and other by-product streams – which could be used to reduce the energy consumption of both the conventional and direct smelting routes.

However, it is emphasised that the results presented in this paper only serve as first-pass comparisons of the various processing routes for low grade ores. More detailed operating data for the various processes are required before more extensive comparisons can be made.

7. Conclusions

Falling ore grades and more complex ore bodies anticipated in the future can be expected to lead to increased energy consumption and associated greenhouse gas emissions for primary metal production. Sustainability concerns have seen the mineral processing and metal production sector come under increasing pressure to address these issues, but choosing the most appropriate processing route for low grade ores is not always clear. A study was therefore undertaken using life cycle assessment methodology to examine various alternative processing routes for extracting metal from low grade ores (down to 0.1% metal), particularly those of copper and nickel, in terms of their life cycle-based energy consumption (embodied energy) and greenhouse gas emissions. The processing routes examined included conventional concentrating and smelting, direct ore smelting, heap leaching, pressure leaching and in situ leaching. The results of the study indicated that the most appropriate route for processing low grade ores in terms of embodied energy and greenhouse gas emissions largely depends on the mineralogy of the ore deposit concerned. Provided no additional grinding of the ore is required as the ore grade falls then conventional concentrating and smelting is the preferred route for both copper and nickel where pyrometallurgical processing can be applied. However, if fine grinding is required (down to 5 μm) then the preferred routes are heap leaching and direct smelting, in that order for copper ores, and in situ leaching, slightly ahead of pressure acid leaching, for nickel ores.

References


