Net Smelter Return Models and Their Use in the Exploration, Evaluation and Exploitation of Polymetallic Deposits

Raymond Goldie
Richardson Greenshields of Canada Limited
Suite 1400, 130 Adelaide St. W.
Toronto, Ontario M5H 1T8

Peter Tredger
Tredgco Mineral Services
Suite 1400, 25 Adelaide St. E.
Toronto, Ontario M5C 1Y2

ABSTRACT

As an aid in visualizing a mineral deposit, the three spatial dimensions are often reduced to two, by projection onto plans or sections. Similarly, the concentrations of different metals in the deposit can be represented in a single dimension through the use of a common denominator, the Net Smelter Return (NSR) per tonne. NSR is defined as the proceeds from the sale of mineral products after deducting off-site processing and distribution costs.

In sulphide deposits in which the economic metals are principally copper, nickel, molybdenum or platinum group metals, a mine's receipts (the NSR) usually correspond to about 56%-60% of the gross value of the metals contained in the ore. This figure drops to around 40% for ores with significant quantities of lead or zinc. The presence of gold or silver in the ore will generally increase these percentages.

An NSR model of a deposit is a representation which attempts to approximate the actual NSR which would be derived from exploitation of the mineralization being modelled. NSRs are calculated from estimates of grades, recoveries, concentrate treatment schedules and commodity prices.

With the advent of cheap, readily available computer processing, it is now practicable for geologists to routinely use NSR models. At the exploration stage, NSR models provide a useful basis for presentation and comparison of drill-hole assays. At the evaluation stage of a project, NSR models are helpful in the estimation of optimum grades, recoveries, and production rates. At the exploitation stage, NSR models provide a rigorous basis for grade control.

A widely used alternative to the NSR per tonne as a common denominator, the "equivalent grade" (*e.g.*, "gold-equivalent" or "copper-equivalent"), is misleading and should be avoided.

INTRODUCTION

The Net Smelter Return (NSR) is a measure of the value of ore. For example, in the income statements of most Canadian mining companies, the top line ("sales", "net sales", "revenues" or "net revenues") is generally the NSR, which is defined as the proceeds from the sale of mineral products after deducting all off-mine costs relating to the transportation, treatment and sale of those products.

In the Preface of *Ore Deposit Models*, Roberts and Sheahan (1988) noted that "there are two components to an ore deposit model: the empirical model, which consists of an assemblage of data, including observational data, which characterizes the deposit; and a conceptual model that attempts to interpret the data through a unifying theory of genesis". NSR models are similar to other ore deposits models: the empirical component is based on an assemblage of data obtained from operating mines; the conceptual component provides the economic geologist with a framework for the evaluation of specific projects.

An NSR model of a mineral deposit is a representation which attempts to approximate the actual NSR which would be derived from exploitation of the mineralization being modelled. NSRs serve two main purposes for the economic geologist: (a) they provide a common denominator for the comparison of assays from polymetallic deposits; and (b) they instill a healthy awareness of the economic factors which determine the value of ore.

Comparison of assays. As an aid in visualizing a mineral deposit, the three spatial dimensions are often reduced to two, by projection onto plans or sections. Similarly, concentrations of different metals a polymetallic deposit can be represented in a single dimension through the use of a common denominator. The best such common denominator is the NSR per tonne.

A widely used alternative to the NSR per tonne is "equivalent grade", in which concentrations of different metals are converted to one metal equivalent according to their relative prices. This common denominator suffers from two disadvantages: (1) it assumes that treatment costs and metal losses in processing are the same for all commodities; and (2) despite fluctuating metal prices, the equivalence factors tend to forever remain at their initially calculated levels.

Awareness of economic factors. The use of NSRs forces geologists to consider a wide range of factors which determine whether or not a mineral deposit can be mined profitably. NSRs are determined not only by ore grades, but also by mill recoveries, concentrate grades, concentrate treatment charges, freight costs and metal prices. In the early exploration stages of a project, rough assumptions are acceptable. However, in more advanced stages of evaluation, careful estimation of these factors becomes critically important. NSR models highlight questions — particularly those pertaining to mineralogy — which must be considered in any project evaluation.

This paper outlines the calculation of NSRs for a variety of concentrate types derived from polymetallic ores, and illustrates applications of the NSR concept in three areas: exploration, evaluation and exploitation.

NOMENCLATURE

The grades of metals contained in the ore fed to a mill are termed *mill head grades* or *mill heads*. In the mill, ore is crushed, ground, and then treated by various processes to separate and concentrate the valuable minerals. The end products from the mill are *concentrates* and *tailings*. Concentrates contain economic metal(s) at grades higher than those in the mill feed; tailings contain the remainder of the treated mill feed. The most important process in producing concentrates is differential flotation, which takes advantage of the metallic surface properties of most sulphides and native metals. Consequently, most oxide, hydroxide, carbonate and silicate minerals such as chrysocolla (copper), gahnite (zinc), nickeliferous olivine (nickel), and even some sulphides (such as the nickel mineral, violarite) cannot be recovered by conventional flotation techniques. The milling of a polymetallic ore generally results in the production of more than one concentrate. A *bulk concentrate* is one which contains significant quantities of more than one economic metal. The *mill recovery* of a metal equals the amount of metal reporting to the concentrates, as a proportion of the total amount in the mill heads.

Concentrates are shipped to smelters and/or refineries, where the contained metals are extracted and purified. A smelter will not pay for all of the metal in the concentrates which it treats: metals for which the miner is paid are termed *payable* or *accountable* metals.

As an example, consider a lead-zinc-silver ore containing 100 g/t Ag. In the mill, 50% of the silver reports to the lead concentrate, 30% to the zinc concentrate and 20% to tailings. The lead smelter pays for 85% of the silver in the lead concentrate, but the zinc refinery gives no silver credits. Therefore:

contained silver = 100 g/t Ag

recovered silver = 80 g/t Ag

(*i.e.*, 100 g/t Ag × (50% + 30%))

payable silver = 42.5 g/t Ag

(*i.e.*, (100 g/t Ag × 50% × 85%)).

The *Net Smelter Return* of a tonne of ore is equal to the proceeds from the sale of mineral products derived from that ore, after deducting all related charges incurred outside the mine property. Such charges include concentrate transportation costs, smelting, refining, insurance and marketing costs.

The *gross value* of a tonne of ore is the value of the economic metals contained in the ore.

We define the *mine netback* of a tonne of ore as the ratio of the NSR to the gross value.

NSRs AND MINE NETBACKS FOR TYPICAL CONCENTRATES

In terms of total world production, the three most important types of base metal concentrates are copper, lead and zinc concentrates. Because of their relevance to Canada, bulk lead-zinc, molybdenum, nickel, bulk nickel-copper, platinum group metals (PGM) and tin concentrates are also discussed below.

Copper in Copper Concentrates

The most common copper ore mineral is chalcopyrite, which also is the major copper mineral in most copper concentrates. The average grade of the 32 Canadian copper concentrates listed in the *Mining Sourcebook* (1985 to 1990 editions, published annually by the *Canadian Mining Journal*) was 23.9% Cu. The concentrates from the Bethlehem and Lornex mills at Highland Valley, British Columbia, graded 40% Cu and were the only ones to exceed the 34.6% Cu content of stoichiometric chalcopyrite. Bornite, stoichiometrically 63.3% Cu, is an important ore mineral at Highland Valley.

A mine shipping copper concentrate to a smelter typically faces a variety of off-site costs or deductions:

(a) *Payable* copper, usually 1.0 to 1.4 "units" less than the copper contained in the concentrate.

(A "unit" is one percentage point. Thus, a 1.0 unit deduction in a concentrate grading 23.9% Cu would mean that (23.9–1.0)/23.9, or 95.8% of the contained copper would be paid for.)

A 1.0 unit deduction is characteristic of low-grade concentrates (say, 20% Cu); higher-grade concentrates are usually subject to higher deductions (*e.g.*, Reynard, 1991).

(b) *Smelting fees*, expressed as dollars per dry tonne of concentrate.

(c) *Penalties*, imposed by the smelter if the concentrate contains unduly high concentrations of certain substances.

(d) *Refining charges*, expressed as cents per pound of payable copper.

(e) *Freight charges*, expressed as dollars per wet tonne of concentrate.

(f) *Insurance costs, marketing expenses* and *physical losses of concentrate* during transportation.

Copper concentrate sales contracts may allow smelters to participate in some of the benefits should copper prices rise. In a typical *price participation* arrangement, the copper refining charge increases by a percentage of the amount by which the copper price exceeds a negotiated trigger price. Occasionally, contracts also provide for lower refining charges when the copper price falls below the trigger price (Reynard, 1991).

When expressed per pound of payable copper, deductions (a), (b) and (e) decline as the grade of the concentrate increases. Thus, all else being equal, an ore grading 1.5% Cu contained in bornite is more valuable than an ore with the same grade contained in chalcopyrite, because bornite makes a higher grade concentrate than does chalcopyrite.

In all of the examples of NSR calculations which follow, dollar amounts are in US dollars and it is assumed that: the deposits are reasonably accessible, with a freight charge of $35/tonne of wet concentrate; the concentrates contain 7.0% moisture; a standard deduction of 2.0% of the value of payable metals covers item (f) above; and that the concentrates contain low concentrations of deleterious materials.

NSR Calculation for Copper

Copper concentrate terms fluctuate depending on the supply/demand balance in the concentrate market. A typical 1990 copper schedule was: 100% of the contained copper payable after a 1.0 unit deduction; a smelting fee of $70/dry tonne; and a refining fee of $0.09/lb with price participation of 10% of the amount by which the price of copper exceeded $0.90/lb. With a copper price at the 1990 average spot level ($1.21/lb) and a grade of 23.9% Cu, a dry tonne of typical copper concentrate *contained* 2204.6 × 23.9/100 × $1.21 = $638 worth of copper. However, *payable* copper was worth 2204.6 × (23.9–1.0) /100 × $1.21 = $611/tonne. After our 2% allowance for insurance, marketing and metal losses during transportation, *net payable* copper was $599/tonne. Smelting charges were $70/dry tonne of concentrate, the refining charge was 2204.6 × (23.9–1.0) /100 × (0.09+0.031) = $61/dry tonne, and freight charges were $38/dry tonne. Because some of the concentrate is lost en route, the miner's smelting and refining charges on a net payable basis were slightly lower than as shown above: to account for this, we assume a 2% reduction in the refining charge, to $60/tonne. *NSR*, therefore, was $599 – $70 – $60 – $38 = $431/tonne of concentrate, representing 67.6% of the value of the copper in the copper concentrate.

In the 30 operations cited in the *Mining Sourcebook* (1985–1990 editions), an average of 84.7% of the copper contained in the mill feed was recovered to copper concentrates. Applying this mill recovery to the above example, the typical *mine netback* was 67.6% × 84.7% = 57% of the gross value of the copper contained in the ore. Column A of Table 1 summarizes the foregoing terms and calculations.

Lead in Lead Concentrates

Lead treatment charges may be expressed, as in the case of copper, as a smelter charge (per tonne of concentrate) plus a refining charge (per pound of payable metal). Alternatively, a smelter might quote a single charge for treatment (smelting plus refining) per tonne of concentrate. Lead smelters usually participate in 15%-25% of any upward or downward moves in lead prices. Price participation works as follows: if the price of lead were to rise by 10% above a reference level and if the smelter's price participation were set at 20%, the smelting charge per tonne would rise by 2%.

Column B in Table 1 summarizes the lead treatment terms prevalent in 1990 and the NSR calculation. As shown, a typical netback for a Canadian lead mine was 35% of the gross value of the lead in its ore.

Zinc in Zinc Concentrates

Some facilities are able to produce refined zinc directly from concentrates. Hence, treatment fees for zinc are commonly reported as a single figure per tonne of concentrate, rather than as a separate smelting charge and a refining charge. As was the case with copper, treatment fees decline (when expressed per pound of payable zinc) as the grade of the concentrate increases. Furthermore, zinc plants penalize zinc concentrates which are high in iron. Thus, all else being equal, an ore grading 10% Zn, with the zinc contained in iron-poor sphalerite, is more valuable than an ore grading 10% Zn, with the zinc contained in iron-rich sphalerite.

Like lead smelters, zinc plants generally participate in moves in metal prices. One such arrangement is shown in Column C of Table 1. Years ago, zinc plants recovered only 85% of the zinc in the concentrates. In comparison, the recovery at a modern zinc plant is about 93%. However, it is still standard practice for the plant to pay only 85% of the contained zinc: the difference of 8% is considered to be part of the smelting fee.

Column C in Table 1 summarizes the 1990 zinc treatment terms and the NSR calculation. Typical Canadian zinc ores yielded a 43% mine netback.

Gold and Silver in Copper, Lead and Zinc Concentrates

Lemieux *et al.* (1989) have estimated that only 49% of the gold contained in Canada's total reserves of base metal ores will be recovered to concentrates. It is difficult to justify this conclusion — or its extension to silver — because few operations publish the distribution of precious metals among their different mill products. Two operations which do publish these data are the Trout Lake mine in Manitoba (Healey and Petruk, 1990; *Mining Sourcebook*, 1985 to 1990 editions), and the Brunswick #12 mine in New Brunswick (Brunswick Mining and Smelting Ltd., 1990). The Trout Lake data show that in 1989, 74.5% of the contained gold was recovered: 59.2% to copper concentrate and 15.3% to zinc concentrate. Brunswick's figures show that in 1989, mill heads graded 109.5 g/t Ag, and that 56.3% of this amount was recovered: 23.2% to lead concentrate, 26.4% to copper concentrate and 6.7% to bulk concentrate.

Precious metals in base metal concentrates are subject to a series of charges and deductions by smelters and refineries. For example, we estimate that about 87% of the gold recovered at Trout Lake would have been payable, and that gold refining charges in 1989 were about 1.3% of the gold price. Hence, the mine netback on the gold content of this ore would have been about 74.5% × 87% × (100%–1.3%) = 64%. Similarly, at Brunswick, about 92% of the recovered silver would have been payable. Since silver refining charges in 1989 were about 7.5% of the silver price, the mine netback on the silver content of Brunswick's ore would have been about 56.3% × 92% × (100%–7.5%) = 48%.

A typical schedule for gold in a *copper* or *lead concentrate* would provide that the miner be paid the lesser of (a) 90% of the contained gold, or (b) the gold content of the concentrate less 1.0 g/t Au. A typical silver payment schedule for such concentrates would be 95% of the silver content of the concentrate, less 25 g/t Ag. For a *zinc concentrate*, payment is usually for the lesser of either (a) 80% of the contained gold and 90% of the contained silver, or (b) the precious metal content of the concentrate less a deduction of 1.2 g/t Au and 110 g/t Ag. Both gold and silver are subject to refining charges (currently about $0.30–$0.40/oz for silver and $5.00–$8.00/oz for gold).

These schedules have two important implications:

(1) Precious metals are more valuable if they report to copper or lead concentrates than to zinc concentrates.

(2) A deposit in which the precious metals all report to a single concentrate is more valuable than a similar deposit in which the precious metals report to several concentrates.

Thus, an important task for an economic geologist is to evaluate the habitat of precious metals in a potential orebody (Berubé, 1982; Gasparrini, 1980; Petruk, 1989). For example, inclusions of gold in chalcopyrite are worth more than inclusions of gold in sphalerite.

Other Constituents of Copper, Lead and Zinc Concentrates

The owner of a metallurgically complex orebody must do a lot of comparison shopping because smelters vary widely in their treatment terms for other constituents of concentrates. One smelter, for example, may pay for the lead and zinc contained in a copper concentrate, minus 3 units. For the same concentrate, another smelter may charge a penalty of $2.50 per unit for the contained lead and zinc beyond a deduction of 4 units of combined lead and zinc. Furthermore, when cadmium prices are low (*e.g.*, below $1.00/lb), a smelter might penalize a miner for the cadmium in zinc concentrate yet, when cadmium prices are high (*e.g.*, above $3.00/lb), the same smelter might pay a credit for the same cadmium in the same zinc concentrate.

Other than for some special bulk concentrates (see below), it is a good approximation to assume that copper, lead and zinc will have no value if they report to concentrates other than their respective concentrates. "Dirty" constituents — especially antimony, bismuth, arsenic, fluorine, cadmium, mercury, selenium, tellurium and thallium — generally result in penalties. Typically, such penalties are in the order of $10–$40 per tonne per unit of antimony, arsenic or bismuth. For mercury, penalties are hundreds of dollars per tonne per unit. Minimum concentrations, below which no penalties are charged, range from 0% to 3% for different materials and different smelters. High concentrations of iron, water or silica may also attract penalties.

The Equity Silver and Samatosum operations in British Columbia and the Ok Tedi mine in Papua New Guinea illustrate the importance of considering penalty charges in assessing the economic potential of a deposit. At Equity Silver, 78% of the antimony (Sb) and 21% of the arsenic (As) in the Southern Tail zone reported to the copper-silver-gold concentrate. As a result, this concentrate graded 7.0% Sb and 4.0% As and was judged to be unsaleable. Equity Silver constructed a leach plant which, with varying degrees of success, reduced the grades to 0.8% Sb and 2.0% As. Although subject to penalties, this material was acceptable to a Japanese smelter which had the right to reject material containing over 5.0% Sb. After mining the Southern Tail zone, Equity Silver moved on to the Main zone. Concentrates produced from the Main zone averaged 3.0%–3.5% Sb and 0.30%–0.35% As. It was more economical to ship unleached concentrates from the Main zone than to continue operating the leach plant (Dayton, 1982; Edwards, 1985).

There are two main mineralized zones at the Samatosum deposit: the Discovery zone and the Silver zone. The Discovery zone has reported reserves of 240,000 tonnes grading 72 g/t Ag plus other metals. It also contains 6.0% As. Although such concentrations of arsenic might be worth special treatment in a larger deposit (as was the case with Equity Silver's Southern Tail zone), they will generally destroy the economics of a small deposit. Indeed, the Discovery zone has not been mined. However, the Silver zone is in production because, although the copper-silver-gold concentrate is rich in both Sb and As, the ore grade and tonnage are sufficiently high to make this zone commercial.

Copper-gold ores at Ok Tedi are of two main types, porphyry and skarn. All of Ok Tedi's ore is rich in fluorine; the skarn ore especially so (3,000 ppm F is typical). However, most smelters impose penalties on copper concentrates containing more than 500 ppm F. By carefully blending mill feed, Ok Tedi's operators have been generally successful in maintaining fluorine levels in the concentrates at just below 500 ppm F.

Bulk Lead-Zinc Concentrates

A bulk concentrate is one containing significant quantities of more than one economic metal. When polymetallic orebodies are small or when they have high grades but complex mineralogy, it may be cheaper to ship a bulk concentrate and suffer a lower smelter return than to install and operate the equipment necessary to provide a cleaner separation of minerals. A "standard" bulk concentrate, which would be cited in establishing a base price in smelter contracts, would contain about 30% Zn and 15% Pb.

Only one producer of bulk lead-zinc concentrates is listed in the *Mining Sourcebook* (1985 to 1990 editions): Brunswick Mining and Smelting Ltd., which produces a zinc-lead-silver bulk concentrate in addition to copper, lead and zinc concentrates. In 1989, the bulk concentrate averaged 33.2% Zn, 19.8% Pb and 390 g/t Ag (Brunswick Mining and Smelting Ltd., 1990). This concentrate is sold to a European smelter. Payable levels of metals are as follows: zinc, 85% or less 7 units; lead, 95% or less 3 units; silver, 95% or less 100 g/t (M. Power, Noranda Treasury, pers. comm., 1988). In all cases, the appropriate deduction is the one which results in the least amount payable.

In recent years, bulk lead-zinc concentrates have become more marketable. This development has enhanced the prospects for the McArthur River zinc-lead-silver project in Northern Territory, Australia (Gerrard, 1990), and has also benefited the Red Dog mine in Alaska. Red Dog produces a zinc concentrate, a lead-silver concentrate and a bulk lead-zinc-silver concentrate (Millbank, 1990).

Column D in Table 1 presents 1990 treatment terms and the mine netback for a hypothetical producer of a bulk lead-zinc concentrate, assuming a recovery to concentrate of 85% for both lead and zinc. Note that smelting charges (per tonne of concentrate) are similar to those for zinc concentrates. In this case, the mine netback is 33% of the gross value of the lead and zinc in the ore.

Molybdenum Concentrates

Concentrates are priced on their contained molybdenum, but at a discount to the price quoted for molybdic oxide. This discount,

Table 1 **Typical 1990 treatment terms and NSR calculations for representative Canadian concentrates.**

Concentrate Type	A Cu in copper concentrates*	B Pb in lead concentrates*	C Zn in zinc concentrates*	D Pb and Zn in bulk lead-zinc concentrates**	E Mo in molybdenum concentrates*
Recovery to concentrate	84.7%	73.3%	83.8%	85%	70%
Grade of concentrate	23.9% Cu	46.0% Pb	51.0% Zn	19.8% Pb 33.2% Zn	53.5% Mo
Compare with ideal stoichio-metric grades	chalcopyrite: 34.6% Cu	galena: 86.0% Pb	iron-free sphalerite: 67.0% Zn	galena: 86.0% Pb iron-free sphalerite: 67.0% Zn	molybdenite: 60.0% Mo
Deduction	1.0 units	1.0 units	none	3.0 units (Pb) 7.0 units (Zn)	none
Proportion of con-tained metal which is payable after the above deduction	100%	94%	85%	100% (Pb,Zn)	100%
Basic treatment/smel-ter charge, $/DMT	$70	$80	$210	$210	N.A.
Escalator in basic treatment/smelting charges, $/DMT of concentrate	0	0	$5/DMT for 1 ¢/lb move above $0.68/lb or −$1.70/DMT for each 1 ¢/lb move below $0.68/lb	for Zn, similar to zinc concentrates	0
Basic refining fee, cents per pound or ounce of refined metal	9.0¢/lb	8.5¢/lb	0.0	8.5¢/lb (Pb) 0.0¢/lb (Zn)	32.5¢/lb
Escalator in refining fees, cents/lb of refined metal	1.0 ¢/lb for each 10¢/lb over 90¢	20% of any move from $0.45/lb	0.0	for Pb, similar to lead concentrates	N.A.
Value of metal con-tained in concentrate, $/DMT	$638	$466	$765	$699	$3,361
Net payable metal in concentrate, $/DMT	$599	$420	$637	$552	$3,294
NSR, $/DMT of concentrate	$431	$219	$389	$273	$2,880
Mine netback	57%	35%	43%	33%	60%

Notes:

* Recoveries and concentrate grades are averages of figures presented in the *Mining Sourcebook* (1985 to 1990 editions, published annually by the *Canadian Mining Journal*).

** Based on figures for Brunswick Mining and Smelting Ltd.'s bulk concentrates excluding its precious metals, and on a hypothetical rate of recovery.

- Dollar figures are US dollars
- The above calculations assume: concentrates with 7% moisture; freight cost $35/WMT (wet metric tonne); no penalties.
- DMT = dry metric tonne
- N.A. = not applicable or not available

Table 1 *cont'd*

Concentrate Type	F Ni in nickel concentrates***	G Ni and Cu in bulk nickel-copper concentrates*	H PGM and Au in bulk nickel-copper concentrates**** (incremental)	I (H + G) *i.e.*, Ni, Cu, PGM and Au in bulk nickel-copper concentrates****	J PGM, Au, Ni and Cu in bulk PGM-sulphide concentrates****
Recovery to concentrate	86.5% Ni	89% Ni 95.5% Cu	82% Au, Pd, Pt & Rh	89% Ni, 95.5% Cu, 82% Au, Pd, Pt & Rh	82% Au, Cu, Ni, Pd, Pt & Rh
Grade of concentrate	17.0% Ni	8.9% Ni 13.5% Cu	1.2 g/t Au 3.3 g/t Pt 3.4 g/t Pd 0.33 g/t Rh	8.9% Ni 13.5% Cu 1.2 g/t Au 3.3 g/t Pt 3.4 g/t Pd 0.33 g/t Rh	2.7% Ni 2.7% Cu 4.8 g/t Au 24.0 g/t Pt 86.4 g/t Pd 4.8 g/t Rh
Compare with ideal stoichio-metric grades	pentlandite: 34.0% Ni	N.A.	N.A.	N.A.	N.A.
Deduction	N.A.	none	none	none	none
Proportion of con-tained metal which is payable after the above deduction	N.A.	90% (Ni,Cu)	62% (Au, Pt, Pd, Rh)	90% (Ni, Cu) 62% (Au, Pt, Pd, Rh)	90% (Ni, Cu, Au, Pt, Pd, Rh)
Basic treatment/smel-ter charge, $/DMT	N.A	$145	none (carried by Ni, Cu)	$145	$165
Escalator in basic treat-ment/smelting charges, $/DMT of concentrate	N.A.	0	N.A.	N.A.	N.A.
Basic refining fee, cents per pound or ounce of refined metal	N.A.	50¢/lb (Ni) 12¢/lb (Cu)	800¢/oz (Au) 2500¢/oz (Pt & Pd) 7000¢/oz (Rh)	50¢/lb (Ni) 12¢/lb (Cu) 800¢/oz (Au) 2500¢/oz (Pt & Pd) 7000¢/oz (Rh)	85¢/lb (Ni) 20¢/lb (Cu) 1500¢/oz (Pt & Pd) 4500¢/oz (Rb)
Escalator in refining fees, cents/lb of refined metal	N.A.	N.A.	N.A.	N.A.	N.A.
Value of metal con-tained in concentrate, $/DMT	$1,525	$1,159	$117	$1,276	$1,607
Net payable metal in concentrate, $/DMT	N.A.	$1,045	$71	$1,116	$1,418
NSR, $/DMT of concentrate	$987	$723	$64	$787	$1094
Mine netback	56%	57%	46%	56%	56%

Notes, cont'd:

*** Based on data in Timmins Nickel Inc. (1989).
**** Hypothetical, but representative, examples.

• Metal prices used in the above calculations are average price levels in 1990, as follows:

Copper (London Metal Exchange (LME) spot): $1.21/lb
Lead (North American producer price): $0.46/lb
Molybdenum (*Metals Week* molybdic oxide price): $2.85/lb
Nickel (LME spot): $4.07/lb
Zinc (European Producer Price): $0.68/lb
Gold (London final fix): $383/oz
Palladium (London spot): $115/oz
Platinum (London spot): $472/oz
Rhodium (London spot): $3,565/oz

quoted for molybdic oxide. This discount, which reflects treatment charges, averaged $0.30-$0.35 per pound in 1990.

Most molybdenum smelters do not pay for other constituents. However, some may give credit for rhenium, and precious metals can be leached from molybdenum concentrates at the mine site, as at Island Copper (Lastra, 1986).

As shown in Column E of Table 1, typical mine netbacks in 1990 for Canadian molybdenum ores were 60%.

Nickel Concentrates and Bulk Nickel-Copper Concentrates

Two nickel concentrates, one from Namew Lake and one from Thompson, Manitoba, and five bulk nickel-copper concentrates from Sudbury, Ontario and Thompson are listed in the *Mining Sourcebook* (1985 to 1990 editions). The nickel concentrates average 22% Ni, and mill recovery at Namew Lake is 90%. The bulk concentrates average 8.9% Ni and 13.5% Cu; mill recoveries average 89.0% for Ni and 95.5% for Cu.

Most nickel and nickel-copper concentrates are smelted and refined by the companies which produce them. As a result, trade in nickel concentrates and bulk nickel-copper concentrates is limited and shrouded in secrecy, and toll smelter/refinery terms are not publicly available.

Operations at the Redstone nickel mine, near Timmins, Ontario, provide some insight into the evaluation of nickel concentrates. Redstone was opened in 1989. Between June 1989 and February 1990, from ore grading 2.36% Ni (plus minor cobalt), a concentrate grading 17% Ni was produced with a mill recovery of 86.5% (Timmins Nickel, Inc., 1990). The concentrate was sent to Sherritt Gordon's plant at Fort Saskatchewan, Alberta. This plant combines the functions of both a smelter and a refinery, and can accept matte or concentrate as feed. Sherritt's treatment fees are linked to the nickel price (Lamphier, 1988). No detailed terms have been made publicly available, but the mine operator has disclosed its anticipated "realization" (or NSR per tonne) for nickel prices between $2.50/lb and $6.50/lb, based on its long-term refining agreement with Sherritt (Timmins Nickel, Inc., 1989), as set out in Table 2. Corresponding calculations of gross value/tonne ore and mine netback are also shown in Table 2.

At the average 1990 nickel price of $4.07/lb, the netback would have been about 56% of the value of the nickel contained in the ore (see Column F of Table 1). Because Sherritt was short of feed in the late 1980s, one can assume that a concentrate producer would have found it difficult to negotiate a contract more favourable than this one.

For a hypothetical producer of bulk nickel-copper concentrates from mill-heads of 1.2% Ni and 1.7% Cu, Column G of Table 1 presents smelting and refining terms which are considered to be a reasonable estimate of what might be obtainable. The resultant netback to the nickel-copper mine is 57%.

PGM and Gold in Nickel-Copper Concentrates

"PGM" refers to the six platinum group metals, only three of which have significant markets: platinum (Pt); palladium (Pd); and rhodium (Rh). Canadian production of PGM is principally from Sudbury as a by-product of nickel-copper mining.

PGM and gold are recovered, along with the nickel and copper, in bulk sulphide flotation concentrates. In Canada and elsewhere typical recoveries are 80%–85% of the by-product PGM plus gold. The concentrates are smelted and converted to high-grade mattes, from which nickel and copper are extracted and refined. The residue is refined into gold and individual PGM constituents in a lengthy and involved chemical process.

In the preceding section, a hypothetical, but representative, Canadian ore grading 1.2% Ni and 1.7% Cu was introduced. Such an ore would also contain about 1.2 g/t PGM plus gold, comprising about 40% Pt, 41% Pd, 4% Rh and 15% Au. Column H in Table 1 outlines representative treatment terms, and the resulting incremental NSR and netback, for these by-product PGM and gold values. Column I is the sum of the two preceding columns. Note that, although the inclusion of the PGM plus gold values enhances the total NSR, the overall mine netback is essentially unchanged.

Primary PGM Concentrates

In Canada, there are no mines in which PGM are the chief economic metals. In fact, only one primary PGM mine, the Stillwater mine in Montana, exists outside South Africa.

Depending on a PGM deposit's particular mix of economic metals (usually the six PGM, plus one or more of Ni, Cu, Au, Ag and Co), widely different treatment terms are possible. Hence, generalizations are difficult. Because of secrecy in the industry, there are no published data. Column J of Table 1 shows simplified, but representative, toll treatment terms which might be obtained for a bulk sulphide concentrate that is relatively rich in PGM as compared to the previously discussed by-product PGM concentrate. Assumed ore grade is 8.56 g/t PGM plus gold (comprising 20% Pt, 72% Pd, 4% Rh and 4% Au), 2.7% Ni and 2.7% Cu.

This example illustrates an important aspect of PGM: mine netbacks are more like those of base metals than gold. The relatively low netbacks in PGM operations (56% in the above example) as compared to gold operations (where mine netbacks of 90% are typical) result from comparatively high metal losses in processing of the PGM, particularly in the concentrating stage; and from treatment costs and deductions which are high because of the complexity of PGM refining processes.

It is common practice to express PGM grades in terms of "gold equivalents". As the example above illustrates, it is also a misleading practice!

Tin Concentrates

In some polymetallic deposits, tin, if in the form of cassiterite, may be both recoverable and payable. Since recovery of cassiterite is usually by simple gravity concentration, the most important consideration in evaluating the tin potential of a polymetallic deposit is the size-frequency distribution of the cassiterite grains. To emphasize the tin potential of massive sulphide deposits, note that Somincor's giant Neves Corvo polymetallic massive sulphide deposit in Portugal has become a major producer of tin as well as copper, zinc, lead and silver, following the installation of tin circuits in the mill.

In Canada, only three mines active in 1990 were (or had been) producers of tin: Sullivan, British Columbia; Kidd Creek, Ontario; and East Kemptville, Nova Scotia. All are polymetallic. Tin recoveries have ranged from 20% to 63% (Hamilton *et al.*, 1982; Rio Algom Limited, 1990). We have been unable to find smelter terms for tin concentrates.

Table 2 Netback of Redstone ore at different nickel prices.

Ni Price ($/lb)	NSR ($/tonne ore) [1]	Gross Value ($/tonne ore) [2]	Netback
6.50	262	417	63%
5.50	213	353	60%
4.50	166	288	58%
3.50	121	225	54%
2.50	79	161	49%

[1] Based on an exchange rate of CDN $1.00 = US $0.846

[2] Based on Timmins Nickel, Inc.'s expectation of a 2.91% Ni mill head grade.

APPLICATIONS OF NSR MODELS

Typical Grades and Recoveries of Copper, Lead and Zinc Concentrates

Table 3 shows the results of a statistical analysis of the relationship between mill head grade and mill recoveries, and between mill head grade and concentrate grades, for the data on copper, lead and zinc concentrates presented in the *Mining Sourcebook* (1985 to 1990 editions).

The table shows that, for copper, lead and zinc, both mill recoveries and concentrate grades tend to improve with increasing head grades. Furthermore, the presence of copper appears to have an adverse effect on the metallurgical characteristics of zinc and lead in mill feed, just as zinc appears to have an adverse effect on the characteristics of copper in mill feed. However, the presence of lead appears to have no significant effect on either the recovery of copper or on the grades of copper concentrates. Moreover, lead and zinc each have mixed influences on the metallurgical characteristics of the other. The coefficients of determination imply that head grades account for only about 40% of the variations in recoveries and concentrate grades; other factors account for the remaining 60%. Nevertheless, because the ranges of variation in recoveries and concentrate grades are relatively small, Table 3 can provide a useful first approximation as to the possible milling characteristics of a deposit with given mill feed grades.

Worksheet for Constructing an NSR Model

A spreadsheet NSR model for a typical polymetallic deposit is presented in Appendix 1. The ore grades used in this model are the mean Cu, Pb, Zn, Ag and Au grades of 432 "massive sulphide deposits related to marine felsic to mafic extrusive rocks" (Cox and Singer, 1986). Where possible, recovery rates and concentrate grades are derived from the relationships shown in Table 3; smelting and refining fees are approximate spot levels as of 1990; other smelting and refining terms represent a distillation of a large number of smelter schedules; metal prices and exchange rates are at their average 1990 levels. Other figures are arbitrary, but realistic, estimates.

Uses of NSR Models

As noted in the introductory section, an economic geologist can make use of NSR calculations at the exploration stage, at the evaluation stage and at the exploitation stage.

Exploration. The major use of NSR models in exploration is to provide a common basis for comparison of the relative merits of different polymetallic drill-hole intersections. For example, on September 20, 1989, Aur Resources Inc. published results from holes 53B and 58 on the Louvicourt project, Quebec. Two of the intersections were:

Hole:	#53B	#58
Width (ft)	56.2	21.5
Cu (%)	2.88	0.47
Zn (%)	0.02	6.92
Au (oz/ton)	0.03	0.02
Ag (oz/ton)	0.03	0.02

The relative merits of these two drill intercepts are not immediately evident. It would be helpful to those exploring the deposit, and those attempting to evaluate it, to use a common denominator to compare the assays. Based on the NSR model in Appendix 1, and metal prices as at September 20, 1989, the NSR of the intersection from hole 53B was $57.27 per tonne and that from hole 58 was $64.55 per tonne. In general, we suggest that companies presenting assays from polymetallic deposits show the estimated NSR per tonne for each assay interval, and note the assumptions behind these estimates.

Even in the initial stages of an exploration project, it is useful to estimate NSRs per tonne for polymetallic assays. The NSR model used at this stage will, of necessity, contain many "ballpark" assumptions. However, the process of making these assumptions will force the geologist to consider economic factors which might otherwise be overlooked.

Table 3 Relationships between mill heads, recoveries, and concentrate grades. These figures are based on regression analysis of figures compiled from the *Mining Sourcebook* (1985 to 1990 editions, published annually by the *Canadian Mining Journal*). For operations listed in more than one issue of the *Mining Sourcebook*, the figures used are those from the most recent entry.

Copper Concentrates			
Recovery of copper to concentrate (%)	= 83.5	+ (2.92 × Cu head grade (%)) – (1.07 × Zn heads/Cu heads)	($\bar{r}^2$=0.49; n=30)
Concentrate grade (%)	= 23.9	+ (0.36 × head grade (%)) – (0.19 × Zn heads/Cu heads)	($\bar{r}^2$=0.03; n=32)
Lead Concentrates			
Recovery of lead to concentrate (%)	= 76.4	+ (2.94 × Pb head grade (%)) – (0.77 × Zn head grade (%)) – (6.81 × Cu head grade (%))	($\bar{r}^2$=0.42; n=13)
Concentrate grade (%)	= 22.1	+ (3.84 × Pb head grade (%)) + (2.60 × Zn head grade (%)) – (2.66 × Cu head grade (%))	($\bar{r}^2$=0.48; n=13)
Zinc Concentrates			
Recovery of zinc to concentrate (%)	= 79.2	+ (1.19 × Zn head grade (%)) + (1.15 × Pb heads/Zn heads) – (5.62 × Cu heads/Zn heads)	($\bar{r}^2$=0.57; n=22)
Concentrate grade (%)	= 49.0	+ (0.85 × Zn head grade (%)) – (12.14 × Pb heads/Zn heads) – (0.42 × Cu heads/Zn heads)	($\bar{r}^2$=0.38; n=23)

$\bar{r}^2$ Coefficient of Determination, a measure of the degree to which the above equations explain the variations in recovery or grade. For example, if $\bar{r}^2$=0.20, 20% of the variation is explained and 80% is unexplained.

n number of operations cited

Evaluation. NSR models are an essential component of an economic evaluation of a mineral deposit. NSRs are also useful in technical evaluations and in mine planning and design. For example, in evaluating the Grevet B-M-J deposit in Quebec, VSM Exploration Inc. has defined cut-off grades in terms of NSRs (VSM Exploration Inc., 1990). One objection to VSM's approach is that too many assumptions are required. Our response is that this is precisely why the exercise is so valuable: it highlights exactly not only where one's knowledge is deficient, but also how sensitive the results are to the assumptions one makes. Furthermore, by estimating how operating profits (*i.e.*, total NSRs minus all operating and development costs) vary at different cut-off grades, an evaluator can estimate optimum cut-off grades and production rates for a project. A similar use of NSRs is in determining the optimum mill recovery rate, given that when the mill recovery increases, there is generally a decline in the grade of the resulting concentrate.

The use of NSR models encourages close co-operation between geologists and metallurgists in choosing representative ore samples for metallurgical test work in assessing the degree that copper, lead and zinc may report to other than their respective concentrates, and in detecting the presence of deleterious materials. According to John Carrington, then Senior Vice-President of Minnova Inc. (pers. comm., 1990), such basic evaluations are "essential but even today are often overlooked".

Exploitation. As pointed out by Lane (1988), mining companies commonly define the limits of their ore bodies according to arbitrary, unchanging cut- off grades. Typical justifications for this approach include:

"We have always worked to 0.3%", or
"Head Office decided 5% combined metals some years ago", or
"I think several cut-offs were examined in the feasibility study and 1% seemed best".

NSR models help provide a more realistic approach to determining cut-off grades. The boundary of an ore deposit is defined as the surface which separates material which could be extracted at a profit from material whose value (NSR) would exceed the cash expenditures required to develop, extract and process this material. Because prices and costs are continually changing, this surface is always moving. As Lafleur (1988) pointed out, "a service raise was developed in waste in 1986 at Mobrun. It is now in ore."

A good example of the use of NSR models at the exploitation level is given by Audrey Resources Inc. In 1988, the company decided that the best approach to grade control at its polymetallic Mobrun mine was to convert assays into NSRs (Lafleur, 1988). All material with an NSR greater than the expected cost of production is treated as ore. As Lafleur stated, "it is standard practice in polymetallic mines to convert grades into one metal equivalent ... the problem is that once the transformation is done, people tend to forget that the conversion is based on fluctuating metal prices and recoveries, and that the dollar is the only true common denominator for any grade conversion."

Audrey Resources' approach, and that of VSM (as outlined previously), have been criticized as being too unstable, requiring constant revisions of mine plans. Border (1991) has addressed this objection: "industry resistance to the optimisation of cut-off grades during the seventies and eighties was partly caused by a lack of widespread computer facilities on mine sites — it was unrealistic to expect operating personnel to make regular changes to cut-off grade when every change involved many man-weeks of laborious manual redrafting, geological interpretation and

Table 4 Mine netbacks for various Cu, Pb, Zn, Au and Ag ore types in different economic environments.

Ore type	Mill products	Mine Netback (%) 1990*	1986**	1980***	Range
Typical massive sulphide [1]	Cu, Pb & Zn concentrates	43.8	39.2	42.4	4.6
Brunswick #12 mine Zn-Pb-Cu-Ag ore [2]	Cu, Pb & bulk concentrates	37.3	37.7	39.7	2.4
Highland Valley Copper Cu ore [3]	Cu concentrate	66.2	59.8	66.8	7.0
Polaris mine Zn-Pb ore [4]	Pb & Zn concentrates	53.2	51.6	54.8	3.2
Opemiska mine Cu-Ag-Au ore [4]	Cu concentrate	67.3	65.1	69.8	4.7
Winston Lake mine Zn-Cu ore [4]	Cu & Zn concentrates	47.0	48.0	50.7	3.7
	Range	30.0	27.4	30.1	

Notes:

* 1990 Case: commodity prices and treatment terms at their average levels in 1990.

** 1986 Case: commodity prices and treatment terms at their average levels of 1986 (when commodity prices were at their lowest levels in the decade).

*** 1980 Case: commodity prices and treatment terms at their average levels of 1980 (the peak year for commodity prices preceding the 1986 trough).

In all three cases, other assumptions are as in Appendix 1.

[1] mean ore grades are based on the assays of 432 "massive sulphide deposits related to marine felsic to mafic extrusive rocks" (Cox and Singer, 1986), as used in Appendix 1; mill recoveries and concentrate grades are based on the relationships in Table 3.

[2] 1989 grades and recoveries, Brunswick Mining and Smelting (1990).

[3] grades and recoveries from Canadian Mining Journal (1990); only the copper content of the ore was considered.

[4] grades and recoveries from Canadian Mining Journal (1990).

stope design. For example ... regular changes to cut-off grades at Mount Isa were impractical until very recently. Today, when virtually every mine has geological and mine planning data stored on computer, this objection is no longer valid."

The Equity Silver and Samatosum deposits, discussed earlier, both provide examples of the value of defining cut-off grades in terms of NSRs. In these deposits, two samples could have identical assays for Ag, Au, Cu, Pb and Zn, yet one could represent ore and the other, if unduly rich in As and Sb, could be waste.

Typical Mine Netbacks

One of the key questions in evaluating a mineral deposit is: "for how much of this stuff would a miner actually get paid?" As shown in Table 1, mine netbacks are substantially higher for copper, molybdenum, nickel and PGM ores than for lead or zinc ores. In 1990, typical netbacks for the former ranged between about 56% and 60% compared to a 33% to 43% range for the latter. Because of the many assumptions underlying the examples, the mine netbacks presented in Table 1 should only be used for "rule of thumb" purposes when specific data is unavailable for a particular project.

Based on data from the *Mining Sourcebook* (1985 to 1990 editions), the unweighted average gold recovery of 34 Canadian gold-only and gold-silver mines is 92.6%. Transportation, refining, marketing, and insurance costs are estimated to account for a deduction of 2–3 percentage points, suggesting that typical netbacks for such deposits are about 90%. This illustrates the danger in the common practice of presenting assays of polymetallic deposits in terms of "metal equivalents". For example, the use of "gold equivalents" implies that the netbacks of all metals in a polymetallic deposit are the same as those for gold in a gold-only deposit. Not only is this decidedly not true — as demonstrated by the mine netbacks in Table 1 — it is not even true for gold in a polymetallic deposit! (As pointed out earlier, netbacks for gold in polymetallic deposits are relatively low.)

Table 4 presents the mine netbacks of six different polymetallic ore types, based on the model of Appendix 1. Note that for comparative purposes, the netbacks in Table 4 have been calculated using a standard concentrate freight cost, as well as standard smelter and refinery terms, including typical penalties. Table 4 demonstrates how these netbacks might vary over an economic "cycle". The "1990 Case" is based on average commodity prices and treatment terms as of 1990. The "1986 Case" is based on average commodity prices and treatment terms in 1986 (the trough of the last base metal price "cycle") and the "1980 Case" on those of 1980 (which was the peak of the preceding "cycle").

The examples presented in Table 4 indicate that:

(a) Ores containing copper, but without lead or zinc, have higher netbacks than ores containing copper plus lead or zinc.
(b) Lead-zinc ore types which are "clean" (*i.e.*, the zinc and lead minerals are readily separable, and there are no penalties) can have higher netbacks than typical polymetallic massive sulphide ore types.
(c) During cyclical peaks in metal prices, netbacks are higher than during troughs.
(d) For any one ore type, netbacks varied by an average range of about 4 percentage points during the period 1980-1990.
(e) From one ore type to another, the netbacks in Table 4 vary by up to 30 percentage points: thus, ore type is a much more important determinant of the mine netback than is the economic environment.

Note that the 1990 netbacks for the operations in Table 4 tend to be higher than those for comparable ore types in Table 1. This is in part because of unusually favourable metallurgical conditions (*e.g.*, at Highland Valley Copper and Polaris) and in part due to gold and silver credits (*e.g.*, at Highland Valley Copper and Opemiska).

CONCLUSIONS

(1) The most important unit of measurement to the economic geologist is not percent, g/t, oz/ton or ppm: it is the dollar. The advent of cheap, readily available computer processing has greatly facilitated the use of this unit of measurement.

(2) For many purposes, the value of polymetallic deposits is best expressed by the Net Smelter Return per tonne. The NSR of an ore is equal to the revenues derived from the sale of the products of that ore, minus all off-property treatment and distribution charges. The NSR per tonne for a hand sample, for a piece of drill core, or for an entire deposit can be readily determined by the use of an NSR model.

(3) By providing a common denominator for polymetallic assays, NSRs are of use to economic geologists in exploration (comparing drill-hole assays in an exploration program), evaluation (preparing a feasibility study or in mine design) and exploitation (providing a basis for grade control in an operating mine).

(4) Constructing an NSR model forces the geologist to address a number of factors which determine whether or not a deposit might be economic. In addition to metal prices, exchange rates, and concentrate treatment and freight costs, NSRs are determined by ore grades, mill recoveries, concentrate grades and the presence or absence of deleterious minerals. These latter factors are mainly determined by mineralogy.

(5) "Mine Netback" is defined as the ratio of the NSR to the gross value of the metals contained in an ore. Typical netbacks, for any one of the polymetallic ore types examined, varied over a range of only about 4 percentage points over the last metal price "cycle". However, from one ore type to another, the range of netbacks was about 30 percentage points. Hence, in determining netbacks, mineralogical factors are far more important than economic conditions.

(6) Some examples of the mineralogical determinants of netbacks are as follows. Copper contained in bornite is more valuable than copper contained in chalcopyrite; zinc contained in iron-poor sphalerite is more valuable than zinc contained in iron-rich sphalerite; and gold in chalcopyrite is more valuable than gold in sphalerite. Furthermore, smelter charges and deductions are smaller, per dollar value of payable metal, for copper and nickel concentrates than for lead and zinc concentrates. As a result, netbacks tend to be higher for nickel and copper deposits than for zinc and lead deposits. Typical netbacks are approximately 56%–60% for copper, nickel, nickel-copper, nickel-copper-PGM-gold or molybdenum ores; 35%–53% for clean lead-zinc ores; and 33%–44% for complex polymetallic ores. (The presence of gold or silver in an ore will generally increase these percentages.) In comparison, netbacks from gold-only or gold-silver ores are typically 90%.

(7) A common technique in reporting polymetallic assays has been to re-cast them in the form of "gold equivalents". We strongly caution against this practice, because it implies that netbacks for polymetallic ores are similar to those for gold-only ores. This is emphatically not the case.

ACKNOWLEDGEMENTS

Our sincere thanks go to Messrs. M. Bouchard of Audrey Resources Inc., G. Bub of Black Hawk Mining Inc., Mark Leichnitz of Placer Dome Inc., John Carrington, now of Brunswick Mining and Smelting Ltd., David Carson of Noranda Exploration, Robert Horn of B.P. Canada and an anonymous reviewer for their help in greatly improving this paper.

REFERENCES

Berubé, M.A., 1982, La mineralogie appliquée au traitement des minerais: Canadian Institute of Mining and Metallurgy, Bulletin, November 1982, p. 121-130.

Border, S.N., 1991, Optimisation of cut-off grades during design of underground mines: Australasian Institute of Mining and Metallurgy, Bulletin, No. 5, September 1991, p. 14-20.

Brunswick Mining and Smelting Ltd., 1990, 1989 Annual Report.

Cox, D.P. and Singer, D.A., 1986, eds., Mineral Deposit Models: United States Geological Survey, Bulletin 1693, 379 p.

Dayton, S., 1982, Equity Silver on line with leach plant: Engineering and Mining Journal, January 1982, p. 78-83.

Edwards, R.R., 1985, The Equity leach plant: Canadian Institute of Mining and Metallurgy, Bulletin, v. 78, December 1985, p. 84-93.

Gasparrini, C., 1980, The role of the ore microscope and electron microprobe in the mining industry: Canadian Institute of Mining and Metallurgy, Bulletin, May 1980, p. 73-85.
Gerrard, T., 1990, MIM Holdings Limited — a major international mining house in the making: Potts West Trumbull and Co., Sydney, Australia.
Given, I.A., 1973, ed., SME Mining Engineering Handbook: Society of Mining Engineers of AIME, New York, p. 5-2.
Hamilton, J.M., Bishop, D.T., Morris, H.C. and Owens, O.E., 1982, Geology of the Sullivan Orebody, Kimberley, B.C., Canada, *in* Hutchinson, R.W., Spence, C.D. and Franklin, J.M., Precambrian Sulphide Deposits: Geological Association of Canada, Special Paper 25, p. 597-665.
Healy, R.E. and Petruk, W., 1990, Petrology of Au-Ag-Hg alloy and "invisible" gold in the Trout Lake massive sulphide deposit, Flin Flon, Manitoba: Canadian Mineralogist, v. 28, p. 189-206.
Howkins, J.B., 1988, Namew Lake nickel mine comes on-stream: Metal Bulletin Monthly, December 1988, p. 36-37.
Lafleur, P-J., 1988, Statistical geology: The Northern Miner Magazine, November 1988, p. 25-29.
Lamphier, G., 1988, Loss of Inco refinery deal "no disaster" for Sherritt Gordon: The Globe and Mail, Toronto, Ontario, August 2, p. B3.
Lane, K.F., 1988, The Economic Definition of Ore: Mining Journal Books Ltd., London, 149 p.
Lastra, M.R., 1986, Recovering gold from molybdenum concentrate at Island Copper: Canadian Institute of Mining and Metallurgy, Bulletin, v. 79, July 1986, p. 32.
Lemieux, A., Jen, L.-S., Bouchard, G. and Cranstone, D.A., 1989, Canadian mines: perspective from 1988: Energy, Mines and Resources Canada, Mineral Bulletin MR 221.
Millbank, P., 1990, "Top Dog": Metal Bulletin Monthly, June 1990, p. 10-11.
Mining Sourcebook, 1985 to 1990 editions, Canadian Mining Journal annual: Southam Business Publications.
Petruk, W., 1989, "Recent progress in mineralogical investigations related to gold recovery": Canadian Institute of Mining and Metallurgy, Bulletin, v. 82, November 1989, p. 37-40.
Reynard, G.F., 1991, Concentrates — the overlooked market?: paper presented to COPPER '91, August 1991, Ottawa, Ontario, 14 p.
Rio Algom Limited, 1990, 1989 Annual Report.
Roberts, R.G. and Sheahan, P.A., 1988, eds., Preface, *in* Ore Deposit Models: Geological Association of Canada, Geoscience Canada Reprint Series 3, p. v.
Timmins Nickel, Inc., 1989, Final Prospectus: August 16, 1989, p. 7.
Timmins Nickel, Inc., 1990, Interim report, nine months ended February 28, 1990.
VSM Exploration Inc., 1990, 1989 Annual Report.

Accepted, as revised, 16 December 1991.

Appendix 1 **Net Smelter Return Model for a typical polymetallic ore.**

	Figures to be input	Calculated figures	Labels
COPPER:			
ore grade, %	1.259		A
% recovery to Cu concentrate	84.8		B *
% recovery to Pb concentrate	3.0		C
LEAD:			
ore grade, %	0.747		D
% recovery to Cu concentrate	15.0		E
% recovery to Pb concentrate	67.9		F *
% recovery to Zn concentrate	10.0		G
ZINC:			
ore grade, %	2.807		H
% recovery to Pb concentrate	4.0		I
% recovery to Zn concentrate	80.3		J *
GOLD:			
ore grade, oz/short ton	0.023		K
% recovery to gravity circuit	0		L
% recovery to Cu concentrate	33.0		M
% recovery to Pb concentrate	12.0		N
% recovery to Zn concentrate	4.0		O
SILVER:			
ore grade, oz/short ton	0.840		P
% recovery to Cu concentrate	31.0		Q
% recovery to Pb concentrate	13.0		R
% recovery to Zn concentrate	5.0		S
Characteristics of Cu concentrate:			
Cu grade of dry concentrate, %	24.0		T*
DMT of concentrate/t of ore processed		0.0441	$U = (A \times B/T)/100$
Deduction (units of Cu)	1.06		V
Payable Cu, lb/t of ore		22.30	$W = A \times B \times 2204.6 \times (T - V)/T$
Lead deduction (units of Pb)	2.50		X
Pb payable after deduction, %	50.00		Y
Payable Pb, lb/t of ore		0.02	$Z = Y \times U \times (((D \times E)/U - Y)/(D \times E)/U \times D \times E/U)$; zero if $(Y > (D \times E)/U)$

Appendix 1 *cont'd*

	Figures to be Input	Calculated figures	Labels
Gold contained in ore, g/t		0.788	AA = K/0.0292
g/t Au in dry concentrate		5.895	AB = AA × M/U
Minimum gold deduction (g Au/DMT of concentrate)	0.7		AC
Payable Au, %		88.1	AD = minimum of (AB – AC)/AB and 95%; zero if (AC > AB)
Payable Au, oz/t of ore		0.007	AE = AA × AD × M × 0.03215
Silver contained in ore, g/t		28.8	AF = P/0.0292
g/t Ag in dry concentrate		202.2	AG = AF × Q/U
Silver deduction (g Ag/DMT of concentrate)	25.0		AH
Ag payable after deduction, %	95.0		AI
Payable Ag, %		83.3	AJ = AI × (AG – AH)/AG; zero if (AH > AG)
Payable Ag, oz/t of ore		0.239	AK = AJ × P × Q × 2204.6/2000
Smelter charge, /DMT of concentrate	$81.66		AL
ditto, /t of ore		$3.60	AM = AL × U
Refining charges, /unit of payable metal: Base charge, /lb	$0.09		
• Cu (/lb)		$0.141	AN = ($0.09+ maximum of ((DU – 0.9) × 0.1) and zero)/DZ
• Pb (/lb)	$0.101		AO
• Au (/oz)	$5.83		AP
• Ag (/oz)	$0.35		AQ
Refining charge, /t ore		$3.28	AR = (AN × W) + (AO × Z) + (AP × AE) + (AQ × AK)
Penalties, /DMT of concentrate	$3.50		AS
Total treatment charge, /t ore		$7.03	AT = AM + AR + (AS × U)
Freight, /WMT of concentrate	$40.83		AU
Moisture content of concentrate, %	7.0		
Freight, /t ore		$1.94	AV = AU × U × 1.075 **
Characteristics of Pb concentrate:			
Lead grade of dry concentrate, %	28.9		AW *
DMT of concentrate/t of ore processed		0.0176	AX = (D × F/AW)/100
Deduction (units of Pb)	1.0		AY
Pb payable after deduction, %	94.0		AZ
Payable Pb, lb/t of ore		10.15	BA = AZ × ((AW-AV)/AW) × D × F × 2204.6)
Copper grade of dry concentrate, %		2.2	BB = A × C/AX
Payable Cu, %	20.0		BC
Payable Cu, lb/t of ore		0.17	BD = BC × AX × BA × 2204.6
Zinc grade of dry concentrate, %		6.4	BE = H × I/AX
Payable Zn, %	50.0		BF
Payable Zn, lb/t of ore		1.24	BG = BF × BE × AX × 2204.6
Gold contained in ore, g/t		0.788	AA
g/t Au in dry concentrate		5.386	BH = (AA × N)/AX
Minimum gold deduction (g Au/DMT of concentrate)	0.7		BI
Payable Au, %		87.0	BJ = minimum of (BH – BI)/BH and 95%; zero if (BI > BH)
Payable Au, oz/t of ore		0.003	BK = AA × BJ × N × 0.03215
Silver contained in ore, g/t		28.8	AF
g/t Ag in dry concentrate		213.1	BL = (AF × R)/AX
Silver deduction (g of Ag/DMT of concentrate)	25.0		BM
Ag payable after deduction, %	95.0		BN
Payable Ag, %		83.9	BO = (BN × (BL – BM)/BM); zero if (BM > BL)
Payable Ag, oz/t of ore		0.101	BP = BO × BL × X × 0.03215

Appendix 1 *cont'd*

	Figures to be input	Calculated figures	Labels
Smelter charge, /DMT of concentrate	$93.33		BQ
Participation in lead prices, %	20		BR
Base price for participation, $US/lb	$0.425		BS
Participation, /DMT of concentrate		$1.79	BT = BR ((DV/BS) – 1) × BR × BQ
Total smelter charge, /DMT of concentrate		$95.12	BU = BQ + BT
ditto, /t of ore		$1.67	BV = BU × AX
Refining charges, /unit of payable metal:			
• Pb (/lb)	$0.101		BW
• Cu (/lb)	$0.141		BX
• Au (/oz)	$5.83		BY
• Ag (/oz)	$0.35		BZ
Refining charge, /t ore		$1.12	CA = (BW × BA) + (BX × BD) + (BY × BK) + (BZ × BP)
Penalties/t of concentrate	$0.00		CB
Penalties, /DMT of ore		$0.00	CC = CB × AX
Total treatment charges, /t of ore		$2.79	CD = CC + CB + BV
Freight, /WMT of concentrate	$40.83		CE
Freight, /t of ore		$0.77	CF = CE × AX × 1.075 **
Characteristics of Zn concentrate:			
Zinc grade of concentrate, %	47.9		CG *
DMT of concentrate/t of ore processed		0.0471	CH = (H × J/CG)/100
Zn in Zn concentrate, lb/t ore		49.69	CI = H × J × 2204.6
Payable Zn, %	85.0		CJ
Payable Zn, lb/t of ore		42.24	CK = CJ × CI
Lead grade of dry concentrate, %		1.6	CL = D × G/CH
Deduction (units of Pb)	2.00		CM
Pb payable after deduction, %	65.0		CN
Payable Pb, lb/t of ore		0.00	CO = CN × CH × ((CL – CM)/CL) × CL; zero if (CM > CL)
Gold contained in ore, g/t		0.788	AA
g/t Au in dry concentrate		0.670	CP = AA × O/CH
Minimum gold deduction (g of gold/DMT of concentrate)	1.200		CQ
Payable Au, %		0.0	CR = minimum of (CP – CQ)/CP and 80%; zero if (CQ > CP)
Payable Au, oz/t of ore		0.000	CS = CR × AA × O × 0.03215
Silver contained in ore, g/t		28.8	AF
g/t Ag in dry concentrate		30.6	CT = AF × S/CH
Minimum silver deduction (g Ag/DMT of concentrate)	110.0		CU
Payable Ag, %		0.0	CV = minimum of ((CT – CU)/CT) and 90%; zero if (CU > CT)
Payable Ag, oz/t of ore		0.000	CW = CV × AF × S × 0.03215
Smelting charge, /DMT of concentrate	$244.98		CX
Base zinc price, $US/lb	$0.68		CY
Escalator, $US/DMT for each 1¢ move above the base	$5.00		CZ
De-escalator, $US/DMT for each 1¢ move below the base	$1.70		DA
Escalator charge/DMT concentrate		$0.00	DB = ((DW – CY) × 100 × CZ/DZ); zero if (CY < DW)
De-escalator charge/DMT concentrate		–$0.08	DC = ((DW – CY) × 100 × DA/DZ); zero if (CY > DW)
Penalties/DMT of concentrate	$11.57		DD
Total smelting charges, /DMT of concentrate		$256.48	DE = DD + DC + DB + CX
Refining charges /unit of payable metal:			
• Pb (/lb)	$0.101		DF
• Au (/oz)	$5.833		DG
• Ag (/oz)	$0.350		DH
Refining charge, /t of ore		$0.00	DI = (DF × CO) + (DG × CS) + (DH × CW)
Total treatment charges, /t ore		$12.07	DJ = DI + (CH × DE)
Freight, /WMT of concentrate	$40.83		DK
Freight, /t of ore		$2.07	DL = DK × CH × 1.075 **

Appendix 1 ***cont'd***

	Figures to be input	Calculated figures	Labels
Treatment and Freight charges:			
Total treatment/t of ore		\$21.89	DM = DJ + CD + AT
Total freight, /t of ore		\$4.77	DN = DL + CF + AV
Total, /t of ore		\$26.66	DO = DN + DM
Payable Metals, /t of ore:			
Copper (lb)		22.47	DP = W + BD
Lead (lb)		10.17	DQ = CO + BA + Z
Zinc (lb)		43.48	DR = CK + BG
Gold (oz)		0.010	DS = (K × L × 2204.6/2000) + AE + BK + CS
Silver (oz)		0.340	DT = CW + BP + AK
Commodity Prices, \$US:			
Copper (/lb)	\$1.21		DU
Lead (/lb)	\$0.46		DV
Zinc (/lb)	\$0.68		DW
Gold (/oz)	\$383		DX
Silver (/oz)	\$4.82		DY
Canadian dollar	\$0.857		DZ
Revenues, /t of ore:			
Copper		\$31.71 (40.6%)	EA = DP × DU/DZ
Lead		\$5.46 (7.0%)	EB = DQ × DV/DZ
Zinc		\$34.49 (44.2%)	EC = DR × DW/DZ
Gold		\$4.47 (5.7%)	ED = DS × DX/DZ
Silver		\$1.91 (2.4%)	EE = DT × DY/DZ
Total Revenues		\$78.04 (100.0%)	EF = EA + EB + EC + ED + EE
Marketing and insurance costs, and losses during transportation, as a % of total revenues	2.0		EG
Net Revenues, /t of ore		\$76.48	EH = EF × ((100 – EG)/100)
Treatment and Distribution Charges, /t of ore		\$26.66	DO
Net Smelter Return, /t of ore		\$49.82	EI= EH – DO
Gross value of ore, /t of ore		\$113.64	EJ = ((((A × DU) + (D × DV) + (H × DW)) × 2204.6) + (((K × DX) + (P × DY)) × (2204.6/2000)))/DZ
Netback (NSR as proportion of gross value)		43.8%	EK = EI/EH

Notes:

DMT = dry metric tonne
SDT = short dry ton
WMT = wet metric tonne
Dollar amounts are Canadian dollars unless otherwise specified

* In the absence of any detailed metallurgical data, one can use the relationships of Table 3, *i.e.*,

B (%) = 83.5 + (2.92 × A(%)) – (1.07 × (H/A))
F (%) = 76.4 + (2.94 × D(%)) – (0.77 × H(%)) – (6.81 × A(%))
J (%) = 79.2 + (1.19 × H(%)) + (1.15 × (D/H)) – (5.62 × (A/N))
T (%) = 23.9 + (0.36 × A(%)) – (0.19 × (H/A))
AW (%) = 22.1 + (3.84 × D(%)) + (2.60 × H(%)) – (2.66 × A(%))
CG (%) = 49.0 + (0.85 × H(%)) – (12.14 × (D/H)) – (0.42 × (A/H))

** assumes that the concentrate is 7% moisture, in which case 1.075 WMT would yield 1.000 DMT