POLYMET MINING CORP.

TECHNICAL REPORT

on the

NorthMet Project

Located in N-E Minnesota, USA, near the City of Hoyt Lakes

Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project

Report compiled from multiple sources under the guidance and supervision of

D. J. Hunter C.Eng. CP (Mining)

October 2006

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GLOSSARY OF TERMS

AERA	Air Emissions Risk Assessment
AIM	Alternative Investment Market
Al	Aluminum
AMDAD	Australian Mine Design & Development, Pty Ltd, Australia.
ARD	Acid Rock Drainage
Au	Gold
AuPGM	Gold plus Platinum Group Metals (particularly platinum and palladium).
Barr	Barr Engineering Company, Minnesota
Bateman	Bateman Engineering Pty Ltd
BFS	Basic Ferric Sulphate
BIF	Biwabik Iron Formation
BSX	Bateman Advanced Technologies
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Cliffs	Cleveland-Cliffs Inc. (parent company of Cliffs Erie LLC)
Cliffs Erie	Cliffs Erie LLC
Cons	Concentrate, produced from flotation
CPT	Cone Penetration Testing
CRO	Control Room Operators
Co	Cobalt
Cu	Copper
CuS	Copper Sulphide
$CuSO_4$	Copper Sulphate

DCS	Distributed Control System
DF250	Dow Froth 250
DFS	Definitive Feasibility Study
EAW	Environmental Assessment Worksheet
EIS	Environmental Impact Statement
EPCM	Engineering, Procurement and Construction Management
ESD	Emergency Shutdown
EW	Electrowinning
FBM	Field bus module
Fe	Iron
Fleck	Fleck Resources
Golder	Golder Associates, Canada and USA
GLI	Great Lakes Initiative
H&S	Hellman & Schofield Pty Ltd, Australia
HVAC	Heating, Ventilation and Air Conditioning
H_2SO_4	Sulphuric Acid
KOA	Krech Ojard & Associates, Duluth, Minnesota
Lakefield	SGS Lakefield Research Ltd., Ontario, Canada
LTVSMC	LTV Steel Mining Company
MDH	Minnesota Department of Health
MDNR	Minnesota Department of Natural Resources
MEPA	Minnesota Environmental Policy Act
Metsim	Steady state mass and energy balance software
$Mg(OH)_2$ slurry	Magnesium hydroxide slurry. Reagent used to increase solution pH, also called (milk of) magnesia
MIBC	Methyl Isobutyl Carbinol – flotation reagent

MOU	Memorandum of Understanding
MPCA	Minnesota Pollution Control Agency
MWCA	Minnesota Wetlands Conservation Act
NaHS	Sodium Hydrosulphide
NEC	Noramco Engineering Co., Hibbing, Minnesota
NEPA	National Environmental Policy Act
Ni	Nickel
NMV	Net Metal Values
NOI	Notice of Intent
North	North Limited
NPDES/SDS	National Pollutant Discharge Elimination System and State Disposal System
NRRI	Natural Resources Research Institute
O_2	Oxygen gas
O/A	Ratio of organic phase mass to aqueous phase mass
Outokumpu	Outokumpu Technology (solid-liquid separation equipment vendors)
P_{80}	The size at which 80% of a sample's mass will pass through a screen
P&ID	Piping and Instrumentation Diagram
PAX	Potassium Amyl Xanthate – flotation reagent
Pd	Palladium
pH	A measure of solution acidity
PFD	Process Flow Diagram
PGE	Platinum group elements
PGM	Platinum group metals
PLS	Pregnant Leach Solution
PolyMet	Polymet Mining Corporation
POX	Pressure Oxidation

PRI	Partridge River Intrusion
Pt	Platinum
QEMSCAN	A technology for quantitative evaluation of materials by scanning electron microscopy
Raffinate	Metal depleted liquor exiting a solvent extraction circuit
RGGS	RGGS Limited
Rio Tinto	Rio Tinto Limited
ROD	Record of Decision
ROM	Run of Mine
RQD	Rock Quality Designator
S	Sulphur
SO_2	Sulphur Dioxide
Spent electrolyte	High acid process liquor returning from electrowinning
SRK	SRK, Vancouver, Canada
SX	Solvent Extraction
US	United States of America
USACE	United States Army Corps of Engineers
USEPA	United States Environmental Protection Agency
USX	U.S. Steel Corporation.
VES	Vertical Electrical Sounding
Whittle optimisation	An industry standard computerised mine planning tool
w/w	Weight of total weight
Zn	Zinc

UNITS

Measurements have generally been stated in US Imperial (or Standard) units, often accompanied by metric units in parenthesis. The following is a listing of the units used in this report.

% (v/v)	Unit of concentration, percentage by volume
% (w/w)	Unit of concentration, percentage by weight
µm	Unit of length, micron
cm	Unit of length, centimetre
d	Unit of time, day
°C	Unit of temperature, degrees Celsius
°F	Unit of temperature, degrees Fahrenheit
ft	Unit of length, foot (feet)
g/t	Unit of mass, grams per metric tonne
g/L	Unit of concentration, grams per litre
gal	Unit of volume, U.S. gallon
h	Unit of time, hour(s)
hp	Unit of power, horsepower
in	Unit of length, inches
lb	Unit of mass, pounds
kPa (g)	Unit of pressure, kilopascals (gauge)
kW	Unit of power, kilowatt
kWh/t	Unit of work index, kilowatt hour per metric tonne
(M) tonnes	Unit of mass, (millions of) metric tonnes
(M) tons	Unit of mass, (millions of) short tons
m	Unit of length, metre
min	Unit of time, minute(s)
mm	Unit of length, millimetre

mt	Unit of mass, metric tonnes
mt/a	Unit of mass flow, metric tonnes per year
M	SI prefix, mega (1×10^{-6})
oz	Unit of mass, troy ounces
ppb	Unit of concentration, parts per billion
ppm	Unit of concentration, parts per million
psig	Unit of pressure, pounds per square inch (gauge)
s	Unit of time, second
st	Unit of mass, short tons
st/a	short tons per year

3. SUMMARY

This Technical Report has been prepared in connection with a Definitive Feasibility Study (DFS) that evaluates the technical and economic feasibility of exploiting a polymetallic deposit located in north-eastern Minnesota. The deposit forms part of the Duluth Complex and contains copper, nickel, cobalt and platinum group metals (PGM) hosted in a large, disseminated sulphide matrix. The objective in undertaking the DFS was to define the capital and operating costs and demonstrate the economic and technical feasibility of the project to a 'bankable' standard which, for the purposes of this document, means that all material aspects of the project have been considered and reported to a level that allows potential finance partners to make informed decisions on the potential of the project. To perform the DFS, PolyMet engaged industry experts in all the engineering and commercial disciplines that such a project encompasses and undertook the extensive drilling, metallurgical and environmental testwork programs needed to define the orebody, determine the optimum processes to extract the metals and to meet all regulatory requirements for permitting the project.

This document describes how PolyMet plans to mine the mineral deposit known as NorthMet by conventional open pit methods at a rate of 32,000 short tons per day of ore, and to extract base and precious metals. The selected ore processing route involves four stages of crushing followed by rod and ball milling with conventional 3-stage flotation to produce a bulk sulphide flotation concentrate. Crushing, milling and flotation will occur in a reactivated former taconite iron ore processing plant, known as the Erie Plant and now owned by PolyMet. The bulk sulphide flotation concentrate will then be subjected to autoclave pressure oxidation leaching followed by hydrometallurgical extractive processes to produce high purity copper cathode, mixed nickel and cobalt hydroxides and a mixed platinum group metals and gold precipitate (AuPGM). An existing tailings basin will be reactivated for flotation tailings and hydrometallurgical residue disposal.

4. INTRODUCTION AND TERMS OF REFERENCE

This report has been prepared by way of collaboration among a number of contributors representing a variety of firms covering specific areas of expertise and technical competence. The consultancy firms that have provided inputs to this report are listed in Table 4-1 below while certificates of qualification of individuals accepting responsibility as Qualified Persons are appended hereto.

Table 4-1 Report Contributors

Overall project engineering and coordination	Bateman Engineering Pty Ltd.
Process design & plant engineering	Bateman Engineering Pty Ltd
Design, management and supervision of metallurgical testwork	Bateman Engineering Pty Ltd
Resource estimation	Hellman & Schofield, Sydney, Australia
Mine planning & scheduling	Australian Mine Design & Development Pty Ltd, Australia.
Mineral Reserve Estimation	Australian Mine Design & Development Pty Ltd, Australia.
Geotechnical assessment & pit slope stability	Golder Associates, Mississauga, Canada.
Blast design	Golder Associates, Sudbury, Canada.
Waste rock stockpile design	Golder Associates, Denver, USA
Waste characterisation	SRK Consulting (Canada) Inc., Vancouver, Canada.
Environmental engineering & permitting	Barr Engineering, Minneapolis, MN.
Ore beneficiation plant refurbishment	Noramco Engineering Co., Hibbing, MN.
Railroad and infrastructure refurbishment	Krech Ojard & Associates, Duluth, MN.
Pressure leaching and hydrometallurgy	Bateman Engineering Pty Ltd
Solvent extraction & electrowinning	Bateman Advanced Technologies, Israel.
Bench and Pilot-scale metallurgical testwork	SGS Lakefield Research Ltd., Ontario, Canada

The purpose of the DFS was to consider and report on all material aspects of developing a mining and extractive metallurgical process for the NorthMet Deposit to a level of definition and confidence that financial institutions could rely upon when making an investment decision.

The general DFS scope of work included the following activities:

- Extensive geological studies and resource definition drilling;
- Detailed geological and resource modelling leading to a Mineral Resource Estimate that is consistent with CIM guidelines and conforms with the reporting requirements of the National Instrument 43-101;
- Development of a mine plan and production schedule based on Whittle pit optimisation incorporating realistic cost, metal price and metallurgical recovery assumptions as well as geotechnical design parameters;
- Environmental baseline data collection, extensive fieldwork and testwork, geochemical characterisation of rock types (both ore and waste), flotation tailings, hydrometallurgical residues and all definable discharges;
- Environmental impact assessment work in accordance with US federal and state environmental permitting requirements;
- Extensive bench and pilot-scale metallurgical testwork;
- Process design and engineering including preparation of process flowsheets (PFDs) and piping and instrumentation diagrams (P&IDs);
- Discipline engineering including civil, structural, mechanical and electrical for new and existing infrastructure and equipment requirements;
- Detailed assessment of existing equipment and facilities to determine the requirements for refurbishing and reactivating existing crushing and milling facilities, reactivating sections of existing railroad, and reactivating related site infrastructure;
- Solicitation and issue of enquiries to equipment vendors, agents and service providers for cost estimates;
- Preparation of overall capital and operating cost estimates to an aggregate accuracy of -5% to +15%; and
- Preparation of a project implementation plan and schedule.

5. DISCLAIMER

Don Hunter acts as NorthMet Definitive Feasibility Study Project Manager on behalf of PolyMet Mining Corp. He has been remunerated by PolyMet as a consultant for his services and holds an equity stake in the company. He is therefore not independent as defined by NI 43-101.

With the exception of Mr. Richard Patelke who is PolyMet's Project Geologist, the other Qualified Persons who have contributed to this report have been remunerated as employees of consultancy firms at their standard rates. With few exceptions, nominated Qualified Persons have visited the project site or the pilot plant facilities at least once and in some cases, repeatedly. In the case of QPs accepting responsibility for reporting on the metallurgical pilot-scale testwork that is an integral part of this study, those individuals visited the pilot plant test facility near Toronto in Canada during the course of testwork; however, they may not have visited the project site in Minnesota. The certificates of qualification, which are appended, provide details of qualifications, experience and specific role in the preparation of this report. They also indicate whether a QP is Independent of the Issuer as defined by NI 43-101.

Responsibility for compiling this report lies with Don Hunter, B.Sc. CP, C.Eng. who is a mining engineer with 33 years of mining industry experience which includes participation in and management of several feasibility studies. Mr. Hunter is a member in good standing of the Australasian Institute of Mining and Metallurgy and the Institute of Materials, Metallurgy and Mining, both of which have codes of ethical conduct applicable to their members. By virtue of his professional training, experience and close association with the NorthMet Project, Mr. Hunter satisfies the criteria for a Qualified Person (as defined by NI 43-101) in his role as report compilation coordinator. In this role, Mr. Hunter has overseen and contributed to several aspects of report preparation including provision of guidance to contributors regarding matters of factual materiality and adherence to appropriate reporting, engineering and safety standards. In the areas and disciplines beyond his professional training and experience, Mr. Hunter has relied on the nominated Qualified Persons and others with appropriate training and experience, to ensure the completeness, correctness, accuracy and content of their contributions to the report.

During the course of the DFS Mr. Hunter has spent extended periods working at the project site and with the various report contributors.

6. PROPERTY DESCRIPTION AND LOCATION

PolyMet's land ownership interests comprise surface ownership, which includes a former taconite processing plant, now known as the Erie Plant, and mineral rights, which cover a large polymetallic deposit known as the NorthMet Deposit which lies approximately eight miles to the east of the Erie Plant at Latitude 47º 36' north, Longitude 91º 58'.

In 2005 PolyMet acquired, by contract for deed, surface ownership of approximately 7,000 acres of real property and the former LTVSMC Taconite Processing Plant, comprising selected parts of a taconite processing facility formerly owned by Cliffs Erie LLC. This property includes crushing and concentrating facilities, tailings basin, warehouses, repair shops and office buildings that will be re-used by PolyMet, and space to construct new hydrometallurgical processing facilities. PolyMet will assume most environmental and reclamation liabilities associated with this property upon issuance of a Permit to Mine and all other environmental permits necessary to conduct its mining and metallurgical operation on or about the property and PolyMet has indemnified Cliffs for the ongoing reclamation cost.

Mineral and surface rights in the NorthMet Deposit area have been severed and, in this case, the surface owners cannot prohibit mining in the leased area. The U.S. Forest Service (USFS) is the surface owner of most of the lands in the mineral lease area.

PolyMet (as Fleck Resources) acquired a 20-year renewable mineral rights lease to the NorthMet Deposit in 1989 from USX, which disposed of much of its non-core real estate to RGGS in 2003 consequently transferring the underlying mineral rights to RGGS. This lease, which has been maintained in good standing, covers approximately 4,162 acres out of a total of approximately 4,680 acres that PolyMet expects to utilise in its mining operations. The remaining area comprises of seven separate parcels within the mine area that are not currently under mineral lease to PolyMet. However, these parcels are small and PolyMet is in advanced discussions to acquire these mineral interests prior to the commencement of mining activities.

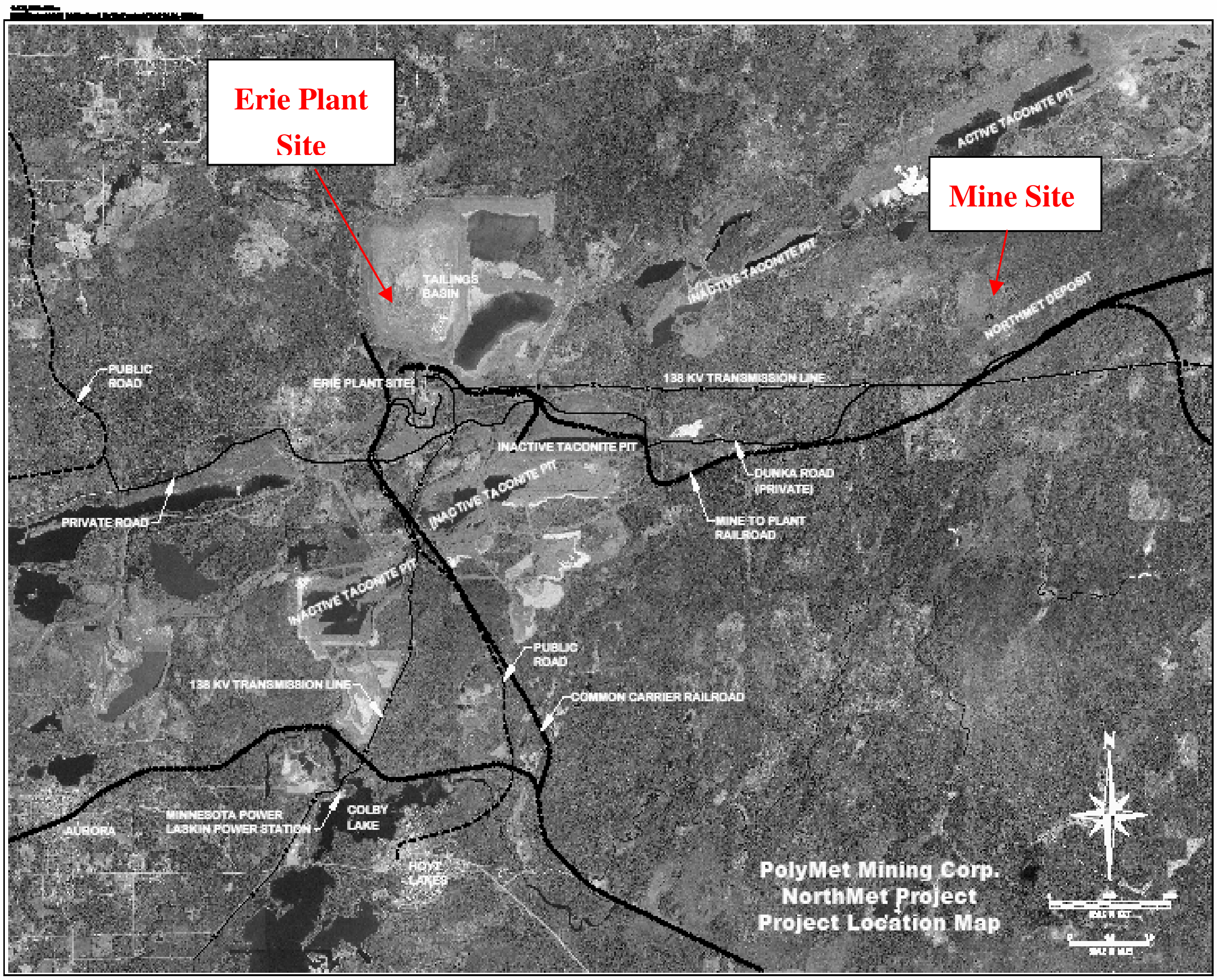
The RGGS lease is maintained by annual payments of US$150,000 until commercial production after which a sliding scale net smelter return royalty based on net metal value per ton processed. At any likely net metal value, the royalty is fixed at 3% of that net metal value. Lease payments made before production will be considered advance royalty payments and credited to the production royalty provided PolyMet pays at least the annual lease payment amount due each year.

There is an existing rail line between the NorthMet Deposit and the processing plant. An agreement with Cliffs Erie provides that Cliffs Erie will either provide rail haul services or sell the required railroad facilities to PolyMet. PolyMet is completing negotiations with Cliffs Erie to acquire additional lands, trackage and other railroad assets in order for PolyMet to secure existing access and assets and control required railroad services.

PolyMet either has or can acquire all the necessary leases and owns or is in the process of acquiring lands, easements and rights of way to be able to develop the NorthMet Deposit and to transport ore to the processing plant. By virtue of the terms of its agreement with Cliffs Erie, PolyMet can be certain to acquire full rights to the property, facilities and associated infrastructure necessary to carry out its plans to develop the property provided it continues to meet its obligations in terms of the contract of deed and conditional on the issuance of the necessary operating permits to mine.

Figure 6-1 Project General Location Map

Figure 6-2 Project Area Location



7. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

The Mesabi Iron Range has been a major centre for iron ore mining for over 100 years and today the region still has six large operating taconite mines and associated process plants. In 2005 over 136 million long tons of taconite ore were mined from which 40.5 million long tons of taconite pellets were produced and shipped, most of which was transported by rail and ore carriers to blast furnaces elsewhere in the Great Lakes region.

As a result of mining activity, an extensive network of railroads and paved roads has developed throughout the region that today provides excellent transport communications. Similarly, the region has an extensive and reliable power supply network with two coal-fired thermal power stations located within 8 and 85 miles of the project site respectively. Power is supplied to the plant site by 138 kV overhead transmission line linked to the regional power grid.

The St. Louis County Inspector of Mines reported that in 2005 the total number of direct employees of the taconite mines and processing facilities was 3,568 compared with over 6,000 a decade earlier and over 11,000 in the 1980s. While mining related employment has shrunk, productivity and efficiency have increased significantly over the same period. The other major employer in north-eastern Minnesota is the forestry, paper and wood products industry. In recent years it too has suffered contraction and the area has experienced a significant decline in economic activity and employment opportunities; a situation that was exacerbated when the LTVSMC facility and mine closed in early 2001.

A major asset of the area is that it retains engineering and technical resources supporting the remaining taconite operations and, importantly, technically skilled men and women are potentially available for employment by PolyMet.

Importantly, and in addition to an excellent infrastructure, the project enjoys strong support among local communities which regard it as an opportunity for economic renewal.

Access to the property is by a combination of good quality asphalt and gravel roads. The nearest centre of population is the City of Hoyt Lakes with a population of about 2,500. There are a number of approximately similarly sized communities in the vicinity, all of which are well serviced, provide ready accommodation, and have been or still are directly associated with the region’s extensive taconite mining industry. The road network in the area is well developed though not heavily trafficked and there is an extensive railroad network which serves the taconite mining industry across the entire Iron Range. There is access for ocean shipping to the Ports of Duluth and Superior via the St. Lawrence Seaway.

While the Iron Range forms an extensive and prominent regional topographic feature, the project site is located a short distance south of the Range where the surrounding countryside is characterized as gently undulating. Elevation at the project site is about 1,600 feet above sea level (1,000 feet above Lake Superior). Much of the region is poorly drained and the predominant vegetation comprises wetlands and boreal forest. Forestry is a major local industry and the project site and much of the surrounding area has been repeatedly logged.

Climate is continental and characterized by harsh winters, wide temperature variations and significant precipitation. The nearest official recording station is at the town of Babbitt, about 6.5 miles north of the deposit, where temperature averages 5ºF (-15°C) in January and 66ºF (18°C) in July. Average annual temperature is 38.5°F (4°C) with minimums as low as – 40°F (-40°C). The average summer temperature is 56°F (13.5°C) though during short periods temperatures may reach as high as 90ºF (32°C) with high humidity. Average annual precipitation is about 28 inches (71 cm) with about 30% of this falling mostly as snow between November and April. Annual snowfall is typically about 60 inches (152cm) with 24 to 36 inches (61 to 91cm) on the ground at any one time. The local taconite mines operate year round and it is rare for snow or inclement weather to cause production delays.

The Erie Plant is connected to the Minnesota Power electrical power supply grid by two 138kV transmission lines. One transmission line comes from a 225 MW coal fired power station at Taconite Harbor on the north shore of Lake Superior and runs along the southern edge of the mining lease area while the second comes from the Laskin 125 MW coal fired facility situated to the west of Hoyt Lakes. Both facilities are owned and operated by local power supply utility Minnesota Power.

There are abundant local sources of fresh water. Plant process make-up water will be sourced from Colby Lake, which serves both the Laskin power generating plant and the community of Hoyt Lakes.

8. HISTORY AND PROJECT BACKGROUND

The deposit lies adjacent to and immediately south of the Mesabi Iron Range, a 100 mile long geological feature in northern Minnesota that has been an historically important centre of iron ore production since the 1880s. Mining was originally focussed on high grade iron ore (haematite) which was largely mined out by the middle of the 20th century, after which production switched to lower grade taconite ore. In order to achieve economies of scale, the US iron and steel industry constructed eight large taconite processing plants along the Range of which six remain in production. The most easterly of these plants is now known as the Erie Plant and was built by a consortium of iron and steel companies in the 1950s. The mine and plant were subsequently acquired by LTV Steel, but operated by Cliffs Erie LLC, a subsidiary of Cleveland-Cliffs, Inc. ("Cliffs"). The Erie Plant is located approximately eight miles west of the NorthMet Deposit. (An annotated aerial photograph of the NorthMet Deposit in relation to the Erie Plant is shown as Figure 6-2). The Erie Plant was closed in January 2001 as a result of bankruptcy of LTV Steel. Shut down was carried out in a systematic and orderly fashion with an expectation of eventually re-opening.

As a result of iron mining, the area has a well developed infrastructure of roads, railroads, equipment vendors, technical and engineering service providers, communications and power supply as well as being served by significant centres of population.

In the early 1950's, local prospectors identified non-ferrous metal outcrops near the town of Ely. In the 1950s, as interest in the area grew owing to the construction of the taconite plants, Inco then Bear Creek Mining (subsequently acquired by Kennecott and, in turn, by Rio Tinto) began systematic exploration and investigation of the area. The NorthMet Deposit was discovered in 1969 by United States Steel Corporation ("USX") and since then has been extensively drilled by USX and PolyMet.

To date over 2,100 holes have been drilled into the Duluth Complex of which 310 holes totalling 261,640 feet have been drilled into the NorthMet Deposit. Over time at least seven bulk samples have been extracted for metallurgical testwork with the most recent testwork campaigns designed and implemented by PolyMet in 2005 and early 2006 to support the requirements of the DFS.

There are several known Cu/Ni/precious metal deposits in the area, each of which lies along the north western outcrop of the Duluth Complex which, in the vicinity of NorthMet, lies above the Biwabik Iron Formation that hosts the iron ore mines. Each of the deposits is now know to contain varying grades of copper, nickel, and precious metals – primarily platinum group elements (PGE). However, in the 1950s and 1960s there was little market for PGEs and assaying techniques could not reliably and economically detect relatively low grades of PGEs so their full economic value was overlooked. More significantly, with metallurgical recovery processes of the period, it was impossible to obtain a clean separation between the copper and nickel – the nickel concentrates were contaminated by copper and, of even greater importance, the copper concentrate was contaminated by nickel, which would have resulted in unattractive smelter terms.

Thus, at that time development would have required construction of dedicated smelting and refining capacity which would have resulted in large capital costs – and without the ability to derive any revenues from precious metals.

In the 1980s, the Minnesota Department of Natural Resources (MDNR) recognised that the Duluth Complex contained PGEs in association with the previously identified nickel and copper. This economic gain led to a number of process development initiatives.

PolyMet's involvement with the NorthMet Deposit commenced in 1989 when PolyMet (then called Fleck Resources) acquired a 20-year automatically renewable mining lease from USX and started investigating the potential for economic development of NorthMet. The evaluation went through a number of distinct stages:

- resource definition which included extensive drilling;
- process development;
- a Pre-feasibility Study completed in 2001 with capital and operating costs reported to +30% -30% accuracy;
- the Definitive Feasibility Study described here - with capital and operating costs reported to +15% -5% accuracy; and
- an environmental assessment and permitting program running in parallel with the DFS.

In 1998 Fleck changed its name to PolyMet and focused on the development of hydrometallurgical processes to recover the metals. Specifically, and in conjunction with Lakefield Research (now SGS Lakefield Research and hereinafter Lakefield), PolyMet developed a chloride-assisted pressure leaching process in which the copper, nickel, cobalt and precious metals are placed into solution in a single operation and metals are then sequentially recovered from solution in the form of electrowon copper cathode, nickel-cobalt hydroxides, and a precious metal precipitate.

In July 2000, PolyMet entered into a joint venture arrangement with North Limited (North), a major Australian mining company under which North would finance the NorthMet Project into commercial production. Under the joint venture arrangement, North had the opportunity to ultimately earn an

87.5% interest in the project through funding a Bankable Feasibility Study and providing the total capital cost to develop the project.

In August 2000, North was taken over by Rio Tinto, which gave PolyMet the opportunity to re-acquire control of the project. NorthMet is now 100% controlled by PolyMet, subject to a 3% royalty payable to RGGS Ltd, a privately-owned Texas-based real estate company, which acquired large amounts of real estate and mineral leases from USX in 2003.

Before the demise of North, PolyMet did secure sufficient funding to complete a pre-feasibility study. During 2000, PolyMet completed laboratory and pilot-scale metallurgical testwork and commissioned a Pre-feasibility Study which was completed in April 2001. The Pre-feasibility Study was for a totally 'greenfield' development project with a mill feed production rate of 50,000 tons per day (18.25 million short tons per year). The high capital cost required for a 'greenfield' development of this scale and the then long term, low metal price projections resulted in marginal economics. Moreover, without the economic support of North, PolyMet was unable to fund the necessary optimisation work and no further project development was undertaken after completion of the Pre-feasibility Study until 2003 when the present management team took over.

The new management team that took control of PolyMet in March 2003 identified that a key component of project development was acquisition of the Erie Plant. PolyMet's management team recognised that, compared with a 'greenfield' development, significant capital cost savings were possible by acquiring and reactivating the Erie Plant assets and that this resulted in greatly improved economics over previous development concepts.

In September 2003 PolyMet negotiated an option with Cliffs that provided for the purchase of crushing facilities, the concentrator, tailings impoundment and certain associated elements of infrastructure that could be reactivated and form part of the proposed NorthMet mining and processing operation. In 2005 PolyMet and Cliffs agreed to expand the option to cover additional assets and land that will secure access to the mine site and facilitate project development and, in November 2005, PolyMet exercised the option. Figure 6-1 shows an aerial photograph covering the NorthMet Deposit and the Erie Plant Site while Figure 8-2 below shows the plant site buildings now owned by PolyMet.

As a result of its exclusive rights to the Erie Plant, PolyMet was able to re-scope the project. In January of 2004 PolyMet engaged Barr Engineering (Barr) to start environmental work for permitting the project and in June 2004 Bateman Engineering Pty Ltd was engaged as Project Engineer. Work on the DFS started formally in August 2004.

Figure 8-1 Erie Plant Site from the air showing part of the tailings basin and the majority of former LTVSMC plant facilities now owned by PolyMet.

9. GEOLOGICAL SETTING

9.1 Introduction

Located in the Partridge River Intrusion of the Duluth Complex, NorthMet is a large, disseminated sulphide deposit associated with the 1.1 billion year old Mid-continent Rift. Metals of interest are copper, nickel, cobalt, platinum, palladium, and gold. The majority of these are concentrated in four sulphide minerals, namely: chalcopyrite, cubanite, pentlandite, and pyrrhotite, with platinum, palladium and gold also found in bismuthides, tellurides, and alloys.

NorthMet is one of eleven known copper-nickel-precious metal deposits along the northern margin of the Complex. All these deposits share a broadly similar geologic setting comprising disseminated sulphides with minor but locally massive sulphides in layered heterogeneous troctolitic rocks which form the basal unit of the Complex. The Complex footwall comprises older rocks of sedimentary origin which host the nearby taconite mines and generally dip at about 25° to the south-east.

9.2 Regional Geology

Two broad age groups dominate rocks with mineral potential in north-eastern Minnesota: Archean and Proterozoic.

9.2.1 Archean Rocks

Archean rocks represent possible hosts for lode gold, Volcanogenic Massive Sulfide (VMS), copper-nickel-PGE, and diamond prospects. Historically, numerous iron ore mines operated in this terrane. North of the PolyMet site is extensive and under explored terrane of exposed Archean rock, similar to that in Ontario (Wawa and Quetico sub provinces). This terrane is comprised of granite-greenstone belts which have an approximately east-west strike. Archean rocks form the basement at the NorthMet site.

9.2.2 Proterozoic Rocks

Proterozoic rocks of interest in northern Minnesota include the Animikian Basin (Paleoproterozoic, 1.8 billion years old) sedimentary rocks (in the NorthMet area these are, oldest to youngest, the Pokegama Quartzite, Biwabik Iron Formation, and the Virginia Formation) and the Keweenawan-aged (Mesoproterozoic, 1.1 billion years old) igneous and sedimentary rocks.

Of the three Paleoproterozoic sedimentary formations that form the footwall at NorthMet, only the Virginia Formation (comprised of turbidites and graywackes that are locally sulfide-bearing) contacts the Duluth Complex igneous rocks.

9.2.3 Duluth Complex

The Mesoproterozoic rocks include the Duluth Complex, a large, composite, tholeiitic mafic intrusion that was emplaced into comagmatic flood basalts of the North Shore Volcanic Group. Other units of the Keweenawan system include the Beaver Bay Complex, and associated minor intrusions. These rocks are all part of the Midcontinent Rift igneous system, an arcuate structure starting in Kansas, trending north to Lake Superior, following the Lake Superior basin, and then curving south towards mid-Ohio. Outcrop of rift related rocks is limited to the St. Croix River Valley in eastern Minnesota and the Lake Superior region.

Rocks of the Complex are varied and include troctolitic, anorthositic, gabbroic, granodioritic, and granitic intrusive bodies. Generally, these rocks are troctolitic / gabbroic and divided into an Anorthositic Series, Troctolitic Series (Taylor, 1964), and a late Felsic Series (Weiblen and Morey, 1980). Initially, on the basis of abundant field evidence, rocks of the Anorthositic Series were inferred to have been emplaced early in the evolution of the Complex. However, high-resolution U-Pb isotopic age dates indicate that the Troctolitic and Anorthositic Series have indistinguishable crystallization ages of about 1,099 Ma (Miller, 1992; Paces and Miller, 1993). The Felsic series has so far been unimportant in economic potential.

Emplacement of the Complex occurred during an episode of extensional tectonism that produced the Midcontinent Rift System. Weiblen and Morey (1980) present a half-graben model for the overall emplacement style with a step-and-riser configuration of the basal contact, due to steep, southeast-dipping, northeast-trending normal faults. Inferred footwall faults at NorthMet conform to this step-and-riser geometry. According to the model, magma was injected into fault-bounded voids, formed during rifting, to produce multiple smaller intrusions that collectively comprise the Complex. They also suggest that these northeast-trending faults may be offset by a series of northwest-trending strike-slip (transform) faults. Some studies suggest that the grades of Cu-Ni and PGE mineralization increases in close proximity to fault zones and other structural features within the Complex.

Eleven, low-grade, large, copper-nickel-PGE deposits are hosted by the Complex in the area of NorthMet. All of these share grossly similar geologic settings to NorthMet, namely disseminated sulphides with minor local massive sulphides in the heterogeneous rocks forming the basal unit of the Duluth Complex along the contact with older rocks.

These deposits occur in two of the largest and oldest of the sub-intrusions of the Complex: NorthMet, Wetlegs, Wyman Creek, and Babbitt (MinnAMAX or Mesaba) are in the Partridge River intrusion (PRI); and Serpentine, Dunka Pit, Birch Lake, Maturi, Maturi Extension, Spruce Road, and South Filson Creek are in South Kawishiwi intrusion (SKI).

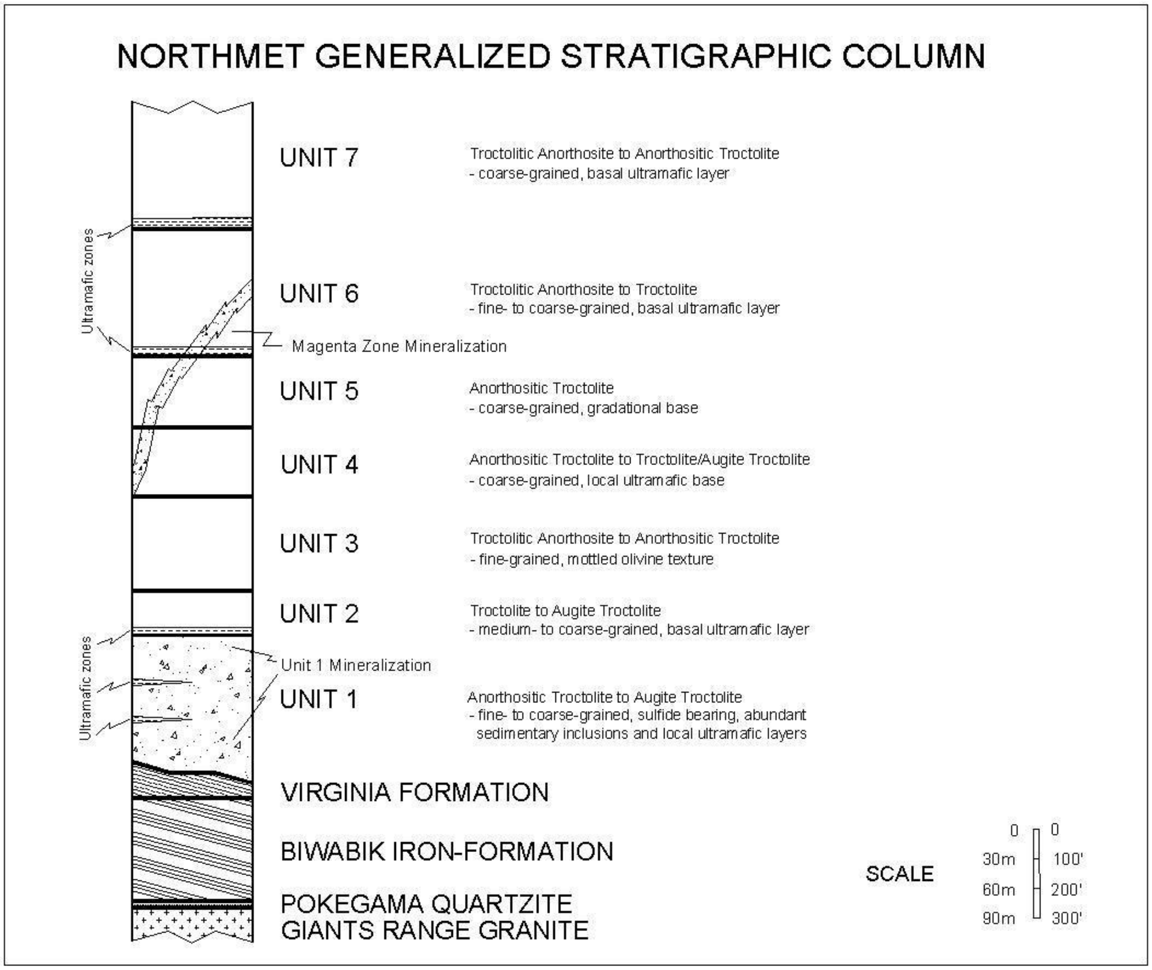
9.2.4 Partridge River Intrusion

The Partridge River intrusion, host to the NorthMet Deposit, has been extensively drilled (about 1,100 drill holes to date). The PRI rocks are divided into at least eight separate and distinct rock units in drill core. These units are known as Units 1 – 8 of which NorthMet drilling has intersected Units 1 – 7. (Figures 9-1 and 9-2). These units are composed primarily of light gray to dark gray, medium- to coarse-grained, troctolitic anorthosite to pyroxene (augite) troctolite and in lesser amounts, gabbroic anorthosite to olivine gabbro. At NorthMet, igneous rocks directly overlie Virginia Formation, elsewhere in the PRI they are locally in contact with the Biwabik Iron Formation.

This basic stratigraphy is present in hundreds of drill holes along a 15-mile strike length. Definition of the stratigraphy has provided a framework by which mineralized zones, containing elevated values of Cu-Ni and precious metals, can be traced and correlated.

In the NorthMet area, the base of the Complex is in relatively sharp (locally gradational over a few feet) contact with Lower Proterozoic metasediments of the Virginia Formation (argillite and graywacke sequence). The underlying iron-formation is seen in deeper NorthMet drill holes and outcrops in the Peter Mitchell Mine of Northshore Mining Company, less than two miles north of the deposit.

Figure 9-1 NorthMet Stratigraphic Column



9.3 Geology of the NorthMet Deposit

9.3.1 Introduction

NorthMet consists of seven igneous units that dip southeast, with most economic sulfide mineralization in the lowermost, basal unit (Unit 1). The following is a summarized description of the geology of the deposit, based on observations from drill core and limited outcrop mapping.

9.3.2 Quaternary Geology

In general the Quaternary geology of the region is comprised of a thin blanket of glacial deposits which varies in thickness from absent to (locally) about 60 feet with an average thickness of about 12 feet. Glacial deposits consist of till, lacustrine materials, and outwash. Topographic low spots are usually filled by peat bog or open wetland and relief is described as subdued and gently undulating with poor drainage.

Site specific geologic studies of the glacial deposits have not been done, though a series of geophysical soundings were carried out in 2006 to better define its thickness around the area to be mined (Ikola, 2006). Regional mapping and observations of test pits and other small excavations made in connection with drilling characterize drift materials as consisting of unsorted sand / silt / clay with cobbles and boulders, locally overlain by post-glacial peat. Boulders can be greater than 10 feet in size.

9.3.3 Structural Geology

The general structure of the deposit, including individual beds within the underlying Biwabik Iron Formation (BIF) and Virginia Formation, has an overall dip ranging from 15°-35° to the south-east, striking approximately N56°E. Dips in the seven igneous units that occur at NorthMet are grossly similar, but within the mineralised zone of the East Pit area may vary up to 60°, a feature which can be attributed to crustal loading, associated with the input of large volumes of magma originating from the Midcontinent Rift System.

At least 14 faults have been postulated across the deposit though there is insufficient evidence in drill core to indicate with certainty the presence and location of major offsets or faulting within the igneous rock units. It is evident, however, that offsets or faulting exist within the footwall rocks but may not extend far into the Complex. Many of the footwall offsets can be correlated between adjacent cross-sections but cannot be correlated in the Complex itself. Drill core contains infrequent brecciated intervals, gouge mineralisation, slickensides on serpentinised fracture faces, and some broken zones but these tend to be localised and do not generally correlate well from hole-to-hole. So far, no apparent local relation between the inferred location of faults and mineralization has been delineated.

The influence of faulting on mining is expected to be minimal and the geometry of regional and known local faulting has been considered in pit planning and in pit slope stability assessments. There is no apparent relation between inferred fault zones and mineralisation. Geotechnical logging and sample testing during 2005 confirmed the overall strength and competence of the rock mass.

9.3.4 Logging and Mapping Units

A summary of the general stratigraphy of the NorthMet Deposit is outlined below. Rock units and formations are listed in descending order, as would be observed from top to bottom in drill core. NorthMet units are labelled as Units 1 through 7, bottom to top. Unit 3 is the oldest, though the intrusion sequence of the other units is not clear.

The broad picture is of a regular stratigraphy of troctolitic to anorthositic rock units, dipping southeast at 20° to 25°, with basal ultramafic units commonly defining the boundaries of these units. The basal ultramafic zones tend to have diffuse tops, sharp bases, and are commonly serpentinized and foliated (See Figures 9-1 & 9-2). Economic sulfide mineralization is ubiquitous in the basal unit, Unit 1, and is locally present, but restricted, in the upper units.

9.3.5 Rock Type and Unit Classification

Igneous rock types in the Complex are classified by visually estimating the modal percentages of plagioclase, olivine, and pyroxene, using the rock classification scheme shown in Figure 9-3. Due to subtle changes in the percentages of these minerals, a variation in the defined rock types within the rock units may be present from interval to interval or hole to hole. This is especially true for Unit 1. Table 9-1 gives the mineral chemistry of common minerals at NorthMet.

Table 9-1 Mineral formulae for the minerals commonly occurring at NorthMet

SILICATES	GENERALIZED FORMULA	ABUNDANCE
Plagioclase (AN60)	$Na_{0.4}Ca_{0.6}Al_{1.6}Si_{2.4}O_8$	45-75%
Olivine	$(Mg, Fe)_2SiO_4$	20-40%
Clinopyroxene (augite)	$(Ca, Na)(Mg, Fe,Al)(Si,Al)O_6$	5-15%
Orthopyroxene	$(Mg, Fe)SiO_3$	0-2%
Biotite	$K(Mg, Fe)_3(AlSi_3O_{10})(OH)_2$	0-2%
Potassium feldspar	$KAlSi_3O_8$	0-1%
Apatite	$Ca_5(PO4)_3(F,Cl,OH)$	trace to 1%
Amphibole (hornblende)	$(Ca, Na)_{2-3}(Mg, Fe, Al)_5Si_6(SiAl)_2O_{22}(OH)_2$	trace to 1%
Chlorite	$(Mg, Fe)_3(Si, Al)_4O_{10}(OH)_2(Mg, Fe)_3(OH)_4$	trace to 1%
Serpentine	$Mg_3, Si_2O_5(OH)_4$	trace to 1%
Sausserite		trace to 1%
SULFIDES[1]		
Chalcopyrite	$CuFeS_2$	0-3%
Cubanite	$CuFe_2S_3$	0-3%
Pentlandite	$(Fe, Ni)_9S_8$	0-1%
Pyrrhotite	$Fe_{1-X}S$	0-5%
OXIDES		
Ilmenite	$FeTiO_3$	0.5-3%
Magnetite	Fe_3O_4	0-1%

[1] Note that the sum of the disseminated sulfide minerals is rarely greater than about 5%, and that proportions can vary over short distances at all scales.

Unit definitions are based on:

- overall texture of a rock type package;
- mineralogy;
- sulfide content; and
- context with respect to bounding surfaces (i.e., ultramafic horizons, oxide-rich horizons).

Unit definitions are not always immediately clear in logging, but usually clarified when drill holes are plotted on cross-sections. In other words, to correctly identify a particular stratigraphic unit, the context of the units directly above and below must also be considered.

Based on drill hole logging, the generalized rock type distribution at NorthMet is about 83% troctolitic, 6% anorthositic, 4% ultramafic, 4% sedimentary inclusions, 2% noritic and gabbroic rocks, and the rest as pegmatites, breccia, basalt inclusions, and others.

The regional and local geology are well known, (Geerts, 1994, Severson et al., 1996, 2000, Hauck et al., 1997, Miller et al., 2001, 2002). There are over 1,100 exploration drill holes on this part of the Complex, and nearly 800,000 feet of core have been re-logged in the past fifteen years by a small group of company and university research geologists (Patelke, 2003). A total of seven Units were recognised in core at NorthMet and these are described below and illustrated in the stratigraphic column shown in Figure 9-1

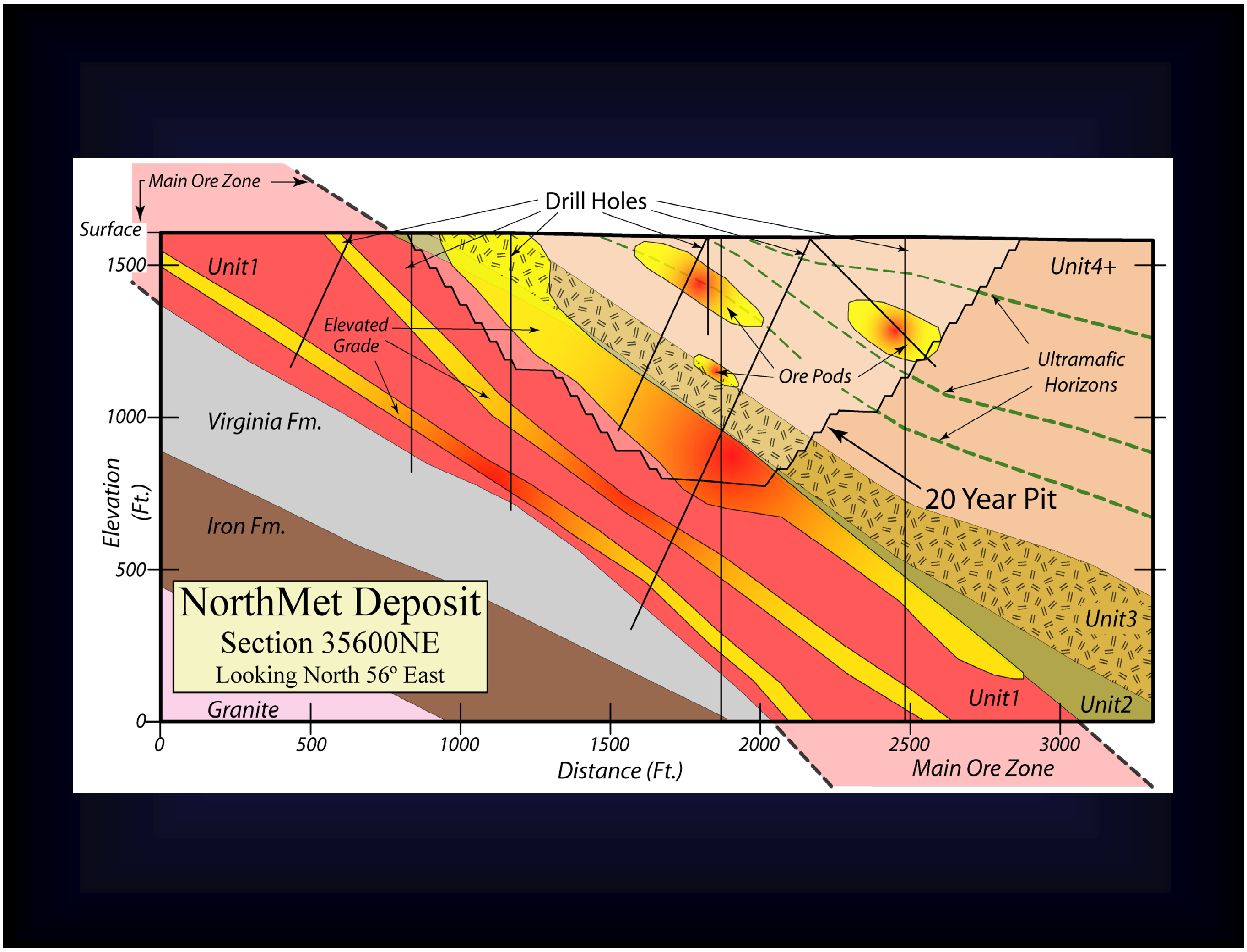
Unit 1:	consists of a heterogeneous mixture of troctolitic to gabbroic rocks, with abundant inclusions of hornfelsed sedimentary footwall rocks and lesser discontinuous layers of ultramafic rock. Unit 1 is the dominant sulphide-bearing member in the NorthMet deposit. At least three Platinum group element ("PGE") enriched "stratabound" layers are present within Unit 1, the uppermost of which has the highest concentrations of PGE. Unit 1 is 200 feet to 1000 feet thick, averaging 450 feet.
Unit 2:	consists of homogenous troctolitic rocks, with minor sulphide mineralization, and a fairly persistent basal ultramafic layer that separates Unit 2 from Unit 1. Unit 2 averages about 200 feet thick.
Unit 3:	consists of a fine-grained, poikolitic, anorthositic troctolite. Unit 3 is the major marker bed within the deposit due to its fine-grained nature and the presence of distinctive olivine oikocrysts that give the rock a mottled appearance. Unit 3 contains little or no mineralization and averages 250 feet thick.
Unit 4:	consists of homogenous ophitic augite troctolite with a local ultramafic layer at, or near, the base of the unit. There is little or no mineralization in this unit and it averages about 300 feet thick.
Units 5, 6, and 7:	consist of homogenous anorthositic troctolite grading to ophitic augite troctolite; units 6 and 7 have persistent ultramafic bases. There is little or no economic sulphide mineralization except for a small horizon in six drill holes in Unit 6. These generally unmineralized units average about 1,200 feet in thickness, but because the top of Unit 7 has not been seen in drill core, this figure is probably a minimum. Preliminary assessment shows that PolyMet would intersect very little of these upper units in its pit development.

The footwall rock at NorthMet is the sedimentary Lower Proterozoic (1.8 Ga) Virginia Formation which is underlain by the Biwabik Iron-Formation.

9.3.6 Alteration and Fracturing

The majority of rock within the NorthMet Deposit is unaltered. Where alteration does occur it is mostly related to the close proximity of fractures and/or joints that cross-cut the troctolitic rocks. The majority of sulphide mineralisation is independent of alteration. Core recoveries during diamond drilling were uniformly high and geotechnical logging of the Rock Quality Designator (RQD) averages greater than 94%.

Figure 9-2 Typical Geologic Cross Section through NorthMet



9.3.7 Controls on Mineralisation

The majority of economic mineralisation occurs in the basal horizon, Unit 1, with copper and nickel in chalcopyrite, cubanite, and pentlandite, all in the presence of pyrrhotite. Cobalt is contained in sulphides. Platinum, palladium, and gold, which show good correlation with sulphur and the other metals, occur in sulphides and also in a variety of tellurides, bismuthides, and alloys.

There is a smaller zone of economic mineralisation at the western end of the property in the upper units, known as the 'Magenta zone.' This zone is generally copper and PGE-rich (sulphur-poor relative to metals) and of good to moderate grade compared with Unit 1.

The major controls on mineralisation, in an exploration sense, are proximity to the footwall and heterogeneous troctolitic host rocks.

Most sulphide mineralisation at NorthMet is of an igneous source though some is locally modified by sulphur derived from footwall metasedimentary rocks (Virginia Formation). Minor veins and other cross-cutting relations indicate some movement of sulphides within the deposit, but there is no evidence for large-scale relocation of sulphides, nor is there macroscopic evidence for any hydrothermal event that may have remobilised PGEs or sulphides.

Virtually all sulphide mineralisation at NorthMet moved in with magmatic pulses, whereas metal enrichment of the magma happened in a deeper chamber. Therefore, the main controls on the location of mineralisation within the deposit are the specific magmatic pulse or pulses making up the individual units.

Figure 9-2 shows a simplified geologic cross-section and illustrates the relationship between the various Units comprising the deposit. Unit 1 is mineralised throughout the deposit and generally shows highest grades near its top. Although current resource estimates have been limited to material within approximately 1,100 feet of surface that has a reasonable expectation of being mined (based on metal price and mining and processing cost assumptions) in the foreseeable future, deep drilling has shown Unit 1 to be mineralised to depths of at least 2,500 feet below surface.

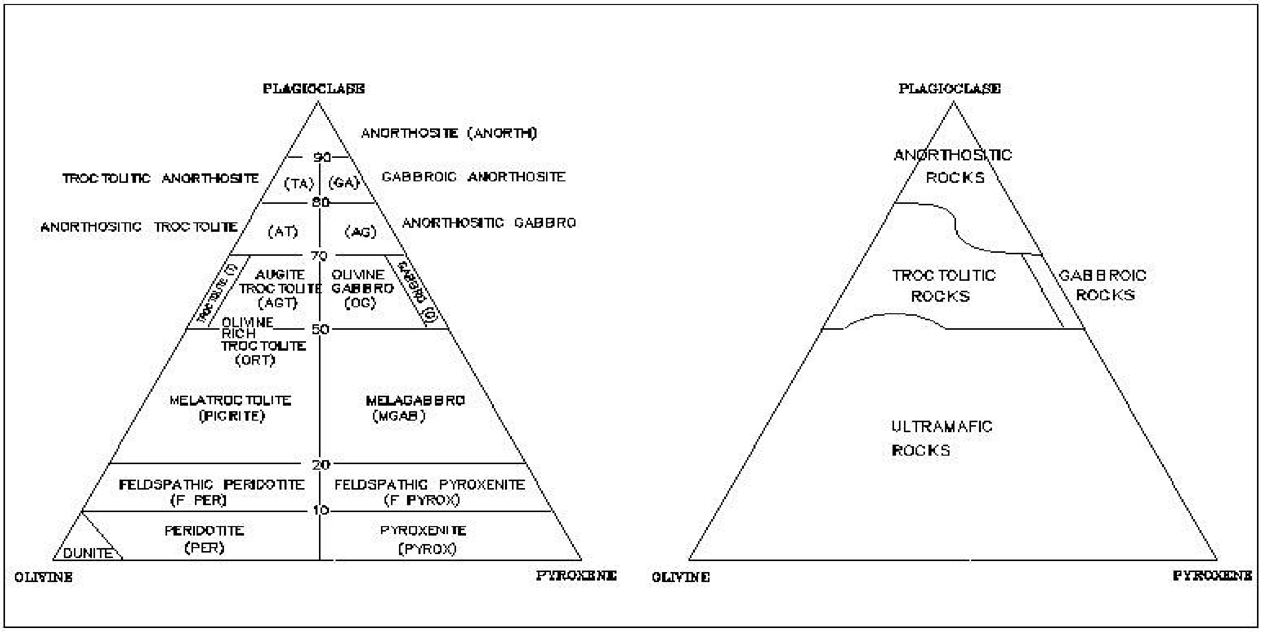
Units 2 through 7 show some economic mineralisation in the western and central parts of the deposit, but essentially no continuous zones in the east. There is no known economic mineralisation in the footwall rocks.

9.3.8 Geophysics

The deposit was originally discovered while drilling a geophysical conductor in the footwall. Geophysical work is largely regional in its resolution and, consequently, as drilling data density has increased less use has been made of geophysical data.

During 2006 a program of Vertical Electrical Soundings (VES) was carried out to investigate overburden depths in areas of sparse drilling and difficult access. This information was required for a combination of general site engineering and for waste stockpile design purposes. Where VES soundings were made near exploration drill holes the depth of overburden comparison is good.

Figure 9-3 Phinney's Rock Classification Scheme



10. DEPOSIT TYPE

The NorthMet deposit is a large-tonnage, disseminated accumulation of sulphide in mafic rocks with rare massive sulphides. Copper to nickel ratios generally range from 3:1 to 4:1. Primary mineralization is probably magmatic, though the possibility of structurally controlled re-mobilization of the mineralization (especially PGE's) has not been ruled out. Sulfur source is both local and magmatic (Theriault et al., 2000). Extensive, detailed logging has shown no definitive relation between specific rock type and the quantity of sulphide mineralization in the Unit 1 mineralized zone or in other units, though the localized noritic rocks (related to footwall assimilation) tend to be of poorer PGE grade and higher in sulphur.

Footwall faults are inferred from bedding dips in the underlying sedimentary rocks, considering the possibility that Keweenawan syn-rift faults may affect these underlying units and have less movement, or indeed no effect on the igneous units. Nonetheless, without faults, the dips do not reconcile well with the overall slope of the footwall. There are some apparent offsets in the igneous units, but definitive fault zones have not been identified. So far, no apparent local relation between the inferred location of faults and mineralization has been delineated.

11. MINERALIZATION

The metals of interest at NorthMet are copper, nickel, cobalt, platinum, palladium, gold, and very small amounts of rhodium and ruthenium. In general, the metals are positively correlated with copper mineralization; cobalt is the exception. Mineralization occurs in four broadly defined horizons throughout the NorthMet property. Three of these horizons are within basal Unit 1, and in some drill core the horizons are visually indistinguishable from each other. The thickness of each of the three enriched horizons varies from 5 feet to more than 200 feet. Unit 1 mineralization is found throughout the base of the deposit. A less extensive mineralized zone is found in Unit 6, and it is relatively enriched in PGE's compared to Unit 1.

Sulphide mineralization consists of chalcopyrite, cubanite, pyrrhotite, and pentlandite, with minor bornite, violarite, pyrite, sphalerite, galena, talnakhite, mackinawite, and valleriite. Sulphide minerals occur mainly as blebs interstitial to plagioclase, olivine, and augite grains, but also may occur within plagioclase and augite grains, as intergrowths with silicates, or as fine veinlets. Small globular aggregates of sulphides (< 2 cm) have also been observed in the small test pit on the site. The percentage of sulphide varies from trace to about 5%, but is rarely greater than 2%. Palladium, platinum, and gold are associated with the sulphides and also in a variety of tellurides, bismuthides, and alloys.

12. EXPLORATION

12.1 Exploration History to completion of DFS

There have been three major drilling programs at NorthMet since its discovery in 1969 and to date at least seven bulk metallurgical samples have been collected. Besides the 310 exploration drill holes totalling 261,226 feet shown on Table 12-1 there are 44 other holes totalling about 34,000 feet that were used for stratigraphic control. These additional holes were drilled as part of other projects, for hydrogeological purposes, or are water supply wells but information derived from them was used in developing a NorthMet geologic model. Figure 12-1 shows drill holes, unit boundaries, lease area, and 20 year mine pit outlines.

In preparation for the DFS, geologic work focused on a careful and total verification and re-compilation of the project drill hole database. This involved rigorous verification and validation of all drilling metadata, drill locations, down hole survey data, lithology, and assay data, and organisation of all related records. This resulted in an increase in the number of acceptable database assays from 12,000 to around 17,200. Since that time, drilling has added a further 13,443 multi-element assay records to the DFS database.

This verified database was used as the basis for a revised Resource Estimate and planning of 2005 drilling program which entailed:

- drilling and sampling 109 holes totalling 77,000 feet;
- collection of a forty ton metallurgical bulk sample for pilot-scale testwork;
- drilling into the projected sidewalls of the open pit to gather geotechnical data;
- sampling of un-sampled sections of previously drilled core; and
- extensive collection of waste rock characterisation data for environmental purposes.

Table 12-1 Summary of Exploration Drilling and Sampling

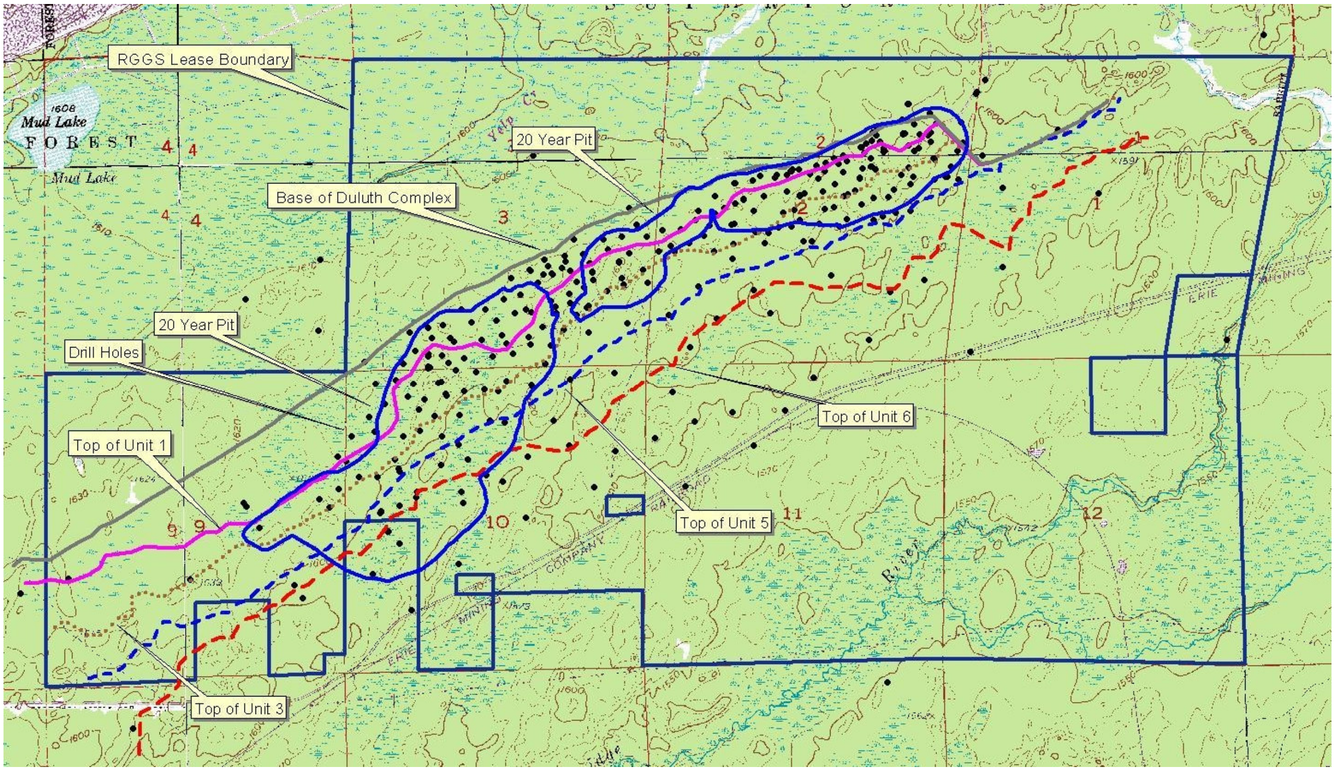
Company	Drilling Date	Date of Assaying	Number of drill holes	Total drilled footage	Number of assay intervals used in database	Assayed footage used in database	Assay Laboratory
US STEEL	1969-1974	1969-1974 1989-1990 2000-2001 2005-2006	112	13,716	9,475	56,525	USX ACME ALS-Chemex
NERCO	1991	1991	2	842	165	822	ACME
PolyMet Reverse circulation	1998-2000	1998-2000	52	24,650	4,765	23,767	ACME
PolyMet Core drilling	1999-2000	2000-2001 2005	32	22,156	4,058	20,727	ALS-Chemex
PolyMet RC with core tail	2000	2000	3	2,696	524	2,610	ALS-Chemex
PolyMet Core drilling	2005	2005-2006	109	77,166	11,656	71,869	ALS-Chemex
Total Exploration Drilling			310	261,226	30,643	176,347	

Drilling during 2005 had two main objectives;

- resource in-fill drilling focused on the area to be exploited during the early stages of a mining operation with the object of reducing average drill spacing to between 200 – 300 ft.; and,
- recovery of approximately 45 tons of material from which to select a 40 ton metallurgical test sample.

Additionally, because metal grades and geology are well known from previous drilling, the additional aims of that program were to more certainly define the geological structure, and to increase the level of confidence in mineralization continuity, long- and short-range variability and grade estimation. The opportunity was also taken to gather geotechnical data by logging each new hole and to collect waste characterization and hydrogeological data.

Figure 12-1 Plan showing drill hole collar locations, 20-year Ultimate Pit, geologic contact boundaries and topographic features.



13. DRILLING

13.1 Drilling History

To completion of the DFS, several exploratory drilling campaigns have been conducted; namely,

- US Steel (USX), 1969-1974,
- PolyMet Mining Inc.
  - reverse circulation (RC) drilling and core drilling in 1998-2000
  - two phases of core drilling in 2005,
- NERCO Minerals Co. two pairs of twin holes in 1991.

This drilling totals 261,227 feet over 310 holes from which over 30,700 acceptable assays have been taken (182,651 feet assayed) for inclusion in the drill hole database. Figure 13-1 shows collar locations for all drill holes on and around the projected mine site. Table 12-1 gives a breakdown of years, footages, and number of assays for all project drilling.

In addition to this resource focused drilling, over 3,400 feet of hydrogeologic drilling was carried out in 2005 for groundwater study purposes. In addition, over an extended period of time, about 26,000 feet were drilled at the periphery of the project area by other companies not directly associated with the NorthMet Project. This drilling provided geologic and stratigraphic information and was used to develop an understanding of the area and structure of the deposit though no assays from this drilling are incorporated in the NorthMet database.

Approximately 89.5% of Unit 1 and about 57% of the upper units have been sampled across the deposit. The sampled percentages are higher in the anticipated area of mining.

Sampling in Unit 1 (the main mineralized zone) is now mostly continuous through the zone for all generations of drilling. The PolyMet RC and core holes have continuous sample through the upper waste zones (which do have some intercepts of economic mineralization). Work in 2005 essentially completed the sampling of historic USX core within the area likely to be mined. This broad sampling limits the possibility of location bias in the sample set.

Resource consultant Dr. Phillip Hellman (Hellman, 2005) compared assays from RC drilling with those from nearby core drilling and found no significant bias between the sample sets.

13.1.1 USX Core Drilling (1969-1974)

USX drilled a total of 112 holes totalling 133,716 feet between 1969 and 1974. A few of these wholly penetrate Virginia Formation and Biwabik Iron Formation, one hole goes to the Archean granite basement rocks in the mine area, and five go to granite to the north of the mine area. PolyMet used the Virginia Formation holes for stratigraphic control, along with some assays from the Virginia Formation for waste characterization work.

USX assayed about 2,200 samples mostly with 10 foot sample lengths. All but 14 holes were vertical, and the angled holes more or less referenced grid north (~325°) and ranged from 40° to 60° in dip. Acids tests were done for the angled holes, but no other down hole records have been found.

USX geologists logged the holes, and sampled those parts with visible mineralization, amounting to about one-sixth of the total USX drilling. Their sampling goal was development of an underground resource, rather than open pit, hence only the most continuous, high grade zones were sampled. PolyMet has since sampled virtually all available USX core in the area of anticipated mining, as well as some outlying areas. Some deep holes outside the expected mining zone remain to be sampled and property wide, over 50% of the Duluth Complex intercept has been sampled.

The original USX drilling was assayed at their own laboratories in Minnesota. The later re-assaying of pulps and coarse rejects, and work on previously unsampled USX core, was carried out by PolyMet using ACME and later ALS-Chemex analytical laboratories.

Virtually all of the USX core from this program exists and is available for further sampling. However, USX did "skeletonised" the upper, (apparently) unmineralized parts of seventeen holes after assaying, retaining only one foot of core for each five or ten foot of unsampled run.

13.1.2 NERCO Drilling (1991)

In 1991 two sites were drilled for metallurgical holes by NERCO (in partnership with Fleck Resources Ltd. - precursor to PolyMet), which twinned USX holes 26086 (east end) and 26101 (west end), and were subsequently labelled 26086A and 26101A (Pancoast, 1991). At each site a BQ size (1 7/16") and PQ size (3 11/32") hole was drilled. The two BQ holes were split and assayed at 5 foot intervals from bedrock intercept to end-of-hole. The core from the PQ holes was shipped directly to Lakefield for compositing and testing on the assumption that the assays would match those in the BQ holes.

All assays for the NERCO drilling (165) were done by ACME Analytical Laboratories Ltd., Vancouver, B.C. (ACME). No sulfur assays were performed for these samples.

13.1.3 PolyMet Reverse Circulation (RC) Drilling (1998-2000)

PolyMet used an experienced local contractor to carry out reverse circulation drilling between 1998 and 2000 with the combined objects of collecting metallurgical sample and improving resource definition. In order to validate the method, a number of RC holes were drilled close to selected USX holes and the results compared. Once validated, 6 inch diameter RC drilling was carried out in the eastern and central areas of the deposit to provide sufficient material for metallurgical testwork. Individual samples comprised two 1/16 parts taken from each 5 foot run, one 1/16 part being used for assay and the other for archive purposes. The initial assaying was done by ACME laboratories, with ALS-Chemex for check assays. PolyMet has retained archive logging sample from all RC drilling and while the MDNR may still have 1/16 samples from this drilling in their Hibbing warehouse the other pulps and coarse rejects are thought to be lost.

Three RC holes (99-304BC, 99-305BC, and 99-310BC) were subsequently re-entered and deepened with AQ core.

13.1.4 PolyMet Core Drilling (1999-2000)

Some core drilling was carried out after completion of the RC program, mostly as in-fill drilling with a small component of exploration / expansion of the area previously drilled by PolyMet. This drilling totalled 32 core holes for 22,156 feet. All of these core holes were drilled by local drilling contractor, IDEA International (IDEA), using Hagby drill rigs with BTW size (19 holes) and NTW size (13 holes). All holes were vertical.

13.1.5 PolyMet Core Drilling (2005)

PolyMet’s 2005 drilling program covered 109 holes for 77,165 feet. These included fifteen 4 inch diameter holes for metallurgical sample (6,974 feet) drilled by Boart-Longyear of Salt Lake City.

In early 2005, IDEA drilled twelve PQ sized holes (3.3 inch) for 6,897 feet, mostly for metallurgical bulk sample, but with a few holes intended as resource in-fill. IDEA also drilled fifty-two NTW sized holes (2.2 inch) for 41,403 feet, and thirty NQ2 sized holes (2.0 inch) for 21,892 feet. The “N” size core was drilled in February-March and September-December of 2005.

In total about 11,650 multi-element assays were collected from the 2005 drilling program. Another 1,790 assays were obtained from assaying of previously unsampled USX and older PolyMet core. All assaying was carried out by ALS-Chemex laboratories.

Of the 109 holes drilled in 2005, 93 were angled. Of these, sixteen NQ2 sized holes were drilled across the deposit and had their core oriented core for structural and geotechnical study purposes. These holes targeted expected pit wall positions, as defined by Whittle pit shells, rather than mineral resource.

Large diameter holes for PolyMet’s 2005 metallurgical sample drilling targeted USX or previously drilled PolyMet holes with known mineralization of approximately the average grade of the deposit. This was to minimize the risk of non-productive drilling and to maximize recovery of material at grades and composition that would be representative of the deposit.

13.1.6 Stratigraphic Drill Holes

A number of holes drilled outside the NorthMet deposit were used to broadly define stratigraphy and structures though no assays are included in PolyMet’s database. These stratigraphic holes included the following:

- Six USX iron ore holes to the north of the deposit,
- eleven Bear Creek / AMAX holes from the Babbitt Deposit to the east,
- three Humble Oil / Exxon holes to the south,
- three INCO holes to the west; and,
- one Bear Creek hole to the west.

There are two water wells included in the data base, WW-1 and WW-2. In January-February 2005 these were drilled near RC holes that were used as water supply wells in prior drilling campaigns. WW-1 produced little water and is capped, but not abandoned; WW-2 produced useable quantities of water for drilling and was used throughout the 2005 drill program.

There are a further twenty-one holes drilled under the supervision of Barr Engineering for hydrogeological studies. Collar location and depth to bedrock information from these is contained in the drill hole database.

13.1. 7 Sample Recovery

RC Sample Recovery

PolyMet has found no definitive record assessing or quantifying recovery from the RC drilling. However, a partial listing of sample weights from the preparation laboratory is available which, combined with written comments by Gatehouse of North Mining contained in a due diligence report, leads PolyMet to estimate that RC sample recovery was greater than 85%.

It is believed that because North Mining did not make reference to RC recovery in their due diligence and feasibility study scoping reviews, and because North personnel were on site at the time of this drilling, there was no issue or concern about RC recovery.

Diamond Drill Core Recovery

Core recovery as recorded by USX was upwards of 99% on about 12,000 intervals. PolyMet (1999-2000) recorded recovery on about 2,400 intervals averaging about 97%, with over 94% of the intervals showing 90% or greater recovery. In 2005, PolyMet estimated core recovery to be about 100% with about 97% of the intervals (n~7,075) showing recovery of greater than 90%.

Zones of poor core recovery are rare in the Duluth Complex. There is seldom any significant or deep weathering at the bedrock sub-surface though the first few feet of some holes show some horizontal fracturing (possibly due to glacial action) and slight to moderate weathering. Most drill core is described as fresh, with more minor alteration than weathering.

13.1.8 Drill Hole Spacing

In general, the early drilling by USX is widely spaced but comparatively regularly distributed (approximate 600 ft x 600 ft grid), with some omissions that left substantial areas undrilled (more so down-dip, particularly in the eastern parts of the deposit). Subsequent programs, largely by PolyMet, focused on extracting metallurgical sample, and on proving the up-dip, more readily accessible parts of the deposit.

Drill spacing in the deepest known section of the deposit (greater than 800 ft below surface) is approximately 1,200 ft x 1,200 ft. The deposit is open at depth and while the deeper parts of the deposit (below about 1,000 ft below surface) may be of interest in the future they are considered to fall outside the scope of current development plans.

Within the currently defined 20-year DFS ultimate pit existing drill hole spacing is approximately 283 feet. Once the drilling planned for the winter drilling season of 2006 – 2007 is added average drill hole spacing will be reduced to about 247 feet.

13.2 Drilling - Post DFS

PolyMet considers that, to date, sufficient resource drilling has been completed for DFS purposes. The drill hole database has been updated to include all checked and validated drill hole data collected up to this point and the resource model has been revised and updated accordingly. A revised Resource Estimate has been produced which provided the basis for Whittle pit optimization work which, in turn, was used to produce a Mineral Reserve Estimate. Both the Mineral Resource Estimate and the Mineral Reserve Estimate comply with CIM guidelines and NI 43-101 requirements.

However, resource estimation and mine planning carried out for the DFS have indicated significant potential to further upgrade the resource and reserve estimates with corresponding potential benefits with respect to optimisation of the mine plan and the mine production schedule. Accordingly, PolyMet plans to carry out additional diamond drilling during the winter of 2006-2007 to upgrade, in terms of estimation confidence, material currently categorized as Inferred material and to provide better definition to the near-surface mineralized zone known as the Magenta zone. Magenta zone was recognized in earlier drilling but drilling during 2005 resulted in better definition and its promotion from geological curiosity to potential economic significance.

The planned 2006-2007 winter drilling campaign includes up to 110 holes totalling about 30,000 feet with an average hole depth of about 270 feet. Since the purpose and position of drill holes will be continually reviewed and adjusted on the basis of actual results as drilling progresses, it is not possible at this stage to be precise about the total number and ultimate depth of drilling. Core diameter will be NQ or near equivalent. There is no intention at this time to use this drilling to collect additional material for metallurgical testwork though assay rejects and half core remaining after sampling could be used to provide limited amounts of test material, if required. As with PolyMet's previous drilling campaigns, core will be geologically and geotechnically logged prior to cutting and sampling. Core handling and sample preparation procedures developed for previous campaigns proved highly successful at assuring sample quality and minimizing mislabelling and handling errors; therefore, similar procedures will be used for the planned 2006-2007drilling campaign.

14. SAMPLING METHOD AND APPROACH

Although the following description describes DFS-related drilling campaigns, the same approach and methods will apply to the post-DFS in-fill drilling campaign planned for the winter of 2006-2007.

All core was photographed and logged for both geological and geotechnical data.

After logging and marking up by an experienced field geologist, core was cut and sampled over virtually all the basal mineralized horizon (Unit 1) and wherever visible sulphide mineralization occurs outside the main mineralized horizon. Consistent with the majority of earlier work, the standard sample interval was 5 feet, adjusted such that samples did not cross geological boundaries and contacts. Samples were analyzed for copper, nickel, sulfur and platinum group elements and for a suite of other elements using an ICP-AES multi-element exploration analysis package. Systematic sampling was also conducted in waste, primarily for waste characterization, with all waste material analyzed in 10 foot increments. Many samples for waste characterization were subject to whole rock and trace element analysis. A total of 12,000 new samples were generated during this program.

15. SAMPLE PREPARATION, ANALYSES AND SECURITY

15.1 Sample Preparation

An integral part of the DFS was to review and assess in detail the sample preparation and analytical procedures used during the various drilling campaigns prior to PolyMet's most recent campaign. This was important because no one involved in PolyMet's 2005-2006 drilling campaign had any first hand knowledge of earlier campaigns.

After retrieval from the drill rig, all drill core was washed, photographed, logged for geology and geotechnical data, and sample intervals flagged by PolyMet geologists for cutting. Half core was sent for analysis and the remainder retained for future reference. Core was cut by diamond saw on-site, sample numbers assigned, and samples bagged and sealed by PolyMet employees. Samples were transported to the selected analytical laboratory (ALS-Chemex) where they were prepared for analysis under the quality assurance program of that laboratory. The chain of custody protocol had the laboratory take responsibility for the bagged and tagged samples at PolyMet's sample preparation facility. Prior to starting the 2005 drilling campaign, an independent expert, Dr. Phil Hellman, reviewed the field, core logging, core handling and sample preparation procedures for quality assurance and concluded they were both suitable and adequate.

PolyMet geologists worked closely with the drillers throughout the campaign to ensure core box numbering and hole depth were recorded correctly. PolyMet geologists also closely supervised drill rig set up after each relocation to ensure collar position and hole inclination were correct.

In the case of large diameter core for metallurgical sample recovery, all core was photographed and logged in the same way as for the smaller core; however, cutting was modified to ensure the maximum amount of material was available for test work without compromising analytical sample quality.

For the 2005 drilling program a chain of custody protocol was established which required the nominated sample preparation and analytical laboratory (ALS-Chemex) to collect all samples at the PolyMet site in Hoyt Lakes and take them directly to the ALS-Chemex preparation laboratory in Thunder Bay, Ontario. From there pulps for assay were flown to the ALS-Chemex chemical laboratory in Vancouver while the coarse rejects were returned to PolyMet.

For the NQ2 and NTW size core, a normal half core was sent for assay. Generally this was a five foot sample in Unit 1 or where there was visible (economic) mineralization, and ten foot samples in all other cases.

Because of its substantial weight per unit length and larger diameter, the standard boxes used for NTW/NQ2 core could not be used for the PQ and four inch diameter core intended for metallurgical sample. Instead, special, large diameter core boxes were used which accommodated between 4 and 5 feet of core compared with a standard NQ2/NTW core box, which typically contained approximately 10 feet. Each large diameter core box weighed approximately 60 lbs when full, which was considerably more difficult to handle than the smaller diameter core boxes. To facilitate preparation of metallurgical composites and to minimize re-handling and re-packing, it was decided to collect an assay sample to represent each box of large diameter core separately. This way a composite was prepared using entire boxes rather than by taking varying intervals from a variety of different boxes. This significantly reduced the possibility of mislabelling errors and made handling easier. Moreover, because each box sample was approximately 5 feet long it was felt that a composite made up of several boxes (rather than a series of standard 5 foot samples used for resource definition) would to some extent represent the dilution effect expected during mining by virtue of the larger sample interval.

The 4 inch diameter and PQ cores had 1/4 of the core cut and removed as two adjacent 1/8 pie slices along its length. One 1/8 slice was sent to the assay laboratory, one 1/8 slice was re-boxed at PolyMet as a geological reference sample. All boxes had the run blocks and sample tags inside, with the sample numbers written on the outside of the box.

Where two 4 inch or PQ holes were drilled at the same site and the upper portion appeared to pass through waste rock only, that upper interval was sampled in only one of the two holes.

Where core field duplicates of large diameter core were prepared, two 1/8 slices were used. This varies from the practice of using two 1/4 cores on the BQ (USX), NQ2, and NTW core sent out in 2005-2006.

In 2005 and 2006 PolyMet essentially completed sampling of existing USX core in the area likely to be mined, and completely sampled a number of holes from bottom to top with previous minimal sampling in the down-dip area of the deposit. Future mine planning may require more sampling of historic cores at the periphery of the currently planned pits.

PolyMet greatly expanded the overall geochemistry data set during the 2005 drill program, the majority of the approximately 400 standards submitted were also tested by an aqua regia digestion and for LECO furnace sulfur to get better reference data on the performance of (and differences between) these methods. About 1,300 aqua regia digestions were performed on core samples that were also assayed by four acid methods; about 700 of these had whole rock analysis, and 250 of those had Rare Earth Element analyses. Most of these samples were also tested for specific gravity/density and Leco furnace sulfur.

15.2 Assay History

15.2.1 General

There are eight generations of sample preparation and analyses that contribute to the overall project assay database:

- Original USX core sampling, by USX, 1969-1974;
- Re-assaying of USX pulps and rejects, selection by Fleck and NRRI, 1989-1991;
- Sampling of previously unsampled USX core, sample selection by Fleck and NRRI in 1989-1991;
- Sampling of two NERCO drill holes in 1991;
- Sampling of RC cuttings by PolyMet in 1998-2000;
- Sampling of PolyMet diamond drill core in 2000;
- Sampling of previously unsampled USX core (sample selection work done by NRRI, done in two phases) in 1999-2001.
- Sampling of PolyMet core from 2005 drilling, as well as continued sampling of previously unsampled USX core in 2005-2006.

Employees of PolyMet (or Fleck Resources) have been either directly involved in or have supervised all sample selection since the original USX sampling. Sample cutting and preparation of core for shipping was done by PolyMet or contract employees. Reverse circulation sampling was done by, or in cooperation with, PolyMet employees and drilling contractor employees.

The USX core has been stored, either at the original company warehouse in Virginia, Minnesota during drilling, or more recently at the Coleraine Minerals Research Laboratory (now a part of the University of Minnesota). The USX core was secured in dry, well sorted, locked buildings within a fenced area that is locked at night. The NERCO BQ size core is also stored at this facility.

The 1998-2000 PolyMet campaign core and RC reference samples were stored in leased warehouse in Aurora, Minnesota during drilling and pre-feasibility. These were later moved to a warehouse in Mountain Iron, Minnesota from 2002 until 2004. They were again moved to a secure warehouse at PolyMet's core logging and sample preparation facility at the Erie Plant site near Hoyt Lakes. Since that time access to the core logging and storage facility has been limited to PolyMet employees.

Core from the 2005 campaign has been processed by PolyMet employees at the core logging and sample preparation facility at the Erie Plant site which is kept locked when not in use. The whole plant site area has 24 hour security with controlled access.

PolyMet has historical fee schedules for ACME and ALS-Chemex laboratories which detail assay methods and detection limits. Also available is some of the original sample submission documentation going back to 1989. All documentation including laboratory sample handling and assay protocols and procedures that pertain to PolyMet's 2005 drilling campaign are stored at PolyMet's project offices near Hoyt Lakes.

15.3 Assay Methods - PolyMet 2005-2006

15.3.1 ICP

Two assay methods were used. All samples were subjected to four acid ("total") digestion (HF-HNO3-HClO4 digestion with HCl leach and ICP-AES for 27 elements, ALS-Chemex code ME-ICP61), in addition, 1 in 10 samples was analyzed by an aqua regia ("partial" digestion, ALS-Chemex code ME-ICP41) method for 34 elements for comparison with older data and any data generated during metallurgical testing.

15.3.2 Precious Metals (AuPGM)

Samples were analyzed for platinum, palladium, gold (AuPGM) by fire assay with an ICP-AES finish on 30 gram samples. ALS-Chemex standard code for this method is "PGM-ICP23."

Four Acid Digestion vs. Aqua Regia Digestion

In the 2005-2006 sampling program, PolyMet switched from historic Aqua Regia digestion to a four acid method. Previous comparisons had shown that the Aqua Regia method had probably understated the copper and nickel contents by a about 5%, but more importantly the four acid method gave a more complete digestion and therefore would be expected to provide better results in assessment of standards.

For copper and nickel the change in digestion method has shown no significant change in copper and a very slight increase in nickel, based on 275 standards where both methods were used. No factoring was used to convert any project metals values.

LECO Furnace Sulfur vs. ICP Sulfur

Prior to the 2005-2006 campaign, PolyMet had about 14,800 samples where both ICP sulfur (aqua regia digestion) and Leco furnace sulfur had been analyzed. In the 2005-2006 sampling program the Leco test was done on about 1 in 8 samples (including most standards). Because of the switch to four acid (total) digestion, a factor based on the relation of the four acid ICP sulfur to the Leco method was needed for modeling and environmental purposes. Analytical Solutions Ltd of Toronto calculated this factor with validation by SRK of Vancouver. Essentially, for sulfur values below 2.0%S, the 4 acid digestion ICP value can be used directly whereas for values above 2.0%S, the four acid ICP value should be multiplied by 1.08 to arrive at a value consistent with expected Leco method values.

15.4 Assay Quality Control

During the DFS, PolyMet staff carried out an exhaustive assessment of quality control procedures used during earlier drilling campaigns and used this information when verifying and validating the drill hole database. This section covers describes only what was done during the 2005 – 2006 drilling campaign to assure assay quality.

15.4.1 PolyMet Assay Quality Control (2005-2006)

Throughout the 2005-2006 program, careful attention was given on quality control and record keeping. In preparation for that drilling campaign three property specific standards were created in late 2004 from coarse rejects of USX samples, blanks were created from iron-formation, field duplicates were prepared from core, coarse reject duplicates were run at the laboratory, and pulp duplicates were also done by ALS-Chemex. No check assays were done through other laboratories as ALS-Chemex performance was determined to be reliable relative to the "round robin" expected values calculated by Analytical Solutions Ltd. (ASL).

After determining through comparative analysis that the material had not deteriorated during storage, PolyMet used coarse-reject material from 63 USX samples to create three property specific standards. Table 15-1 compares the 2004 ALS-Chemex assay results of each standard against the calculated composite USX assay values determined in the period 1969 to 1974. In each case the newer value reported is a composite of twenty samples and in the case of PolyMet's 2004 work, each sample was assayed using the 4-acid method.

Table 15-1 Analytical Standards: ALS-Chemex 2004 assays compared with older USX assays

	Cu %	Ni %	S %
Standard 1 expected value based on 1969 to 1974 USX assays	0.18	0.08	1.04
Standard 1 assayed value-2004 - Chemex	0.20	0.11	1.08
Standard 2 expected value based on 1969 to 1974 USX assays	0.36	0.14	0.88
Standard 2 assayed value-2004 – Chemex	0.37	0.15	0.82
Standard 3 expected value based on 1969 to 1974 USX assays	0.55	0.18	1.17
Standard 3 assayed value-2004 – Chemex	0.57	0.21	1.04

Approximately every twelfth sample submitted to ALS-Chemex in 2005 was a standard, blank, or field duplicate. The low, medium, and high grade standards were distributed as best as possible to match the expectation of grade in the surrounding samples. Chemex ran a crusher duplicate every 20 samples, and a pulp re-run every 10-12 samples.

In summary, ALS-Chemex had very few standards or blank failures (a failure is an assay result varying by more than 10% from accepted “round robin” value), none on copper-nickel assays, one on sulfur, and a scattering on PGE. Because PGE’s have a wider range of values than the base metals these failures are not considered as critical as those for base metals if the failures do not seem to be systematic.

15.4.2 Standards

Standards were prepared at CDN Labs of Delta, British Columbia. The three standards were prepared separately and in an identical manner. The sample was ground to 200 mesh, screened and the oversize re-ground. The +200 mesh fraction was bagged and ultimately returned to PolyMet. The -200 mesh fraction was mechanically mixed for 72 hours. Twenty samples of each were cut from the standard and sent to ALS-Chemex for assay. Homogeneity was checked and approved by both PolyMet and resource consultant Dr. Phillip Hellman. The standards were then bagged in lots of approximately 110 g. in tin-top kraft paper bags and shipped back to PolyMet for further testing and use.

15.4.3 Coarse Blanks

The material used for coarse blanks was “Biwabik Iron Formation”, which comprises quartz, magnetite, hematite, iron carbonate and minor iron silicates. Material selected was minus 2 inch material taken from a crushed rock pile at the Erie Plant site. This material had been originally crushed for road- and rail-bed fill. The material was recovered by front end loader and transported to the PolyMet core preparation facility. Once there, it was only handled with plastic tools (to avoid possible contamination from metal tools). Each sample was weighed to approximate a typical weight of submitted core sample, and for anonymity, samples were shipped in the same type of containers used for core samples.

15.4.4 Duplicates

Field Duplicates

Field duplicates were quarter core for BQ, NQ2, and NTW size core and one eighth samples for PQ and 4 inch core.

Laboratory Duplicates

Chemex performed two duplication steps in the 2005-2006 work, crusher duplicates and pulps.

15.4.5 Chain of Custody

During the 2005 program all samples were collected by ALS-Chemex at the PolyMet core facility and transported to Thunder Bay for sample preparation before being air freighted to the ALS-Chemex Vancouver laboratory for analysis. All coarse reject material was returned directly to PolyMet by ALS-Chemex. Core continues to be kept in PolyMet’s own secure, locked warehouse facility at the Erie Plant near Hoyt Lakes.

15.4.6 Laboratory Audits

Dr. Barry Smee audited the ALS-Chemex Thunder Bay sample preparation laboratory in 2004 at PolyMet's request. Richard Patelke, PolyMet's chief geologist, visited the laboratory in May 2005 and observed PolyMet samples being processed. This was an informal review based on a checklist supplied by Dr. Smee. No issues were noted and all work observed was being done in accordance with the contract and ALS-Chemex standard procedures. Geochemical and quality assurance consultant, Lynda Bloom, principal of Analytical Solutions Ltd. (ASL) visited ALS-Chemex in Vancouver in 2005 to review procedures and observe laboratory practices. No matters of concern were identified.

15.5 Nickel in Silicates (Laboratory Assay versus Recoverable Nickel)

It has been characteristic of NorthMet and other Duluth Complex deposits to show lower nickel recoveries in process test work than would be expected from laboratory assays on drill core. Generally, there is a loss of about 25-35% of the nickel compared to drill core assays when concentrating sulphides. From previous work, it is known that small amounts of unrecoverable nickel occur as a magnesium-iron-nickel silicate [$(Mg,Fe,Ni)_2\ SiO_4$] that is tied up in the mineral olivine, which is one of three significant gangue minerals that occur across the NorthMet deposit. Testwork has shown that most of the very small amount of nickel contained in silicates would not be recovered during the autoclaving process proposed.

Mineralogical studies show that approximately 25% to 35% of the rock in NorthMet is composed of olivine. Previous microprobe studies, plus work by PolyMet in 2006, have shown an average of about 0.10% nickel in olivine. The approximate nickel grade of the PolyMet metallurgical bulk samples is 0.10%. Because the average nickel in the olivine is the same as the average nickel in the bulk samples, the unrecoverable nickel in the olivine would be expected to reduce nickel recovery by the amount of olivine in the bulk sample, namely 25% to 35%. Nickel recoveries on the six PolyMet metallurgical bulk samples have ranged from 69% to 77% which, is in line with an approximate 25% to 35% loss of nickel to silicate.

16. DATA VERIFICATION

Great attention was paid during the 2005 drilling program to sampling and assay quality assurance and as a result there were very few standard or blank failures (a failure is an assay result varying by more than 10% from the accepted 'round robin' value), no failures on copper-nickel assays, one on sulphur, and a small number on PGE. Because PGEs have a wider range of values than the base metals these failures were not considered significant. Analytical laboratory selection was made on the basis of the results from a 'round robin' sample analysis campaign overseen by Analytical Solutions Ltd. (ASL) of Toronto. As a result, ALS-Chemex of Thunder Bay, Ontario and Vancouver, B.C. was selected as the principal analytical laboratory. To assist with quality assurance, PolyMet retained an expert, independent third party consultant, Analytical Solutions Ltd. (ASL) to review procedures and to audit the selected laboratory. At the conclusion of the drilling campaign, ASL reviewed all analytical results and determined that no significant analytical bias or other systemic errors were detectable in the assay results.

The drilling program was quality controlled and quality assured by insertion of property specific standards, blanks of similar matrix, field duplicates, as well as laboratory preparation duplicates. Three standards covering low, average, and high grade material were prepared from sample recovered from previous drilling programs. These standards were subjected to multi-laboratory, round-robin analyses and statistical analysis. Blanks consisted of material from a nearby iron mine. Field duplicates were quarter core, with half of core retained. Samples or batches of samples failing quality assurance controls were re-analyzed.

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17. ADJACENT PROPERTIES

The DFS, which is the subject of this Technical Report, is focused entirely on mining and processing material from the NorthMet Deposit. PolyMet does not control or have interests in any adjacent properties, neither is it currently planning to explore or drill any properties in which it does not already have a direct interest .

18. MINERAL PROCESSING AND METALLURGICAL TESTWORK

18.1 Ore Mineralogy

The ore mineralogy was derived at Lakefield using proven QEMSCAN technology. Analysis shows copper is present in chalcopyrite and cubanite (both copper-iron sulphides), nickel is present in pentlandite (a nickel-iron sulphide) and in solid solution in gangue minerals, and iron in pyrrhotite (an iron sulphide). Palladium, platinum, and gold are associated with the sulphides. There are only three major gangue minerals in the remainder of the suite: feldspar, olivine and clinopyroxene. The mineralogical proportions of minerals were very similar between all composites and are shown in Table 18-1.

Table 18-1 Typical Mineral Composition of NorthMet Ore

Mineral	Formula	Average %
Pentlandite	$(Fe,Ni)_9S_8$	0.24
Chalcopyrite	$CuFeS_2$	0.58
Cubanite	$CuFe_2S_3$	0.12
Iron Sulphides (Pyrrhotite)	$Fe_{(1-x)}S$	0.58
Feldspar	$(NaSi,CaAl)AlSi_2O_8$	54.1
Olivine	$(Mg,Fe)_2SiO_4$	26.5
Clinopyroxene	$[Mg,Fe,Ca][Mg,Fe]Si_2O_6$	6.10
Minor Gangue Minerals		11.8

18.2 Introduction to Metallurgical Testwork

The aim of the DFS testwork program was to develop and demonstrate a complete process flowsheet for treatment of polymetallic sulphide material from the NorthMet deposit with an average head grade of approximately 0.31 % copper, 0.09 % nickel, 0.08 g/t platinum, 0.28 g/t palladium and 0.04 g/t gold. The flowsheet arising from this testwork subsequently served as the basis on which the plant was designed to process 32,000 short tons (29,030 metric tonnes) per day or 11.68 million short tons (10.6 million metric tonnes) per year of run of mine (ROM) ore.

The process route selected for recovering the base metals and AuPGMs is based on the mineralogy and involves an initial concentration step to recover the sulphide minerals and AuPGMs by crushing, grinding and bulk sulphide flotation. The bulk sulphide concentrate is then treated by a hydrometallurgical process that includes chloride-assisted pressure oxidation leaching (POX) with subsequent metal recovery. Copper is recovered as LME grade cathode. Nickel and cobalt are recovered together as a mixed hydroxide precipitate. The gold and PGM are collected in a precipitate with some copper and sulphur. The mixed hydroxide precipitate and gold-PGM precipitate are refined off-site by off-take parties. The advantage of this hydrometallurgical method is that all the base metals and AuPGMs are extracted in a single step (the chloride assisted POX) and can be subsequently separated and recovered onsite.

The metallurgical plant features the following unit operations:

- Ore Crushing;
- Ore Grinding;
- Sulphide Flotation;
- Flotation Tailings Disposal;
- Flotation Concentrate Grinding;
- Pressure Oxidation of Finely Ground Flotation Concentrate;
- Filtration and Washing of POX Residue;
- AuPGM Precipitation from pregnant leach solution (PLS) using CuS to form an AuPGM Concentrate for offsite processing;
- Primary Neutralisation with Limestone to produce Clean Gypsum;
- Copper Recovery via conventional solvent extraction and electrowinning (SX/EW) to produce LME Grade A copper metal;
- Oxidation and Neutralisation (with limestone) of a portion of the Copper Solvent Extraction raffinate to produce a residue consisting of a mixture of gypsum and iron and aluminium hydroxides;
- Nickel, cobalt and zinc are recovered as a 'mixed' hydroxide precipitate using magnesium hydroxide precipitation;
- Magnesium removal using lime to precipitate magnesium hydroxide; and
- Hydrometallurgical plant residue disposal.

There are two waste streams from the ore processing plant. Flotation tailings are pumped directly to a separate storage facility. The hydrometallurgical plant residue is formed by mixing the final POX residue with gypsum, iron/aluminium hydroxide and magnesium hydroxide residues. This combined residue is placed in the lined hydrometallurgical residue facility.

The development and demonstration of this process has taken place via several integrated pilot plant testwork campaigns from as early as 1999. A thorough review of the most recent 2005 and 2006 testwork has been presented in the DFS with findings and conclusions from all pilot campaigns incorporated into the current plant design.

18.3 Testwork History

18.3.1 Testwork in 1997 and 1999-2000

PolyMet launched an intensive testwork program in 1998 and 1999-2001 to examine the potential for hydrometallurgical processing of the NorthMet ore. After extensive analysis, flotation of Cu, Ni, and AuPGM to a bulk concentrate followed by a high temperature, chloride-assisted POX approach was selected, and the process was fully demonstrated at the bench scale.

18.3.2 Pilot Plant Campaigns - 1999 – 2001

Pilot plant campaigns were completed in 1999-2001 at Lakefield to produce a bulk concentrate from the NorthMet ore and to investigate the recovery of Cu, Ni, and AuPGMs from the bulk concentrate. The flotation process to produce the bulk concentrate included rougher, scavenger and cleaner unit operations. The final bulk concentrate contained 14.7% Cu, 3.05% Ni, 32.9% Fe, 0.14% Co, 26.7% S, 1.41 g/t Au, 2.22 g/t Pt and 9.9 g/t Pd.

Bulk concentrate was ground to P_{80} of 15µm, a fine grind being important for complete extraction of AuPGM, and re-pulped to approximately 10% solids in an agitated vessel prior to injection into a six-compartment autoclave. The autoclave operated at conditions identified in earlier batch scale testwork to be optimum: 225° C, 690 kPa oxygen gas overpressure and 120 minutes residence time. The discharge residue was filtered and the pressure leach solution treated in a number of ways to recover AuPGMs from the PLS. The AuPGM depleted liquor was then stage neutralised to pH 2.0, using limestone, and copper cathode was produced via conventional SX/EW. A portion of the raffinate was bled from the circuit and set aside with the balance of the raffinate recycled as a cooling solution to the autoclave. The main autoclave pilot plant operated successfully for 14 days including a 10-day integrated run with Cu SX raffinate recycled back to the autoclave.

A further pilot plant was used to demonstrated a process for treatment of raffinate that included rejection of Al and Fe, and production of high purity nickel and cobalt metals by a solvent extraction and electrowinning process.

18.3.3 Testwork in 2005-2006

The 2005-2006 pilot plant program was overseen by Bateman and undertaken to confirm the entire metallurgical flowsheet feasibility from ore processing to final product recovery, to provide the design basis for the process plant, to collect extensive environmental data and to optimise aspects of the process, in particular:

- Increasing sulphide recovery from the ore to the bulk flotation concentrate (to minimise environmental impacts of sulphide in tailings).
- Recycling of a portion of the leach residue to the autoclave for improved AuPGM extraction and autoclave design optimisation (reduced autoclave sizing).
- Precipitant selection and optimisation for iron reduction and AuPGM recovery.
- Investigation of an option to separate Co and Zn via solvent extraction prior to Ni hydroxide precipitation, as an alternative to precipitation of a mixed Ni-Co-Zn hydroxide product.

The pilot-scale testwork program evaluated continuous and fully integrated testing of the proposed flowsheet in several phases, accompanied by bench scale variability and optimisation testwork:

- Phase 1 – Comminution and Flotation
- Phase 2 – Leaching and Metal Recovery (Cu Cathode, AuPGM Precipitate and Ni-Co Mixed Hydroxide Precipitate) from the Phase 1 Flotation Concentrates
- Phase 3 – Testing of Solvent Extraction and Electrowinning for Cu, Ni and Co and Precipitation for Separate Nickel, Cobalt and Zinc Product Recovery (Hydroxides)
- Phase 4 – Optimization Flotation and Autoclave Bench and Pilot Plant Testing in March–April 2006.

In 2005 a 44 short ton bulk sample of large diameter diamond drill core was delivered to Lakefield for flotation testwork and subsequent production of concentrate for hydrometallurgical pilot plant program.

Another nine short tons of drill core sample was provided in April 2006 for additional pilot scale testwork. Table 18-2 below indicates the head grade assay range for the composites prepared from these drill core samples, and these values are generally representative of the life of mine ore feed grade range.

Table 18-2 Composite Head Grade Assay Range

Sample	Cu, %	Ni, %	S, %	Pt, g/t	Pd, g/t	Au, g/t
Range of 4 test samples (composited from drill core)	0.31-0.40	0.095-0.11	0.73-0.91	0.05-0.09	0.23-0.33	0.03-0.08

18.3.4 Comminution and Flotation Testwork

Each composite was tested separately using optimised comminution and flotation parameters established in previous testwork.
The flotation pilot testwork provided bulk concentrate products for further hydrometallurgical testing.

Comminution Testwork

Comminution parameters were determined for the composites and show a high level of consistency. The ore can be broadly categorised as mildly abrasive and towards the higher end of the hardness scale. A review of the specifications of the existing crushing and grinding equipment has confirmed that it is more than capable of reducing the particle size to suit flotation at the required throughput.

Average values determined from Bond tests for the rod and ball mill work indices were 13.4 and 15.5 kWh/t respectively, and 0.40 for the abrasion index.

Flotation Laboratory Batch and Locked Cycle Testwork Outcomes

This work mimicked the flowsheet derived from past testing and confirmed that the metallurgical behaviour of all composites was consistent. The flowsheet adopted a standard rougher scavenger circuit followed by two stages of cleaning. A regrind mill on the combined scavenger concentrate and first cleaner tailing was also included to ensure middling particles (particles containing both sulphides and gangue) underwent further size reduction.

The testwork confirmed optimum flotation parameters for maximum sulphide and base metals recovery to the concentrate, and determined:

- A reagent regime, including:
    - flotation collector (potassium amyl-xanthate, PAX) dosage rate;
    - copper sulphate addition as an activator to enhance metal and sulphide recovery; and
    - combined frother of 3:1 MIBC:DF250.
- A selected grind size of 125 µm for flotation piloting feed.

- Total rougher and scavenger time of 15 minutes.

Flotation Pilot Plant Outcomes

A total of 53 short tons were processed, in four composite groupings. Flotation performance was similar for all composites and circuit changes were introduced to enhance sulphide recovery to concentrate and thus reduce sulphur content of the tailings.

The bench and pilot plant work confirmed the importance of copper sulphate ($CuSO_4$) as an activator for sulphide mineral flotation. The addition of copper sulphate to a conditioning step prior to the scavenger flotation step successfully reduced the sulphur grade of the final tailings to ≤0.15%, thus meeting PolyMet's objective of minimising possible environmental impacts of sulphur in tailings.

Table 18-3 shows flotation circuit performance for non-activated versus activated pilot-scale tests in 2005.

Table 18-3 Impact of Copper Sulphate on Pilot Plant Recovery

Description	Distribution, %					
	Cu	Ni	S	Pt	Pd	Au
Non-activated	94.3	69.3	72.4	69.1	75.8	58.5
Activated	94.2	72.5	82.2	67.5	83.1	57.9

The additional flotation pilot testwork undertaken in 2006 was able to confirm the reagent regime, provide a reduction in overall residence time (and hence circuit size), and attain similar metals recoveries and reduced sulphide in tailings. The additional work also led to refinement of the circuit to include a scavenger conditioning stage and splitting scavenger concentrate (first scavenger concentrate directly to the cleaner circuit and the second scavenger concentrate to the regrind mill before returning to the rougher circuit) to reduce the solids loading in the regrind circuit.

The range of grade for concentrates produced in the 2006 testwork is shown below in Table 18-4.

Table 18-4 Concentrate Composition from 2006 Piloting

Grade	Assays					
	Cu (%)	Ni (%)	S (%)	Au (g/t)	Pt (g/t)	Pd (g/t)
Concentrate	7.16-10.1	1.66-2.20	18.4-21.5	0.65-1.28	1.17-1.59	5.76-6.71

Flotation pilot testing covered a range of samples with head grades from 0.27% to 0.41% Cu and from 0.094% to 0.122% Ni. In the grade range tested, flotation recovery did not appear to change with head grade hence a constant flotation recovery was used.

Flotation tailings and concentrate were tested by Outokumpu Technology (solid-liquid separation equipment vendors) in a continuous, high rate thickening rig to determine flocculant and thickening design parameters.

A detailed mineralogical analysis was made on flotation tailings.

18.3.5 Hydrometallurgical Bench Testwork - 2005

Pre-Piloting

Hydrometallurgical pre-piloting bench testwork was conducted to optimise circuit conditions for the pilot-scale testwork, in particular temperature, residence time and reagent additions for a number of unit operations. The results of this testing were then incorporated into the pilot plant design and operating philosophy.

During and Post-Piloting Bench Testwork

A number of bench programs were undertaken to provide important information for final design. These included:

AuPGM stability studies **-** The stability of the leached Au and PGM species in the autoclave discharge were tested by timed sampling of slurry taken from the pilot plant discharge. This was important to confirm that the Au and PGM would not be re-precipitated and lost during the post autoclave solid-liquid separation steps. The stability of Au and PGM in solution was proven to be independent of agitation and temperature within the range of conditions tested.

Rheology - Rheology tests were carried out on slurry samples recovered from the Pilot Plant operation.

AuPGM concentrate upgrading **-** Autoclave testwork was conducted to upgrade the Au and PGM content of the AuPGM concentrate. This was done by selective re-leaching of base metals and sulphur from the AuPGM concentrate product, at both high and low temperatures. This work confirmed a window of temperature to upgrade the AuPGM precipitate (with an optimum average temperature at 195 °C). It was possible to upgrade the AuPGM precipitate from approximately 1000 g/t (Au+Pt+Pd) to 16,000 g/t (Au+Pt+Pd).

Co and Ni recovery from SX strip liquor - The separation of cobalt and nickel by solvent extraction (Phase 3 of the piloting referred above) was successful in producing separate and pure products. Cobalt was recovered in bench scale testwork as a cobalt hydroxide by treating the cobalt strip liquor with magnesium hydroxide slurry. Nickel precipitation was performed as part of the pilot plant continuous operation.

18.3.6 Hydrometallurgical Pilot-Scale Test Campaigns

The flowsheet tested during the August-September 2005 pilot campaign covered POX through to recovery of Cu, AuPGMs, Ni, Co and Zn. The autoclave feed material consisted primarily of the concentrates produced in the flotation piloting described above, as well as some concentrate remaining from year 2000 testwork. This concentrate, which had been carefully stored in a freezer, was used to extend the circuit running time and provide additional product for characterisation.
A separate pilot campaign was conducted in October 2005 to test an option for separate recovery of Ni, Co and Zn hydroxide products via a Co/Zn SX circuit.

An additional autoclave pilot program was performed in April 2006. This short program was designed to confirm the viability of recycling a portion of the autoclave leach residue for improved AuPGM recovery and shorter autoclave residence time (1.1 hours of residence time instead of the 2 hours used in the “non-recycle” configuration

The pilot plant design was developed by Bateman using a metallurgical flowsheet produced by METSIM modeling software. METSIM is an industry standard metallurgical simulation and design computer software package and METSIM models developed by Bateman were delivered to the Lakefield staff for design and operation of the pilot plant facilities.

As part of the hydrometallurgical pilot plant design, corrosion coupons were strategically placed in various parts of the circuit to obtain information on materials selection for the commercial plant.

Outcomes and conclusions from hydrometallurgical pilot plant work are summarised below in Table 18-5.

Table 18-5 Pilot Plant Test Outcomes and Conclusions

Flowsheet Area	**Pilot Plant Conditions and Outcomes**
POX	Optimum autoclave operating parameters included: operating at 225°C, ~3100kPag Total Pressure, ~800kPa O_2, 10g/L chloride and a 1.1 hour first pass residence time. Metals extractions were shown to improve by the introduction of a 200% residue recycle stream (i.e. a 2:1 ratio of leach residue to fresh feed). Average extractions for metals at optimum conditions were: Cu 99%, Ni 99%, Co 98%, Au 89%, Pt 93% and Pd 94%.
AuPGM Recovery	Au and PGM were precipitated from solution by adding CuS, recycled from the residual Cu recovery circuit. Recoveries were excellent with below detection limit values for AuPGM remaining in solution, and corresponded to a minimum precipitation efficiency of Au 88%, Pt 98% and Pd 99.5%. Further testwork led to the reintroduction of sulphur dioxide (SO_2) in the final flowsheet as a reductant for iron prior to CuS addition. This reduces the consumption of CuS and limits the elemental S content of the concentrate. The SO_2 pre-reduction system was tested and piloted in the year 2000 pilot plant at Lakefield.
Solution Neutralisation	This circuit operated to a pH of 1.3-1.4 using ground limestone addition while gypsum thickener underflow was recycled as seed to the first reactor. Analysis of the gypsum residue reported insignificant base metal content and low residual carbonate (0.07%).
Copper Solvent Extraction	Copper was extracted at 40° C in 3 counter current stages, scrubbed in 1 stage (to prevent chloride transfer to Cu electrowinning) and stripped in 2 stages. Two organic extractants, Acorga® M5640 and LIX® 973NS LV, were pilot tested. Orfom® CX80CT diluent was used in each case. Recovery of Cu to the strip liquor averaged 95.5% for both extractants, producing raffinate with Cu <1.0 g/L from PLS ranging 18-25 g/L Cu. No evidence of crud formation during testing was noted.

Flowsheet Area	Pilot Plant Conditions and Outcomes
Copper Electrowinning	A total of 69 kg of copper metal was produced. Cathodes were harvested twice during the campaign. Four cathodes were sampled for purity – 2 from each extractant cycle. Cathodes from Cycle 2 met LME grade A specifications while cathodes from Cycle 1 showed minor contamination of Pb and S attributed to erratic temperature control during test start-up.
Raffinate Neutralisation	Raffinate is neutralized prior to recycle of raffinate as cooling solution back to the autoclave. This is necessary to reduce the free acid level in the autoclave product solutions and prevent the formation of basic ferric sulphate (BFS). The pH set points for raffinate neutralisation varied between 1.2-1.5 and were controlled via limestone slurry addition. Loss of Ni and Co to the residue was minimal.
Iron Removal	A portion of neutralized raffinate solution was directed to nickel and cobalt recovery. The first step in the Ni and Co recovery circuit is iron removal by oxidation and neutralization. Ferrous iron was oxidised to ferric iron by addition of gaseous oxygen and were removed from solution (along with aluminium) by hydroxide precipitation. Limestone was added to achieve the target pH of 4.2. Iron removal residue consisted predominantly of gypsum with low levels of iron and aluminium hydroxides. Ni and Co losses in the residue were minimal. Iron and aluminium removal efficiencies were 99.9% and 94.1% for this circuit.
Aluminum Removal	A separate stage of aluminium removal was included in the pilot plant circuit. In practice, this circuit did not consume limestone, as pH naturally rose to 4.6-4.7 due to an excess of alkalinity from the iron removal stage. Iron and aluminium removal efficiencies were 71% and 96% respectively (to give overall precipitation efficiencies of nearly 100% after two stages).
Residual Copper Recovery	Residual copper was precipitated as copper sulphide (CuS) using sodium hydrosulphide (NaSH), and was collected for use in AuPGM recovery. Stoichiometric addition of NaSH was required for copper precipitation. Solution analysis confirmed precipitation of 92% of the Cu for this circuit, with insignificant co-precipitation of Ni and Co.
Mixed Hydroxide Precipitation Stage 1 (HP1)	Ni and Co were precipitated as a mixed hydroxide using magnesium hydroxide slurry to a target efficiency of 85%. The mixed hydroxide precipitates collected during the pilot plant analysed 31.5-36.3% Ni, 1.67-1.92 % Co, 0.31-0.37% Cu, 0.51-0.59% Fe, 4.27-4.84% Zn and 0.62-1.04% Mg.
Mixed Hydroxide Precipitation Stage 2 (HP2)	This circuit recovered residual nickel and cobalt from solution by precipitation with hydrated lime slurry at pH 8. Precipitate was thickened and recycled to the neutralisation circuit (where the residual metal hydroxides redissolved). Removal efficiency of residual Ni and Co from the feed solution averaged

Flowsheet Area	Pilot Plant Conditions and Outcomes
	93% and 92% respectively giving overall precipitation efficiencies through the two stages of hydroxide precipitation of nearly 100% for both Ni and Co.
Magnesium Removal	Magnesium was removed from the barren solutions after Ni and Co recovery by addition of hydrated lime slurry to pH 9. Mg precipitation was close to the target 50%. The magnesium hydroxide – gypsum product slurry was thickened, with overflow used as process water and underflow directed to tails. The absence of pay metals in the feed to magnesium precipitation resulted in negligible Ni and Co losses (0.14% and <0.02% respectively).
Co/Zn Solvent Extraction	The cobalt and zinc solvent extraction circuit was run as part of the campaign to produce purified metal hydroxides (rather than mixed hydroxide precipitation). Bulk Co/Zn extraction was achieved in 4 stages at pH 5.0-5.5 and 55° C, using 5 %v/v Cyanex® 272 extractant in Orfom® SX80CT diluent. The higher temperature favoured Co extraction and displacement of co-extracted Mg. Co stripping then proceeded in 3 stages at pH 3 and 45°C, before Zn stripping in 2 stages at pH <1 and 40°C. Co extraction rates greater than 96% were achieved, with raffinate grades of below 10 ppm Co. Zinc extraction was greater than 99.9%. No evidence of crud formation during testing was noted and the circuit operated smoothly.

A variety of specialist vendors for thickening, filtration and flocculant selection were present during piloting to perform bench tests on slurry samples withdrawn from the operating pilot plant. The results of this testing have been used to provide equipment design parameters.

Flotation and hydrometallurgical piloting provided data for the development of a flowsheet generating maximum overall base and precious metal recoveries to final marketable products. The DFS engineering design incorporates the data from the various pilot campaigns that provides confidence for the capital cost and operating cost estimates.

18.4 Process Flowsheet and Key Design Criteria

The simplified process flowsheet for the NorthMet plant is illustrated in the block flow diagram below.

Design criteria for the process plant were derived or obtained from a number of sources including testwork reports, vendor testing and/or recommendations, previous Bateman projects, consultants, contractors and from PolyMet. Key design criteria and key operating parameters are summarized below (see Table 18-6).

Key Design Criteria

Ore Feed: 32,000 st/d (1,333 st/h)

Plant Availability: 90.0%

Assumed Ore Grade and Process Recoveries

Table 18-6 Assumed Average Ore Grades and Recoveries for Metal Production

	Ore Grade	Concentrate Grade	Concentrate Recovery	POX Extraction	Overall Recovery
	(% / g/t)	(% /g/t)	(%)	(%)	(%)
Cu	0.40 %	10.8 %	94	99	92
Ni	0.11 %	2.48 %	73	99	70
Co	84 g/t	0.12 %	42	99	41
Au	0.06 g/t	0.96 g/t	76	89	67
Pd	0.30 g/t	7.26 g/t	80	95	73
Pt	0.06 g/t	1.64 g/t	77	95	75

* Note: Ore grades will vary in accordance with the mine plan

Design Metal Production:

Cu	38,821st/a
Ni	9,037 st/a
Co	400 st/a
Au	15,940 oz/a
Pt	16,475 oz/a
Pd	91,468 oz/a

Crushing:

Number of stages	4
Feed to crushers F_{80}	740 mm
Primary crusher discharge P_{80}	83 mm
Secondary crusher discharge P_{80}	39 mm
Tertiary crusher discharge P_{80}	11.4 mm
Quaternary crusher discharge P_{80}	8 mm

Milling:

Rod Mill Work Index	13.4 kWh/t
Ball Mill Work Index	15.6 kWh/t

	Abrasion Index	0.403
	Feed to Rod Mills	F_{80} – 8 mm
	Milled product	P_{80} – 120 µm

Flotation (Residence Time/ Number of Stages):

	Rougher Flotation	7 min, 1 stage
	Scavenger Flotation	38 min, 1 stage
	Cleaner Flotation	15 min, 2 stages
Concentrate Grind:	P_{80} – 15 µm	
Pressure Oxidation:		
	Temperature	225 °C
	Pressure	3,380 kPag
	Retention Time	1.1 h
	Solids Recycle Ratio (residue recycle to fresh feed ratio)	200%
Solution Neutralisation:	Discharge	pH = 2
Copper SX:		
	PLS Grade	17 g/L
	Raffinate Grade	<1 g/L
	Extraction stages	3
	Strip stages	1
	Scrub stages	2
Copper EW:		
	Current density	300 A/m^2
	Current efficiency	90%
Magnesium Precipitation:		
	Final discharge	pH = 9
	% Mg precipitated	50%
Reagents/Services:		
	Oxygen Production	730 st/d
	Make-up Water	415 st/h
	Limestone (dry)	1,984 st/d

Tailings:

	Flotation	34,300 dry st/d
	Hydrometallurgical	2,430 dry st/d

Power: 69 MW (at steady state draw)

Product Quality:

Cathode Copper	LME Grade A, 99.995% Cu
AuPGM Concentrate	1,137 g/t AuPGM
Mixed Hydroxide Product	43.5% Ni, 1.8% Co

18.5 Process Description

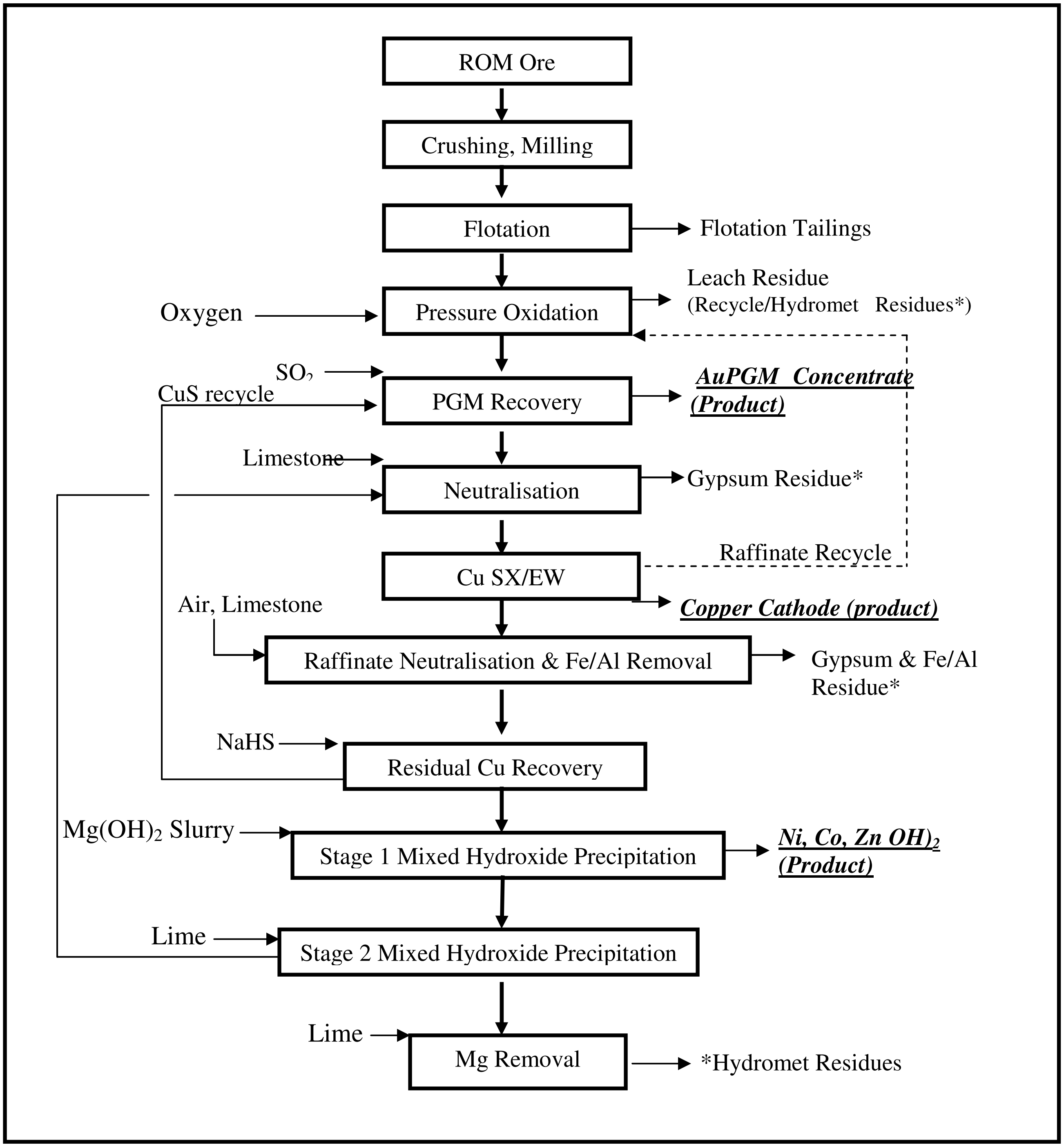
Plant production capacity was designed on the basis of an average daily plant feed rate of 32,000 short tons of ore with due consideration for plant availability and with varying ROM head grade to provide the following maximum metal production streams:

- 38,821 short tons per annum of high purity Copper cathode metal;
- 9,037 short tons per annum of Nickel as a mixed hydroxide;
- 400 short tons per annum of Cobalt as a mixed hydroxide; and
- 123,137 oz per annum of combined Palladium, Platinum and Gold as a precipitate.

The plant is designed for a minimum 20 year mine life and is based on a process engineering package developed by Bateman, from data generated by batch and pilot plant testwork.

The process plant design has been reviewed by PolyMet representatives plus external and Bateman process auditors and has subsequently been used as the basis for the capital and operating costs presented.

Figure 18-1 Simplified Process Flow Block Diagram



Overall metal recoveries adopted for design are: Cu 92.3%, Ni 70.3%, Co 40.7%, Au 67.0%, Pt 72.7% and Pd 75.2%. The flotation process recovers 87% of the sulphur from the ore feed with 13% of the sulphur reporting to tailings.

The metallurgical process is described in two main sections, ore beneficiation and hydrometallurgical processing, accompanied by schematic flow diagrams. Figure 18-2 shows Ore Beneficiation (crushing, grinding and flotation) and Figure 18-3 shows the Hydrometallurgical plant.

18.5.1 Ore Beneficiation

Existing equipment will be reinstated and used for both coarse and fine ore crushing (including gyratory and cone crushers), and in the ore milling circuit (including rod and ball mills). The flotation plant is a new circuit that will be housed in the existing Concentrator building.

Coarse ore crushing – In the coarse crushing area, ROM ore (with a top feed size of 48 inch /1200 mm) is reduced in two stages to 100% passing 3 inch (75 mm) prior to further size reduction in the fine crushing circuit at an average feed rate of 1,666 st/hr (1,512 mt/h). ROM ore is delivered by rail from the open pit and dumped sequentially from 100 short ton side tipping rail cars into a 110 short ton dump pocket above the 60 inch (1200 hp/900 kW) gyratory primary crusher. Following primary crushing, the ore gravitates to a second stage of crushing in three parallel 36 inch (540 hp/400 kW) gyratory crushers. The discharge from the secondary crushers is conveyed to a coarse ore bin above the fine crushers, which has a live capacity of approximately 2,200 tons which is equivalent to approximately 80 minutes of continuous feed.

Fine ore crushing – The coarse crushed product is further reduced in two stages to 8 mm suitable for feed to the milling circuit. Coarse crushed ore is delivered to a coarse ore storage bin that extends the length of the Fine Crushing Building. Since only three fine crushing lines will be reactivated only a portion of the total live storage capacity will be used. From the coarse ore bin material gravitates to three parallel fine crushing lines, each line consisting of a 7 ft (470hp/350kW) standard cone tertiary crusher discharging onto two double deck vibrating screens from where oversize discharges to two 7 ft (470 hp/350 kW) short head quaternary cone crushers. The screen undersize material passes directly to the conveyor below, which also collects quaternary crusher products. The final crushed product is conveyed to a fine ore bin in the Concentrator Building, which has a live mill feed storage capacity of approximately 17 hours.

Ore grinding – The milling circuits liberate sulphide minerals contained in the ore through a process of particle size reduction. The milling circuit comprises twelve parallel circuits each consisting of a 12 ft diameter and 15 ft long 800 hp (600 kW) rod mill feeding a 1,250 hp (930 kW) ball mill operating in closed circuit with a cyclone, with a circulating load of 250%. Each rod mill receives a proportion of finely crushed ore, approximately 128 st/h (116 mt/h) at P_{80} of 8 mm, and discharges product to a ball mill which produces milled product at P_{80} 120 of microns.

Sulphide flotation – The objective of the flotation circuit is to recover a bulk sulphide concentrate containing the base and precious metals whilst rejecting largely siliceous tailings. The concentrate produced is then fed to the POX in the hydrometallurgical plant.

Milled primary cyclone overflow along with flotation regrind cyclone overflow is split to two parallel trains of rougher/scavenger flotation. Rougher concentrate from both trains is combined and undergoes two stages of cleaner flotation to reduce mass and increase sulphide grade ahead of POX. Scavenger concentrate is combined with Cleaner One tailings and is fed to a regrind circuit, which includes one regrind mill operating in closed circuit with a regrind cyclone. The regrind cyclone overflow is directed back to the head of flotation.

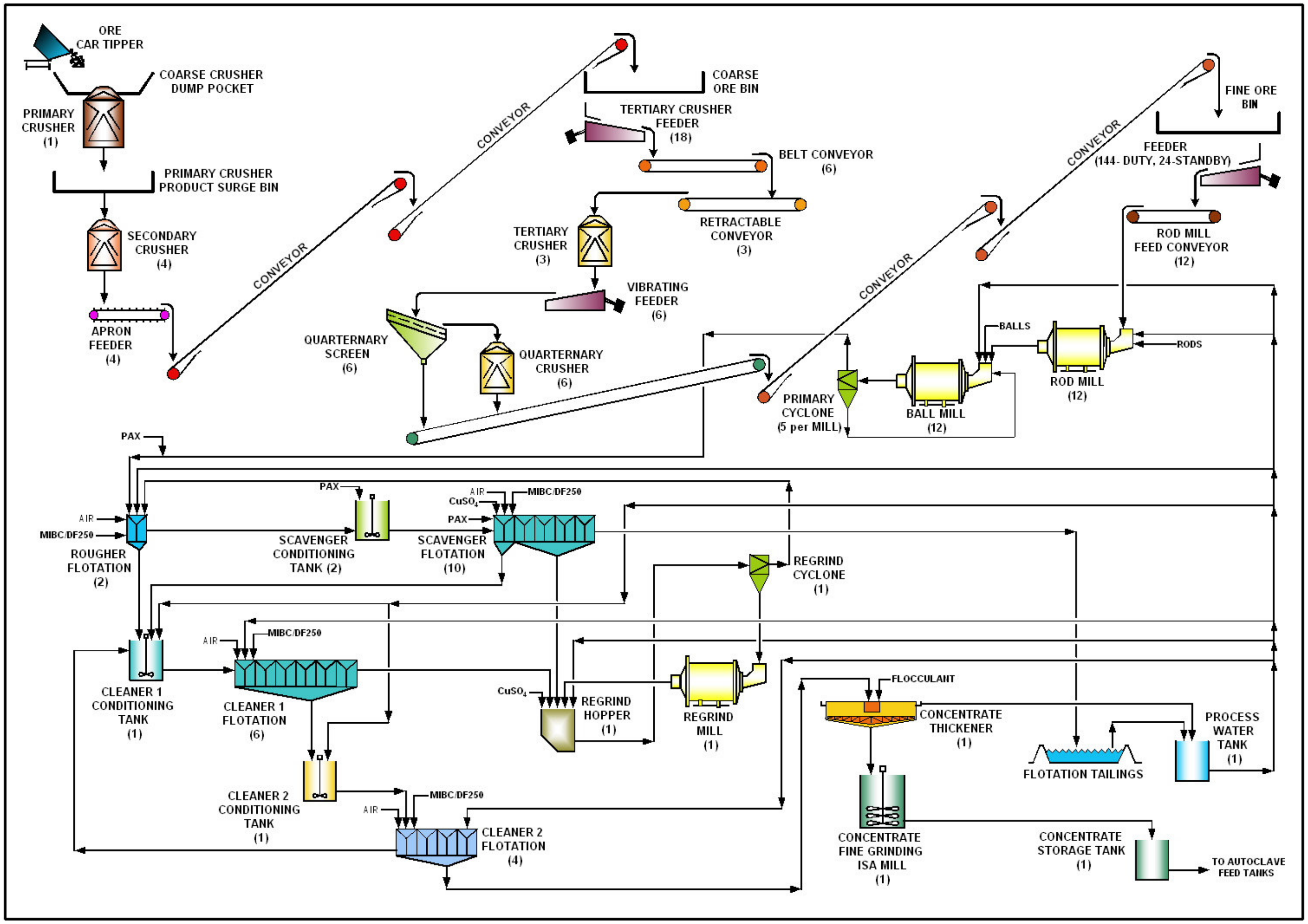
Scavenger tailings are pumped to the flotation tailings facility.

Flotation requires a number of reagents and make-up storage tanks and dosing pumps are provided within a nearby dedicated flotation reagents area.

Bulk Sulphide Concentrate Fine Grind - The final cleaner concentrate is mixed with flocculant and thickened, and the resulting underflow is pumped to a fine grinding ISA Mill to produce the POX feed at P_{80} of 15 µm.

Flotation tailings – Flotation tailings are pumped to the established Tailings Basin. Existing seepage collection systems will be augmented and upgraded to more efficiently capture seepage and return it to the basin.

Figure 18-2 Schematic Flowsheet - Beneficiation Plant



18.5.2 Hydrometallurgical Plant

The hydrometallurgical plant will be housed in three new buildings, the largest building is located adjacent to and south-east of the Concentrator and two separate buildings for the SX and EW operations are located further east on a lower elevation.

Pressure Oxidation (POX) Autoclave – The autoclave is designed to fully oxidise and extract the copper, nickel, cobalt, gold and PGMs contained in the flotation concentrate. The base metal sulphides are converted to soluble metal sulphates and acid using high temperature and pressure oxidative leaching conditions. The gold and PGMs form soluble chloride complexes under autoclave conditions in the presence of acidic liquor and chloride ions. Two autoclaves are required for the NorthMet circuit.

Each six compartment autoclave operates at 225° C (437°F), with a total pressure of 3,380 kPag (490 psi), and with oxygen supplied by an onsite 730 st/d (660 mt/d) cryogenic oxygen plant. Hot pressurised slurry is discharged from each autoclave in parallel single stage flash processes. Flash steam is recovered for heating process streams as required and excess vent gas is directed to the gas scrubbing system.

Slurry discharging from the flash vessels is further reduced to 60° C using dedicated spiral heat exchangers and pumped to the leach residue thickener. Approximately one-quarter of the 55% solids (w/w) thickener underflow reports directly to a vacuum belt filter where the final residue is washed, re-pulped in process water and pumped to the Hydrometallurgical Residue Facility. The remaining three-quarters of underflow slurry is recycled to the autoclave feed tanks. This corresponds to design for up to 300% recycle around the autoclave to maximise base and precious metal extractions.

Leach thickener overflow and filtrate are combined to feed acidic PLS rich in soluble base and AuPGM metals to AuPGM recovery.

Gas scrubbing – The gas scrubbing area provides treatment of the autoclave flash vessel and vent gases in two stages, firstly by dedicated venturi scrubbers and then by a common packed tower scrubber. This tower also receives AuPGM precipitation vent gases. A portion of the partially scrubbed gas is utilised for heat recovery in the mill process water heaters.

AuPGM precipitation – The AuPGM Precipitation area allows the precipitation of gold and PGM from the PLS as a saleable product, while maintaining the base metals in solution. The saleable concentrate contains AuPGM in a background of copper sulphide. This is suited to treatment in an offsite PGM processing facility.

Solution neutralisation – The aim of the solution neutralisation area is to remove residual acid from the copper solvent extraction feed solution using limestone to precipitate gypsum. This reduces the process liquor stream acidity to a level appropriate for copper solvent extraction while avoiding any precipitation of iron or copper. The gypsum produced is a potentially saleable product.

Copper solvent extraction – The copper solvent extraction process (SX) removes copper from the neutralised solution using three stages of extraction to produce a Cu rich organic. This organic is then scrubbed prior to two stages of stripping with spent electrolyte to produce a copper rich aqueous solution (electrolyte) for copper electrowinning. Raffinate from SX is then pumped to the combined raffinate neutralisation/iron and aluminium removal circuit.

Copper electrowinning – Copper metal is electro-deposited from filtered electrolyte onto stainless steel cathode blanks over a seven day cycle. Cathodes are harvested on a daily basis using an overhead crane, a cathode washing system and an automated stripping machine. Stripped metal is packaged in 3.5 tonne pallets and shipped for sale. Spent electrolyte is returned to the solvent extraction strip circuit.

Raffinate neutralisation/Iron and aluminium removal – After the SX process has removed the copper, the raffinate is subjected to neutralisation to further reduce the acidity produced during the copper extraction process. During the neutralisation process the pH is increased using limestone to a level such that iron and aluminium precipitate.

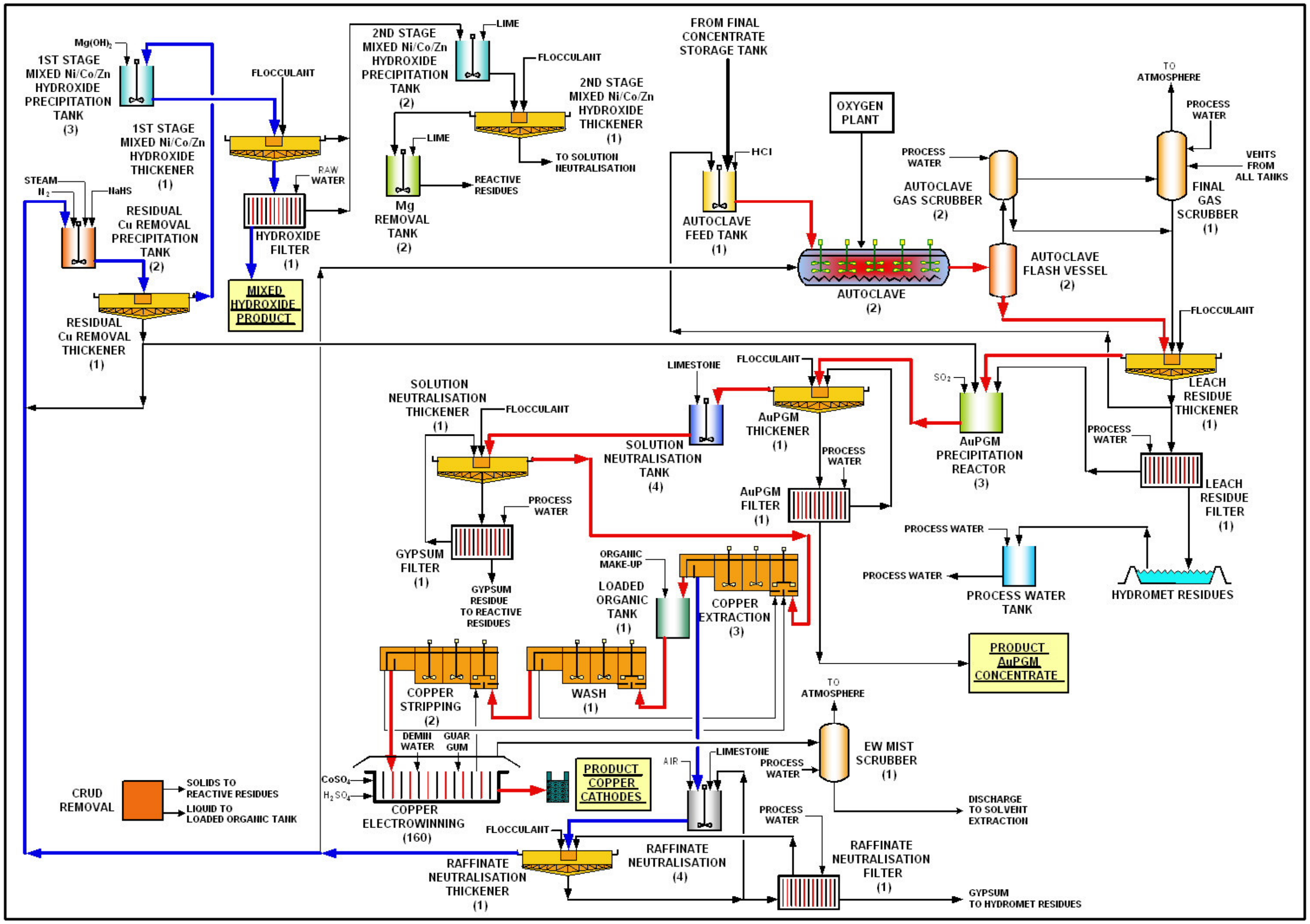
Plant gas scrubbing – This circuit is installed to scrub the hydrometallurgical plant vent gases allowing safe discharge to the atmosphere. All tanks within the main hydrometallurgical building will be vented to a packed tower gas scrubber. A common vent fan will be used to draw gases to the scrubber and a caustic addition system will be available to treat H_2S that may escape from the residual copper removal circuit.

Residual copper recovery – The objective of the residual copper recovery stage is to reclaim soluble copper from the Fe/Al removal circuit discharge liquor. This is achieved by precipitating the copper as copper sulphide. Sodium hydrosulphide solution is used to precipitate the copper sulphide along with sulphides of minor amounts of other base metals.

Mixed hydroxide precipitation – The mixed hydroxide precipitation circuit allows the recovery of a nickel and cobalt precipitate as a saleable product using a magnesium hydroxide slurry. This is achieved by using a two-stage precipitation process that utilises an increase in solution pH to promote the precipitation of metals as hydroxides. The first stage produces a high purity nickel and cobalt precipitate with minimal magnesium co-precipitation. The second stage recovers all of the nickel and cobalt in solution, but with significant magnesium co-precipitation. This precipitate is returned to the neutralisation stage of the process to re-leach the precipitated nickel and cobalt.

Magnesium removal – In magnesium removal, approximately one half of the magnesium in solution is precipitated from the process stream exiting the nickel and cobalt recovery circuit. This is performed to prevent magnesium build-up in the exiting process liquor, which is recycled throughout the hydrometallurgical plant as process water. This also minimises the recycle of other metals present. The magnesium removal precipitate is directed to the Hydrometallurgical Residue Facility.

Figure 18-3 Schematic Flowsheet - Hydrometallurgical Plant



18.6 REAGENT SUMMARY

The principal plant reagents and approximate annual consumption rates are shown in Table 18.7. Reagents will be delivered to site in bulk bags for smaller quantities or by rail or road tanker/truck for bulk reagents. KOA have identified suppliers for bulk reagents (including limestone, lime, magnesia, hydrochloric acid, sulphuric acid, caustic and sodium hydrosulphide), and determined the transport, delivery schedule and storage requirements for each.

Flotation reagents include collector, frother, activator and concentrate flocculant, and will be delivered to the Concentrator building for preparation, storage and distribution. Storage of flammable flotation reagents will be in a new warehouse building and bulk storage tanks located outside the Concentrator building. Reagents used in the principal new hydrometallurgical building (lime, caustic, sodium hydrogen sulphide, hydrochloric acid, magnesium hydroxide slurry, coagulant and flocculants) will generally be received, stored and distributed from the General Shop. Liquid sulphur dioxide will be stored in a pressurised storage bullet adjacent to the General Shop, and copper solvent extraction and electrowinning reagents will be stored in their respective areas.

The hydrometallurgical process has been designed to remove acid from all streams that discharge to the hydrometallurgical residue facility. Acid neutralisation is mainly by using limestone, at 250,000 st/a (226,400 mt/a), and lime, at 12,700 st/a (11,500 mt/a).

In terms of volume, limestone is the most significant process consumable and two suitable sources have been identified. Both are situated on Lake Michigan and have docking and lake freighter loading facilities and are the source of large volumes of limestone used by the taconite producers as flux in their pelletizing process. Taconite ore carriers returning empty to the ports serving the Iron Range would be used to back-haul limestone to two possible locations, namely the Port of Duluth (Hallet Dock) and Taconite Harbor, north of Duluth on the north western shore of Lake Superior.

- Taconite Harbor is jointly owned by Cleveland-Cliffs and Minnesota Power (MP) and is connected directly to the Erie Plant by the Cliffs private railroad. MP uses the facility to import steam coal for the 225MW power station situated dock-side. Cliffs uses the facility less frequently to export taconite. Use of Taconite Harbor would require agreement of both Cliffs and MP and modification of the ship unloading facilities and stockpiling arrangements along with acquisition of rolling stock and upgrading of the existing railroad.

- The other possible destination for limestone is the Port of Duluth which is a major commercial shipping centre and has off-loading, stockpiling capacity and train loading facilities. Duluth is served by a number of common carrier railroads including Canadian National (CN) which owns the track serving the Erie Plant. Use of the Duluth facilities would be on a fee paying basis and subject only to contract.

Both options were evaluated and the Port of Duluth selected on the basis of certainty of outcome rather than for purely economic reasons although, in the longer term, Taconite Harbor may offer economic and other benefits. Cost estimates for limestone are based on written quotes for supply FOB at producers quarry with shipping, unloading, stockpiling, train loading and rail haulage costs based on a synthesis of current rates applicable to several taconite producers.

At full production, annual consumption of limestone will total 260,000 tons and the delivery schedule and site stockpile requirements (140,000 ton capacity) have been determined taking into account the Great Lakes shipping season, which runs from early April to mid-November. (Shipping stops during the winter because of freezing of the lock system between the lakes that make up the shipping route.)

During plant production ramp-up, limestone will be trucked from Duluth. As production rate builds up to full rate the transition will be made from truck to rail haulage to benefit from the economies of scale offered by rail haulage.

Initially, limestone will be truck hauled on an as required, just-in-time basis and dumped directly into a hopper constructed over the existing 4A Conveyor between the Fine Crushing Plant and the fine ore bins on the south side of the Concentrator. (Only the fine ore bins on the north side will be reactivated for NorthMet ore.) The 4A conveyor will deliver limestone to existing transfer conveyor 5S and through a tripper to the fine ore bins feeding the 3S rod and ball mills. Ball mill product will be sized by hydrocyclone with the minus 30 micron slurry pumped to the hydrometallurgical plant while oversize material will recycle to the grinding circuit for further milling.

As plant ramp-up progresses the rail unloading and stacking system previously used for coal will be upgraded and reactivated for limestone. This involves installing a larger discharge hopper to accommodate the bottom discharge, 100 ton capacity rail cars typically used for limestone, replacing feeders and upgrading conveyors to increase the dumping rate from 225t/hr to 2500t/hr. (This rate is sufficient to comfortably unload an 80-car train in a single shift, thereby minimising the risk of incurring demurrage.) An enclosure over the dumping point and dust collection arrangements are also included for environmental reasons. A 210 ft radial stacking conveyor will be installed to construct a stockpile of sufficient capacity to minimise re-handling and dozer push-out. Limestone will be reclaimed from the stockpile by front end loader through the refurbished coal reclaim system and onto a belt conveyor which will transport it via the former boiler house (adjacent to and just east of the rail unloading facility) and from there via the existing pipe gallery to the area of the existing re-grind mills on west side of the Concentrator. A single regrind mill is calculated to have sufficient capacity to handle limestone at the required full production rate but an additional re-grind mill can be quickly reactivated if the single mill proves inadequate. Regrind mill product will be hydrocyclone sized before the slurry is pumped to the hydrometallurgical plant with oversize recycled to the regrind mill.

Oxygen for the POX process will be supplied from a 700 tonne/hr cryogenic oxygen plant built on the plant site but operated and maintained by a third party. This type of "over-the-fence" supply arrangement is relatively common and several prospective suppliers have already expressed interest.

Table 18-7 Principal Reagents and Consumption Rates

Reagent	Annual Consumption st/a
Flotation Circuit	
Collector - Potassium-Amyl Xanthate (PAX)	600
Frother - MIBC & DF250	360
Activator - Copper Sulphate	(equiv. 650 t/a from EW stream)
Flocculant - Magnafloc 10	16
Hydrometallurgical Plant	
Limestone	250,000
Lime	12,700
Hydrochloric Acid (32%)	8,400
Magnesium Hydroxide Slurry (61%w/w)	17,500
Caustic Soda (50%)	66
Sulphuric Acid (93%)	2,800
Liquid Sulphur Dioxide	3,800
Sodium Hydrogen Sulphide (45%)	1,800
Leach Residue Flocculant - Magnafloc 351	224
Plant Flocculant - Magnafloc 342	1.0
SX - Diluent	130
SX - Cu Extractant	24
EW - Cobalt sulphate	7
EW - Guar Gum	9

In addition to reagents, various ancillary systems will also be provided, including:

- Packaged demineralisation plant for boiler feed water;
- Fire water ring-main system;
- Plant air from three compressors with a drying/filtration system for instrument quality air (e.g. for driving pneumatically actuated valves); and
- Boilers for providing steam on autoclave start-up and for process preheating duties.

18.7 Flotation Tailings and Hydrometallurgical Residue Disposal Facility

The former LTVSMC taconite tailings basin covers an area of about 3,000 acres and consists of two separate sections. The western half has been largely re-vegetated as part of Cliff Erie's post-closure site restoration plan, prior to PolyMet acquiring the facility. The tailings basin was being actively used for tailings deposition right up to closure and it is estimated that remaining capacity will be sufficient for approximately 28 years at PolyMet's planned production rate. Therefore, reactivation of the tailings basin is both technically feasible and highly cost effective.

The western half of the basin is known as Cell 2W while the eastern part of the basin is sub-divided into Cells 1E and 2E. PolyMet plans to use Cell 2W for disposal of hydrometallurgical plant residues and Cells 1E and 2E for flotation tailings.

18.7.1 Flotation Tailings

Flotation tailings at 60% (w/w) solids will be pumped to Cells 1E and 2E for deposition under conditions similar to those used by LTVSMC. A movable, single point discharge pipeline will initially discharge into Cell 1E for a minimum of three years and then into Cell 2E. Thereafter and through the life of the mine, discharge will alternate between the two cells which have an estimated total flotation tailings storage capacity equivalent to about 28 years at projected production rates. Clear, settled, transport water will be reclaimed and pumped back to the beneficiation plant for re-use.

Flotation Tailings - Evaluation of Tailings Geochemical Reactivity

In the context of Minnesota Metallic Mining Rules, testwork on NorthMet tailings produced during pilot testing has shown them to have low geochemical reactivity, or to be non-reactive. The principal source of leachable metals in tailings is oxidation of sulphide minerals and dissolution of the resulting weathering products. However, extensive testwork to date indicates that tailings will not generate acid and metals leaching from flotation tailings should not be problematic because:

- The flotation circuit has been designed to maximise sulphide recovery and testwork has shown that sulphur concentration in tailings will be low. When copper sulphate was used during testwork as an activator to improve bulk sulphide flotation recovery, the highest sulphur concentration in tailings was 0.15%. Waste characterization has shown that this is well below the concentration level at which acid generation will be balanced by alkalinity generation in the tailings. Therefore, tailings will not produce acid.
- Because of the low to negligible concentration of metal sulphides, leaching of copper (and other metals) from the tailings is expected to occur slowly with resultant copper concentrations expected to be below the appropriate surface water discharge limits. Oxidation and metals leaching rates are governed by oxygen availability which in the case of tailings will be diffusion controlled and limited to near surface material. In addition, pyrrhotite in the tailings produces iron precipitates which tend to adsorb metal ions.
- Any seepage from the tailings basin will interact with the underlying taconite tailings which contain abundant carbonates as well as iron oxides for acid neutralisation. Taconite tailings are known to sequester mercury and may also provide base metal sorption surfaces.

Flotation Tailings Deposition

Cells 1E and 2E were fully operational at LTVSMC closure and it is expected that reactivation will simply involve a continuation of former operating practices which relied upon upstream construction of the tailings dikes from peripheral, single point discharge. The discharge point will be moved regularly around the perimeter with coarser material accumulating near the discharge point. Some re-handling of the coarse fraction will be necessary using a rubber-tired dozer to form the perimeter dam along the outer edge of the tailings basin cells.

The anticipated dam crest elevations for Cells 1E and 2E at different stages of the operation are shown in Table 18-8.

Table 18-8 Flotation Tailings Basin - Minimum Dam Crest Elevations

Year of Operation	Baseline Elevations (ft)	
	Cell 2E	Cell 1E
Prior to Operation	1557	1637
5	1582	1707
15	1607	1727
20	1682	1767

Flotation Tailings Basin Surface Water Handling
Similar pumping and water-handling systems will be used for Cells 1E and 2E. Initially tailings deposition will occur in Cell 1E and clear water will be recycled back to the beneficiation plant using the refurbished pontoon-mounted pump station. Deposition will alternate between cells on approximately a three year cycle; however, in the event that there is not sufficient clarification of the water in the deposition cell, the other cell can be used as a clear water source.

Flotation Tailings Basin Operation
In addition to the normal management of pipes, pumps and perimeter dam construction, an annual design and stability assessment will be carried out by a specialist consultant.

Minnesota dam safety rules provide for a minimum freeboard above the water line in order to prevent overtopping during storm conditions. At least seven feet of freeboard for storm event run-off containment and wave run-up management are planned. Minnesota Rules also stipulate that dust lift-off must be minimised. This will be achieved in part by managing water levels and, where necessary, by surface treatment or by temporary seeding and re-vegetation to bind and stabilise the tailings.

Flotation Tailings Dam Slope Stability Simulations
The design process for the proposed upstream construction method included preliminary slope stability simulations and factor of safety determinations at selected locations through the projected life of the tailings basin. Results of these simulations indicate one area where from year 10 onwards factors of safety will be marginally below the recommended minimum values for both Peak Undrained Strength (without pore pressure reduction) and Post Liquefied Strength. At this stage the effects of pore pressure reduction have not been analysed and more detailed engineering and design work together with accumulation of additional geotechnical data will determine whether remedial action such as installation of additional drains and flattening of slope angles will be required.
From cone penetration testing (CPT) investigation and analysis to date, it can be concluded that the stability of the LTVSMC tailings dam may be affected by excess pore pressure. Pore pressure, however, can be dissipated by installation of drains and final design will include additional testing and staged-construction analysis to design remedial measures. Moreover, stability analyses have shown that even if excess pore pressures generate a liquefied state within the tailings dam due to rapid loading, the LTVSMC tailings dam still has a factor of safety greater than 1.0, and so a large-scale failure is not likely.

18.7.2 Hydrometallurgical Residue Disposal

The hydrometallurgical residues shown below will be produced as separate process residue streams which will be combined and pumped to the residue containment area in Cell 2W:

- Leach residue;
- Solution neutralisation residue (gypsum);
- Raffinate neutralisation residue (iron, aluminium and gypsum residue);
- Magnesium residue; and
- Crud solids from solvent extraction crud removal.

The projected rate of total residue generation is 882,000 short tons per year of which 290,000 short tons are gypsum from acid neutralisation which will be separated as a by-product for beneficial use offsite. Therefore, the net annual discharge to the reactive-residue disposal facility will be 592,000 short tons per year. Residue solids will consist primarily of non-cohesive silt-size particles contained in process water characterised by high concentrations of sodium, calcium, magnesium, and sulphate salts (several 1,000 milligrams per litre each).

Reactive-residue will be disposed of in specially designed and engineered, lined cells constructed on the western part of the existing LTVSMC taconite flotation tailings basin (Cell 2W). The first of these cells will be extended progressively in three vertical lifts thereby providing sufficient capacity for five years of storage at current production rates. Each residue cell will act as a sedimentation basin allowing solids to settle over time and supernatant to be pumped back to the process plant for re-use. The hydrometallurgical process as a whole is a net water consumer and requires make-up water provided by recycling reclaimed residue transport water balanced, as necessary, by fresh water make-up from Colby Lake.

As one cell approaches capacity a second cell will be constructed nearby. Supernatant and accumulated precipitation will be pumped out of the first cell as solids settlement takes place. Settlement will create freeboard at the top of the cell into which flotation tailings will be pumped to create a cover. This cover will consist of a layer of flotation tailings on top of settled residue to provide a base for an impermeable membrane, which will be covered with more flotation tailings. The final layer of flotation tailings will ultimately provide the basis for re-vegetation.

Hydrometallurgical residue disposal facility design must satisfy the requirements of Minnesota Rules for Reclamation of Non-ferrous Metallic Mineral Mineland, the object of which is to prevent the release of substances that result in adverse impacts on natural resources. Residue cells will be lined to reduce seepage and covered to reject precipitation; however, any seepage that does escape through the liner will pass through approximately 200 vertical feet of taconite tailings before reaching existing seeps at the basin perimeter, where an upgraded seepage-water collection system will return seepage to the plant for re-use.

Factors considered during preliminary design of the liner and cover systems include:

- Compatibility with residue chemical characteristics;
- Hydraulic conductivity and containment ability;

- Construction season limitations;
- Tolerance of differential settlement;
- Liner freeze-thaw cycle tolerance;
- Ease of installation;
- Wet transport and disposal versus dry transport and disposal;
- Ability to accommodate vertical cell development;
- Compatibility with dissolved metals, salts, and petrochemicals;
- Procurement and installation cost;
- Availability of material; and
- Ability to permit.

Several broad categories of liner types were considered as part of preliminary design including natural clay, geosynthetic clay, and a wide range of synthetic geomembranes. The various liner types can be installed in configurations as needed to address specific regulatory and performance requirements and a variety of liner configurations were considered. Single liners that were considered included a single layer of geomembrane, geosynthetic clay, compacted clay, or other barrier layer component. Composite liners were considered consisting of a variety of upper and lower barrier layer components. Double liners were also considered but are typically reserved by regulatory agencies for installations that contain hazardous wastes. Given that the reactive residue is not a hazardous waste, a double liner system was not carried forward in the design effort.

The selected liner option consists of a polyethylene or poly-vinyl-chloride geomembrane placed above compacted LTVSMC tailings. This type of geomembrane liner system is compatible with residues containing dissolved metals and salts. Furthermore, there is nothing in the characteristics of the leachate that would prove detrimental to the long-term performance of the selected geomembrane liner.

Each residue cell will be sized to accommodate approximately 4,000,000 cubic yards of residue and residue transport water. There is also provision for additional freeboard for storm event protection, and to accommodate a protective cover over the cell liner on closure. Cells will typically be a minimum of 2,000 feet long by 850 feet wide at the crest of the perimeter dike which will provide five years of storage capacity. Cell depths will be 80 feet and embankment slope angles will be a maximum of 4-horizontal to 1-vertical for stability.

At closure of each cell, water used to transport the residue, water accumulated in that cell from precipitation, and leachate collected from the base-of-cell leachate collection system will be returned to the plant for re-use. A variety of closure cover systems were identified and compared on a qualitative and quantitative basis. The selected cover system consists of a 24-inch thick base layer of PolyMet flotation tailings above the reactive residue, overlain by a polyethylene or poly-vinyl-chloride geomembrane barrier layer, an additional 48 inches of PolyMet flotation tailings, and surface vegetation. The added weight above the residue from this cover system will induce pore water pressure build-up in the residue which can reduce the residue's strength and ability to support construction equipment. To allow pore water pressure to dissipate as construction proceeds, the cover layers will be constructed over a two to three-year timeframe.

Surface water run-off from the cover system will be clean and, after undergoing sediment control, will be discharged from the facility. Discharge of surface water run-off by gravity flow through engineered piping and drainage swale systems will minimise surface erosion during severe precipitation events.

The State of Minnesota requires submission, review, and state approval of a quality control/quality assurance (QA/QC) program for liner and cover systems prior to construction. In addition, the state requires submission of a construction report that summarises the details of construction and presents the results of the quality assurance testing. This documentation will be prepared as part of subsequent permitting and construction activities. Quality assurance testing will be performed during construction by a qualified independent testing laboratory.

18.8 Plant Water Balance and Water Treatment

The water balance for the NorthMet Process Plant is a quantitative description of all input and output flows of water required for the effective operation of the facility. The water used in the facility is physically separated into two distinct plant areas:

- Concentrator Water Balance, and
- Hydrometallurgical Plant Water Balance.

Testwork, modelling and simulations have shown the Concentrator will require a continuous, net make-up of fresh water at a rate of 276 short tons of water per hour. There is no requirement to purge water from the Concentrator. The major streams comprising the water balance are:

- Water entrained in the ore feed,
- Fresh water make-up from Colby Lake (this was the make-up water source used by LTVSMC and is situated approximately 4 miles to the south west),
- and treated mine-site process water,
- Precipitation with seasonal snow and ice melt,
- Tailings transport water and settled, clear water reclaimed from the tailings basin, and
- Water entrained in concentrates pumped to the Hydrometallurgical Plant.

Largely because of water losses as steam vented to atmosphere on the pressure let-down side of the autoclave, the Hydrometallurgical Plant will require continuous fresh water make-up and there will be no requirement to periodically purge water from the system. Testwork, modelling and simulations have shown the need for a net make-up of fresh water at a rate of 139 short tons of water per hour. The major streams considered when simulating the plant water balance were:

- Water entrained in incoming concentrate feed,
- Fresh water make-up, also from Colby Lake,
- Precipitation with seasonal freezing and snow and ice melt,
- Water entrained in residues and return water from residue cells, and
- Various plant vents but principally as steam from the autoclave pressure, flash let-down vessels.

As much for economic as for environmental reasons, strong emphasis has been placed on maximising water recycling at all stages of the process. From the environmental point of view it was important to avoid build-up of impurities in process streams that might require purging to the environment. To ensure this would be the case, water and mass balance simulations were carried out with particular reference to impurities such as chloride, magnesium and sodium. None was found that would require purging and discharge to the environment, a conclusion which was supported by the absence of impurity build-up during the pilot plant testwork.

19. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

19.1 Mineral Resource Estimate

During the course of project development, Dr. Phillip Hellman of Hellman & Schofield Pty Ltd (H&S) prepared a number of resource estimates with the most recent estimate completed in July 2006. The July 2006 estimate shown in Table 19-1 below is based on a conventional resource block model and takes into consideration all drilling carried out to date. Based on considerations of grade variability and size of the selective mining unit, a block size 100 feet on strike, 100 feet perpendicular to strike and 20 feet vertically was selected. Ordinary Kriging was used for grade estimation with mineralisation categorised as Measured, Indicated or Inferred according to CIM guidelines. Each mineralisation category was specified by a minimum number of data points found within a search ellipse whose dimensions were determined from variograms. Measured category has the most stringent search criteria (shortest search distance and most number of encountered data points) with the criteria for Indicated and Inferred material becoming progressively more relaxed. The Resource Estimate reported in this section forms the basis for the Reserve Estimate.

Mining engineering consultant, Australian Mine Design & Development Pty Ltd (AMDAD) has worked closely with H&S throughout this DFS. To assist H&S develop a resource estimate that complies with the requirement that it reports only material with a reasonable expectation of being mined, AMDAD developed a preliminary Whittle pit optimisation based on an earlier resource model by H&S. This optimisation indicated an economic pit floor at the 560 foot elevation (approximately 1,040 feet below surface). Therefore, the Resource Estimate reported in Table 19-1 below, contains only material above the 500 foot elevation. This resource is also reported for a Net Metal Value (NMV) of greater than US$7.42/st. Both the pit optimisation and the NMV were based on metals price assumptions of Cu = US$1.25 /lb, Ni = US$5.60 /lb, Co = US$15.25 /lb, Pt = US$ 800 /oz, Pt = US$210 /oz and Au = US$400/oz. Metal recovery assumptions were based on metallurgical testwork. The US$7.42 NMV was selected on the basis of a preliminary estimate of mining and processing costs that would provide a reasonable envelope within which a coherent mine plan could be developed.

Table 19-1 July 2006 Mineral Resource Estimates, above 500ft elevation and NMV cut-off of US$7.42

Category	Metric Tonnes	Short Tons	Cu %	Ni %	Co ppm	Pd ppb	Pt ppb	Au ppb	PGE ppb	S %	SG	NMV ($US)
Measured	121.3	133.7	0.298	0.087	74	269	67	35	373.4	0.786	2.98	15.11
Indicated	261.6	288.4	0.266	0.078	72	231	66	33	330.7	0.711	2.96	13.54
Meas+Ind	382.9	422.1	0.276	0.081	72	243	66	34	344.2	0.735	2.96	14.04
Inferred	109.4	120.6	0.247	0.074	70	217	65	33	316.4	0.707	2.94	12.72

19.1.1 Resource Upside Potential

In addition to the tonnages shown in Table 19-1, there is a substantial amount of potential mineralisation contained within the interpreted mineralised units above the 500 feet elevation which does not yet constitute a Mineral Resource but which may be upgraded to Resource status with further drilling. Using the DFS resource model and an extended search radius for grade interpolation, H&S estimates this potential material to range between 75 and 150 million tons at grades which may be comparable to the reported resource. Neither this non-Resource, potential material nor the expanded pit resource mineralisation shown in Table 19-3 take into account material below the 500 feet elevation where previous drilling has provided evidence of continuity of mineralisation to a depth of 2,500 feet. To further quantify the considerable upside potential PolyMet believes exists, it is planned to carry out additional, very focussed drilling, with the object of upgrading potential material and Inferred category material to a point where it can be included in a mine plan.

To provide an indication of the possible ultimate scale of the resource, AMDAD prepared two further pit optimisations using the metal price assumptions listed in Table 19-2 below. Table 19-3 shows the range of Measured, Indicated and Inferred category material contained within each of these optimised pit shells. Table 19-3 shows the case for each optimisation that is driven by value maximisation. At this stage no mine plan has been developed for either case shown in Table 19-3 though both cases indicate the expected significant increase in resource of all categories as metal prices increase. The NMV of Measured plus Indicated material within the expanded, optimised pit shells is shown for simplicity as a surrogate of grade.

Table 19-2 Metal Price Assumptions for Extended Pit Optimisation

Scenario	Cu	Ni	Co	Pd	Pt	Au
	US$/lb	US$/lb	US$/lb	US$/oz	US$/oz	US$/oz
1	1.50	6.50	15.25	225	900	450
2	2.25	7.80	16.34	274	1040	540
Resource Case	1.25	5.60	15.25	210	800	400

Note: Scenario 1 above approximates the three-year trailing average prices to July 31, 2006 and Scenario 2 reflects the weighted average of the three-year trailing average (60%) and the two-year forward price averages (40%) in July 2006.

Table 19-3 Extended Pit Optimisation Results

Price Scenario	Mine Life (years)	Strip Ratio ore:waste	Measured (Mtons)	Indicated (Mtons)	Inferred (Mtons)	Meas + Indicated (Mtons)	M+I NMV (US$/ton)
1	38.0	1.52	131.1	241.7	71.4	372.8	18.68
2	56.5	1.38	155.6	354.2	151.5	509.8	21.29

19.1.2 Resource Model Development

Figure 19-1 shows the collar positions of holes drilled during the 2005-2006 (most recent) drilling campaign in relation to previous drilling into the Duluth Complex in the vicinity of the NorthMet deposit.

The geologic model which forms the basis of the resource model created by H&S was developed by PolyMet's geologists using all previously available geologic and drill hole data, plus detailed core logging and assaying data from the 2005-2006 drilling program. A surface defining the base of till/overburden was also modelled using drill intercept data.

Two principal mineralised domains have been identified by drilling and their distribution is illustrated in Figure 19-2 and Figure 19-3. Figure 19-2 illustrates in cross section the location of these mineralised domains as represented in drillhole traces with the lower and most significant domain shown in yellow and the upper domain shown in red. Figure 19-3 shows in plan view the extent of these domains as found in drill holes.

Figure 19-4 illustrates a typical cross-section through the resource model showing the modeled tops of Duluth Complex Units 1-7, footwall Virginia Formation (20), and Biwabik Iron Formation (30). In the geologic model Units 2 & 3 are modelled as a single package, as are Units 4 & 5. The scale is shown by easting and northing though the section is non-orthogonal. In the diagram, high confidence Measured and Indicated blocks (categories 1 & 2) are shown with red cross-hatching, blocks with Inferred estimates (category 3) are marked with green vertical stripes and potential mineralization is shown by open boxes. Drill hole traces are marked and a preliminary pit shell is shown by a heavy dashed line which, on this section, bottoms out at about 1,000ft elevation. To maintain consistency with previous studies, only blocks that exceed a NMV value of US$7.42 have been shown. NMV was selected (rather than a metal equivalent grade) because it provides a means of representing the value of all contributing metals and approximates a lower cut-off of 0.2% Cu and 0.06% Ni. The NMV value for each block was calculated using the Base Case metal prices listed in Table 1-4. Metallurgical recovery assumptions were based on pilot plant testwork.

Variography was completed for Cu, Ni, Co, Pt, Pd, Au, and S. Grades for these were estimated by Ordinary Kriging. Densities were modelled using inverse distance weighting within appropriate geological units.

Figure 19-1 Drill hole location plan showing pre- and post-2005 drill hole collar positions

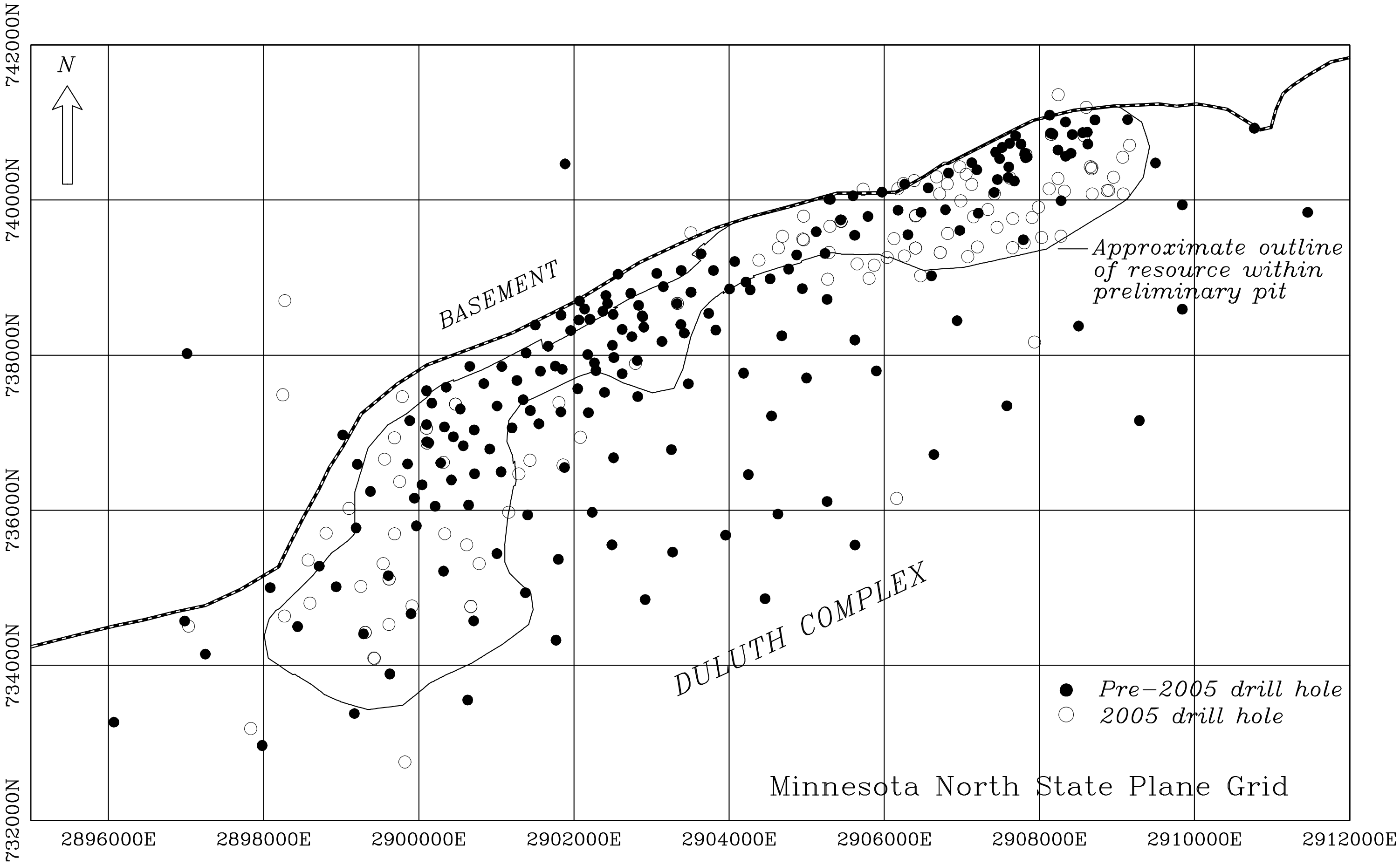


Figure 19-2 North-west to south-east cross-section showing mineralised domains

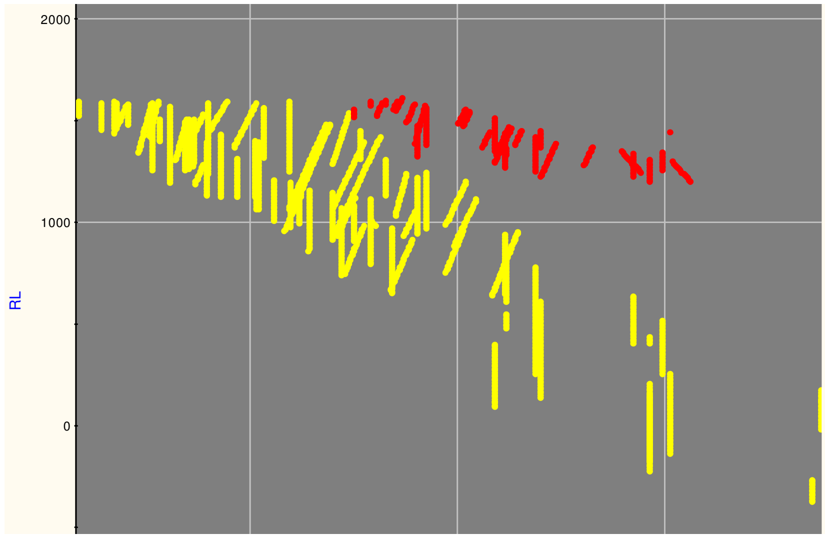


Figure 19-3 Plan distribution of mineralised domains across the deposit

(yellow = mineralisation hosted by Units 1 & 2; red = overlying mineralisation)

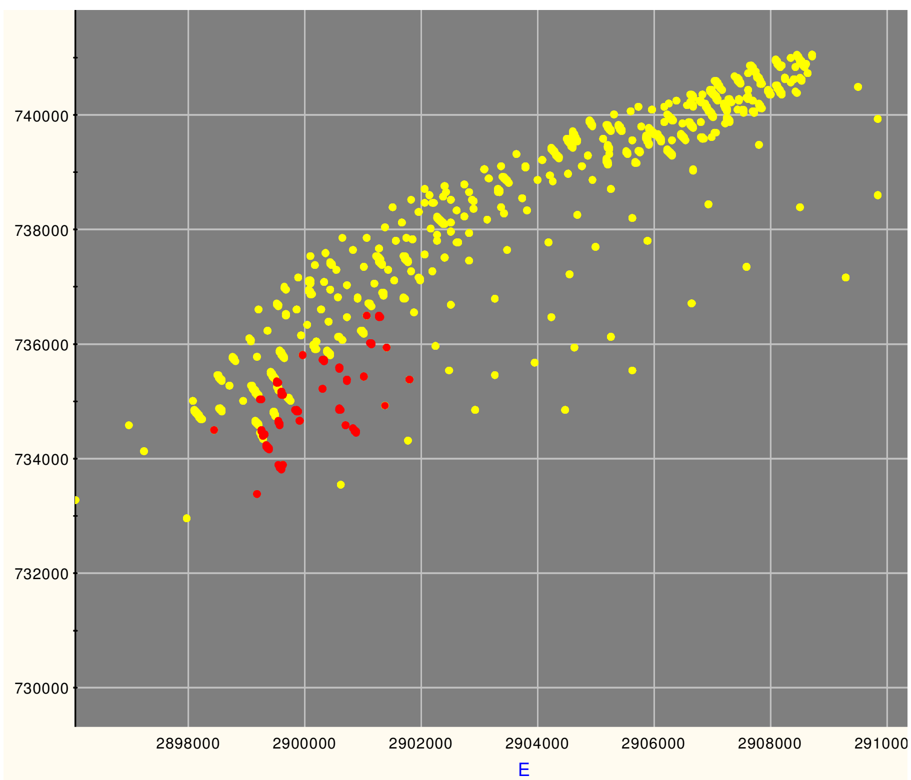
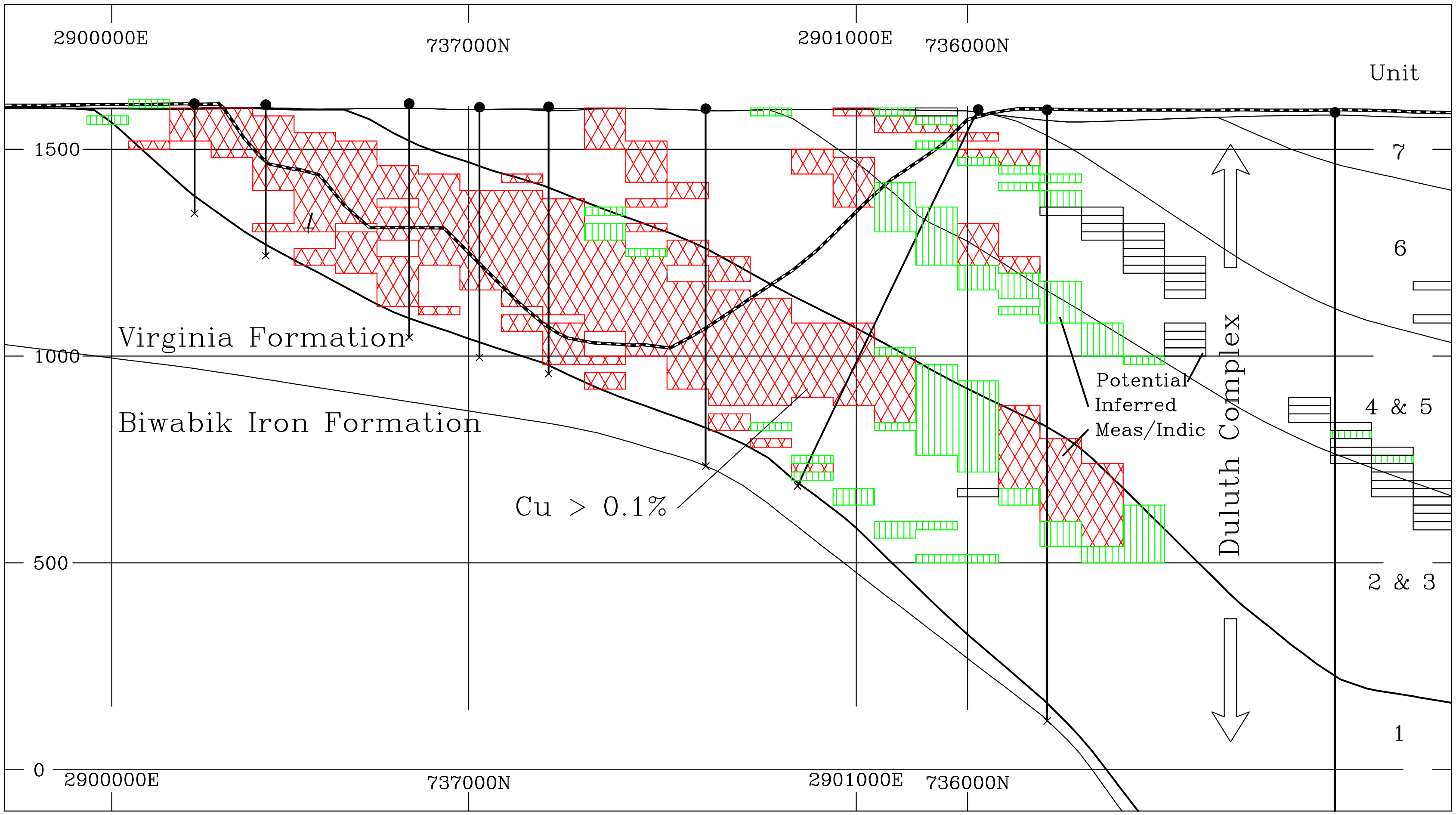


Figure 19-4 Typical cross-section with modelled lithological surfaces and Cu mineralisation superimposed on an optimized pit



19.2 Mineral Reserve Estimate

19.2.1 Reserve Categories

Mineral Reserves are part of the published Mineral Resource and only the Measured and Indicated categories from the Mineral Resource Estimate can be used in the Mineral Reserve Estimate. Proved Reserves are derived from Measured Resources and Probable Reserves are derived from Indicated Resources. No other non-geological factors were identified to alter this allocation.

19.2.2 Mineral Reserves

The Diluted Mineral Reserve Estimate for the NorthMet Deposit is shown in Table 19-4 below.

Table 19-4 Diluted Mineral Reserve Estimate

ORE	s. tons	Copper %	Nickel %	Cobalt ppm	Palladium ppb	Platinum ppb	Gold ppb
Proved	80,390,000	0.32	0.09	76	292	75	39
Probable	101,260,000	0.30	0.08	74	268	79	38
Total	181,650,000	0.31	0.09	75	279	77	39

WASTE	302,310,000
Waste : Ore Ratio	1.66

Note: This table shows the reported tons and grades rounded to significant digits appropriate for the confidence of the estimate. As a result total quantities and grades may not exactly match the sums and averages of the individual classes.

The following section describes the basis for and method of calculating the Mineral Reserve Estimate.

19.2.3 Basis of Reserve Estimate – Resource Model

NorthMet is a large, disseminated sulphide deposit in which most of the target zones for mining have true widths of over 150 feet though zones down to 20 feet thickness are included in the Mineral Reserves. Most of the mining area is covered by a thin layer of un-mineralised glacial till which varies in depth between 5 and 30 feet.

Earlier mining studies concluded that the deposit is not suitable for underground mining and subsequent pit optimisation studies have consistently shown good potential for open pit mining. Moreover, as a result of the drilling and additional sampling of previously drilled but unsampled core, the geological model has undergone structural re-interpretation which has enhanced open pit project economics.

Mine planning since early 2005 has been an iterative process concentrated on refining the pit optimisation inputs, designing pit stages, designing and scheduling the construction of waste stockpiles to meet environmental and waste rock management requirements, all of which had to be done in such a way as to achieve the value suggested by the optimisations. Production scheduling took into consideration equipment fleet size and performance requirements, minimum working width requirements for efficient equipment operation, the need to balance production from several work places while minimising stockpiling requirements and delivering the best cashflow profile possible.

The Reserve Estimate reported above has been diluted to reflect expected ore losses and dilution that inevitably accompany mining operations. Dilution was simulated by running an ore dilution routine on the block grade model prior to running the pit optimisation. This dilution routine, in effect, shifts the centroid of each block relative to the original model such that 25% of each block overlaps the block next to it, thereby 'smearing' the grade of one block into the other. In this way the mixing effect at block boundaries that results from movement of rock during blasting and subsequent excavation is simulated mathematically. While this effectively represents what happens during mining, the overall effect on ore tonnage and grade is insignificant because the ore zones are very wide relative to the size of the simulated block displacement.

19.2.4 Pit Optimization

Mine planning is an iterative process that starts with an optimised pit shell representing the highest value envelope that can be mined under certain assumptions of cost, pit slope geometry, metal recovery and metal prices. Having generated an initial optimum pit, AMDAD developed a mine design within that pit shell from which a preliminary production schedule was prepared. This pit design and preliminary production schedule were used as the basis for prospective mining contractors to prepare DFS level of accuracy quotes for mining the NorthMet deposit which, in turn, were used to develop the DFS mining cost estimate.

DFS pit optimisations were performed using Whittle 4X multi-element optimisation software to define the highest value open pit shell and to guide design of pit stages to deliver a value as close as possible to that theoretical maximum, subject to practical mining constraints. Inputs to the initial pit optimisation included:

Pit Slopes

The overall slope angle of 51.4° is taken from recommendations made by Golder Associates Ltd in their March 2006 report.

Process Recoveries

Table 19-5 shows the Process Recovery assumptions used for optimisation. These values are based on a detailed assessment of all metallurgical testwork results carried out on NorthMet material to date.

Table 19-5 Process Recoveries used for Whittle Pit Optimisation

Metal	Flotation (%)	POX (%)	Hydromet (%)	Overall (%)
Copper	94.20	99.00	99.00	92.33
Nickel	72.50	99.00	98.00	70.34
Cobalt	42.00	99.00	98.00	40.75
Palladium	79.60	95.00	99.50	75.24
Platinum	76.90	95.00	99.50	72.69
Gold	75.70	89.00	99.50	67.04

Process Feed Rate

Average plant feed rate of 32,000 short tons per day for 365 days per year to give 11.68 Mtons per year.

Mining Costs

For purposes of the initial pit optimisation, AMDAD prepared a preliminary mining cost estimate of US$1.30 per short ton of rock (ore and waste) at surface, with an increase of US$0.02 per ton for each 20 feet of depth. An additional US$0.05 per ton was added to the ore cost to cover grade control. Rail haulage operations from the pit to the concentrator were estimated at US$0.25 per ton. This value was based on approximately comparable rates elsewhere on the Iron Range.

Note that the preliminary cost estimates shown above only provided the basis for a Whittle pit optimisation; they are not the mining costs used in the economic evaluation. The DFS mining cost estimate was developed from budget-level cost estimates provided by prospective mining contractors who had prepared quotes against the preliminary mine plan and schedule developed from the initial pit optimisation. The DFS economic evaluation is based on mining costs synthesised from mining contractor quotes.

Processing Costs
Bateman's preliminary estimate of the variable processing cost used as input for pit optimisation was US$5.96 per ton. This value comprises labour, maintenance parts, reagents, consumables and power. As with mining costs, this value is not that which was used for economic evaluation. The DFS processing cost estimate used for economic evaluation was developed from further extensive process design and engineering work by Bateman.

Site Fixed Costs
Site fixed costs used for the optimisation were estimated at US$18.9 million per year. This comprises administration and owner's workforce costs.

Metal Prices
Table 19-6 shows the metal prices assumed for base case pit optimisation exercises. These prices are lower than those assumed for economic evaluation purposes therefore pit optimisation results are considered conservative. Realisation costs are the assumed costs of treatment or refining and are deducted from the base case price to provide the net realisable metal value.

Table 19-6 Metal Prices and Realisation Costs used for Pit Optimisation

Metal	Reserve Estimate Price	Realisation Costs	Net Price
Copper	US$1.25/lb	0%	US$1.25/lb
Nickel	US$5.60/lb	25%	US$4.20/lb
Cobalt	US$15.25/lb	40%	US$9.15/lb
Palladium	US$210/oz	US$17.00/oz	US$193.00/oz
Platinum	US$800/oz	US$18.00/oz	US$782.00/oz
Gold	US$400/oz	US$9.50/oz	US$390.50/oz

The metals prices used for optimisation are significantly lower than current prices and are also lower than metal prices used for the economic evaluation which is reported in Section 25.2.

Because PolyMet anticipates that metal prices will remain higher than those shown in Table 19-6 into the future, further pit optimisation iterations will be carried out using higher prices which will have the effect of increasing both the optimised pit shell size and its contained metal. Because the optimised pit forms the basis of mine design, the production schedule and hence the Reserve estimate, the reported

Mineral Reserve can be considered conservative. This conservative approach was deliberate and provides a measure of the project's robustness.

The optimisation used as the basis for DFS mine planning results in a 3-stage pit with separate final shells in the south west and north east. The total tonnage and average grade of material contained within the optimum pit shell is set out in Table 19-7 below. It should be noted that to comply with NI 43-101 reporting requirements, this base case optimization assigned value only to Measured and Indicated blocks. Inferred blocks were regarded as waste.

Table 19-7 Optimum Pit Shell Tonnage & Grade

Tons	Copper %	Nickel %	Cobalt ppm	Palladium ppb	Platinum ppb	Gold ppb
206,000,600	0.29	0.08	73	263	71	36

The overall average waste to ore ratio before addition of ramp access, scheduling considerations and pit design refinements is 1.35 tons of waste per ton of ore.

20. OTHER RELEVANT DATA AND INFORMATION

20.1 Open Pit Design

A staged pit design was prepared based on the results of a Whittle pit optimization. Pit design includes batter angles and catch berms consistent with the Golder recommendations and 10% grade access ramps down the pit walls. The design has two separate pit shells. The larger, deeper West Pit is 880 feet deep while the East Pit shell extends to 760 feet below surface.

Both shells include pit staging. The West Pit shell has two separate internal stages which could be mined concurrently. The East Pit shell has two nested stages which must be mined sequentially and a third, lower grade stage adjoining the western end which can be mined at the completion of the main shell.

The stage designs maintain at least 150 feet width on each bench and the ramp positions are designed to maintain access as the lag height between successive stages varies. The ramps are also positioned so that the pit exit points minimise the out of pit haul distances to the rail transfer hopper located near the pit rim and to the waste stockpiles.

Most of the ore mined will be truck hauled directly to a rail transfer hopper which is located at ground level south of the approximate centre of gravity of the deposit. From here the uncrushed run-of-mine (ROM) ore will be railed to the Primary Crusher located approximately eight miles to the west. Some lower grade ore will be stockpiled temporarily on a prepared, impermeable base near the rail transfer hopper.

20.2 Mining Operations and Production Schedule

Although owner mining will be assessed in more detail at the next stage of engineering, for the purposes of this study it has been assumed mining will be carried out by a mining contractor. Apart from reducing equipment fleet acquisition costs, use of an established, reputable mining contractor will reduce the risks associated with a start-up mining operation. Using an established, experienced mining contractor will also provide PolyMet with access to the contractor’s management, personnel, safety and operational support systems which will facilitate a smooth entry to production.

Mining will be by two 16 to 20 cubic yard class electro-hydraulic shovels or electric rope shovels supplemented by a single 12 cubic yard wheeled loader for operational flexibility. The truck haulage fleet will consist of up to fifteen 100 - 120 ton class diesel, rigid body, rear dump haul trucks. Support equipment will include bulldozers, graders, water trucks, service vehicles, a rough terrain crane, pit pumps, lighting towers, compactors, trailing cable reelers and other ancillary equipment.

All material except till and surficial overburden will be blasted and two electric rotary drills capable of drilling 9.875 inch holes to 45 feet deep will be used for blast hole drilling.

Grade control will be an essential and integral part of the mining operation, not only for ore grade control but also to identify and categorise waste rock according to sulphur content. Sulphur content will determine the degree of reactivity of waste rock and hence the type of stockpile on which it must be placed. Ore grade control sampling will use a single, dedicated reverse circulation drill rig while waste rock characterisation will rely on blast hole sampling. Transition zones defined by block modelling will first be drilled for grade control to determine the position of the ore/waste contact. The stockpile destination of waste rock will then be determined by blast hole sampling.

Mining bench heights will be varied to suit the thickness and continuity of the ore zones defined by grade control drilling. The initial assumption is for 20 foot high benches with potential to increase in increments of 20 feet in waste zones, though with the equipment nominated the maximum bench height will be 40 feet.

The Base Case mine production schedule shown in Table 20-1 is based on Proved and Probable category material only and runs for 19 years. Using the same Base Case assumptions, designs were developed which included Inferred Resources for the purpose of planning further resource drilling and for determining the position of mine infrastructure and waste stockpiles. Based on the current resource estimate these extended schedules, which include Inferred category material but which were not used for economic evaluation, show potential for the mine to deliver 32,000 short tons per day for up to 22 years.

Drilling is planned for the winter of 2006-2007 with the objective of promoting material currently defined as Inferred to Indicated or Measured Resource category. Historically targeted drilling has achieved a relatively high conversion rate. This, combined with the knowledge and understanding of the deposit resulting from previous drilling, leads PolyMet to be confident that the planned, carefully targeted in-fill drilling will result in a positive change in the Mineral Resource with potential to optimise the mine plan and hence increase the Mineral Reserve. The opportunity will also be taken to run additional iterations of the mine plan and schedule and to review equipment fleet size.

As currently defined, the resource lends itself to the development of two separate pits. Since a significant proportion of the ore is concentrated adjacent to the footwall, each pit stage tends to incur high waste to ore ratios near surface but very low ratios near the base of each mining stage. Accordingly, the mining sequence aims to balance stripping ratios and ensure continuity of mill feed by always having one face in the upper benches of a stage while the other face is nearing the base of another stage. This approach will result in a regular total mining volume of 13 to 15 million cubic yards of ore and waste per annum over most of the mine life.

Because the reduction in working room at the base of the stage cone causes increased delays for blasting, grade control and floor preparation production rates were reduced over the lower 100 to 150 feet of each stage. This is particularly evident over the last five years of mining when only the final stage of the West Pit is in operation.

A short period of ore mining occurs during the pre-production mine development period (Year 0) to establish sufficient ore stocks for the mill to run during the start-up phase without interruption. The schedule then targets high grade ore to allow the mill to be fed at an elevated cut off grade for as long as possible. As a result, lower grade ore accumulates in the ROM ore stockpile until it peaks at 5.7 million tons in Year 13. From Year 14 onwards only the final West Pit stage is in operation and the stockpile is drawn down to maintain the feed capacity of the concentrator.

Table 20-1 Mine Production Schedule – Proven and Probable Category Material Only

Period	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Ore Mined											
M+I only (s tons x 1,000)	78	6,469	11,935	13,903	10,470	12,692	12,599	12,729	9,879	11,080	14,013
Cu %	0.29	0.31	0.33	0.36	0.38	0.28	0.29	0.30	0.28	0.25	0.28
Ni %	0.09	0.09	0.09	0.10	0.10	0.09	0.09	0.09	0.08	0.08	0.08
Co ppm	72	76	71	78	81	75	73	71	72	73	72
Pd ppb	267	309	317	332	299	247	261	278	272	234	247
Pt ppb	62	80	89	103	78	64	71	77	72	69	70
Au ppb	30	39	43	49	41	34	37	39	36	32	34
Volumes (cu. yds x 1,000)											
Ore	31	2,598	4,793	5,584	4,205	5,097	5,060	5,112	3,967	4,450	5,628
Waste	1,771	8,172	8,386	9,216	10,764	9,697	9,056	8,951	9,692	9,651	8,525
Ratio	56.85	3.14	1.75	1.62	2.54	1.92	1.79	1.77	2.46	2.16	1.52
Waste Breakdown (cu.yds x 1,000)											
Till	1,669	1,618	24	570	1,424	1,864	1,563	489	0	0	0
0 to 0.24%S	132	13,523	17,135	18,092	19,531	16,118	16,249	17,512	21,541	20,773	18,250
0.24 to 1.00%S	43	2,463	1,988	2,322	2,276	1,994	1,690	2,968	2,072	2,743	2,403
> 1.00%S	0	148	1,416	795	1,081	1,285	520	307	221	118	250
Stockpiles End of Year											
(S tons x 1,000)	78	240	494	2,718	1,507	2,519	3,438	4,487	2,686	2,085	4,419
Process Feed											
(s tons x 1,000)	0	6,307	11,680	11,680	11,680	11,680	11,680	11,680	11,680	11,680	11,680
Cu %	0.00	0.32	0.33	0.39	0.37	0.29	0.30	0.31	0.27	0.25	0.30
Ni %	0.00	0.09	0.09	0.10	0.10	0.09	0.09	0.09	0.08	0.08	0.09
Co ppm	0	76	71	82	79	76	74	72	71	72	73
Pd ppb	0	312	319	354	291	253	264	282	262	233	263
Pt ppb	0	80	90	108	79	67	71	76	71	69	74
Au ppb	0	39	43	51	41	35	37	39	35	32	35

Table 20-1 Mine Production Schedule – Proven and Probable Category Material Only/Continued

Period	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22
Ore Mined												
M+I only (s tons x 1,000)	11,121	12,736	12,443	10,360	4,083	9,334	10,231	8,467	8,894	7,598	9,566	6,026
Cu %	0.30	0.26	0.27	0.28	0.25	0.26	0.30	0.32	0.32	0.29	0.36	0.38
Ni %	0.08	0.08	0.08	0.09	0.07	0.08	0.08	0.08	0.09	0.08	0.09	0.10
Co ppm	70	70	73	83	68	72	73	74	75	73	72	72
Pd ppb	279	272	270	262	261	256	265	254	236	207	325	342
Pt ppb	78	73	73	74	74	78	91	77	74	62	92	103
Au ppb	37	36	36	37	37	37	41	39	36	34	51	51
Volumes (cu. yds x 1,000)												
Ore	4,466	5,115	4,997	4,161	1,640	3,749	4,109	3,400	3,572	3,051	3,842	2,420
Waste	9,595	8,930	7,468	8,238	10,550	8,653	6,892	6,090	6,328	7,128	4,178	302
Ratio	2.14	1.73	1.50	1.84	3.81	1.87	1.08	1.29	1.34	2.43	0.83	0.21
Waste Breakdown (cu.yds x 1,000)												
Till	1,830	0	0	2,198	39	0	0	0	0	0	0	0
0 to 0.24%S	15,079	16,859	13,902	11,904	22,945	19,052	16,136	13,443	13,114	13,617	7,801	113
0.24 to 1.00%S	3,518	4,936	4,358	2,648	2,814	2,010	680	1,348	2,231	3,681	2,366	551
> 1.00%S	346	111	108	293	202	274	171	101	111	130	95	53
Stockpiles End of Year												
(S tons x 1,000)	3,860	4,916	5,679	5,271	10	0	877	0	648	0	1,320	
Process Feed												
(s tons x 1,000)	11,680	11,680	11,680	11,680	9,344	9,344	9,344	9,344	8,246	8,246	8,246	7,346
Cu %	0.29	0.26	0.27	0.28	0.24	0.27	0.34	0.34	0.36	0.28	0.43	0.30
Ni %	0.08	0.08	0.08	0.09	0.07	0.08	0.09	0.09	0.09	0.08	0.11	0.08
Co ppm	70	71	73	83	69	73	78	77	77	71	75	69
Pd ppb	272	268	277	266	242	263	309	283	284	218	378	263
Pt ppb	77	74	74	74	69	78	102	84	83	70	106	83
Au ppb	36	36	37	37	35	38	45	42	42	36	57	41

Year 1 includes a provision for progressive production ramp-up with the concentrator and hydrometallurgical plant running at full capacity of 11.68 million tons per annum from Year 2 to Year 14. Because project economics can only be based on Proven and Probable Mineral Reserves the production schedule shows a reduction in mill feed from year 15 onwards. To a large extent this is an artefact of reporting requirements. Some of the material currently reported and scheduled as waste is, in fact, Inferred category material. Table 20-2 below shows the Inferred category material included within the mining envelope (but not economic evaluation) from year 15 onwards and which, if upgraded by drilling, would extend mine life.

Table 20-2 Effect on Mine Life of including Inferred Category material in the Production Schedule after Year 14

Period	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22
Ore Mined									
M+I only (st x 1,000)	10,360	4,083	9,334	10,231	8,467	8,894	7,598	9,566	6,026
Inferred (st x1,000)	912	2,774	3,338	5,432	3,421	2,900	2,881	2,951	955
M+I material grades									
Cu %	0.28	0.25	0.26	0.3	0.32	0.32	0.29	0.36	0.38
Ni %	0.09	0.07	0.08	0.08	0.08	0.09	0.08	0.09	0.1
Co ppm	83	68	72	73	74	75	73	72	72
Pd ppb	262	261	256	265	254	236	207	325	342
Pt ppb	74	74	78	91	77	74	62	92	103
Au ppb	37	37	37	41	39	36	34	51	51

20.3 Rail Ore Loading and Haulage

The LTVSMC taconite mining operation depended entirely on rail transport of ore to the Primary Crusher. To minimise capital cost, PolyMet plans to re-use large parts of the former LTVSMC railroad system which will be refurbished to transport run of mine ore approximately nine miles from the NorthMet open pit to the Primary Crusher at a planned rate of 32,000 short tons per day, 365 days per year.

Truck haulage from mine to primary crusher was evaluated as an alternative to rail haulage but was rejected on the following grounds:

- Rail haulage is a well proven, reliable and demonstrably cost effective method of hauling over long distances;
- Extensive use of rail haulage on the Iron Range results in locally available operating expertise, and support facilities (engineering, vendors etc.)
- Truck haulage would require extensive widening and up-grading of the Dunka Road resulting in additional wetlands impacts;

- Truck haulage would result in additional fugitive dust and diesel emissions with corresponding permit compliance requirements;
- Winter time trucking operations may be subject to weather delays that would not effect railroad operations to the same extent;
- The dump pocket at the primary crusher would require extensive modification to convert to truck rear dumping at an estimated capital cost in the order of US$10-12 million.

Consequently, ore will be mined conventionally and transported by mine haul trucks to a rail transfer hopper located near the pit rim. With a live storage capacity of 3,600 tons, the rail transfer hopper will allow for rapid and efficient loading of rail cars while effectively separating and de-coupling the mining and rail haulage systems. Storage capacity provided by the rail transfer hopper plus the adjacent ore stockpile will allow a degree of independence between the mining and the rail haulage systems; however, limitations on ore storage capacity in the crushing system and at the Concentrator will require railroad haulage to operate 7 days per week, year round to ensure concentrator feed can be maintained.

The rail transfer hopper will be constructed from reclaimed and refurbished components of two approximately similar structures which PolyMet has acquired from Cleveland-Cliffs. Built in the latter part of the 1990's and known as "Super Pockets", the two former LTVSMC rail transfer hoppers transferred taconite ore very efficiently from mine haul trucks to rail cars until closure in 2001. PolyMet has already recovered for re-use the mechanical, hydraulic and electrical components of these two hoppers and proposes to build a single, purpose-built structure, similar to the original LTVSMC hoppers, on the south side of the NorthMet pit. The newer equipment will be refurbishment for reactivation while the second, older, set will be retained and refurbished in due course as operating spares.

Figure 1 shows one of the two LTVSMC transfer hoppers operating with taconite while Figure 20-2 shows the mechanical loading arrangement of the same transfer hopper before the equipment was salvaged. Equipment condition is good and estimates for its refurbishment have been obtained from original equipment manufacturers. PolyMet is confident that the re-built system will work efficiently and cost-effectively.

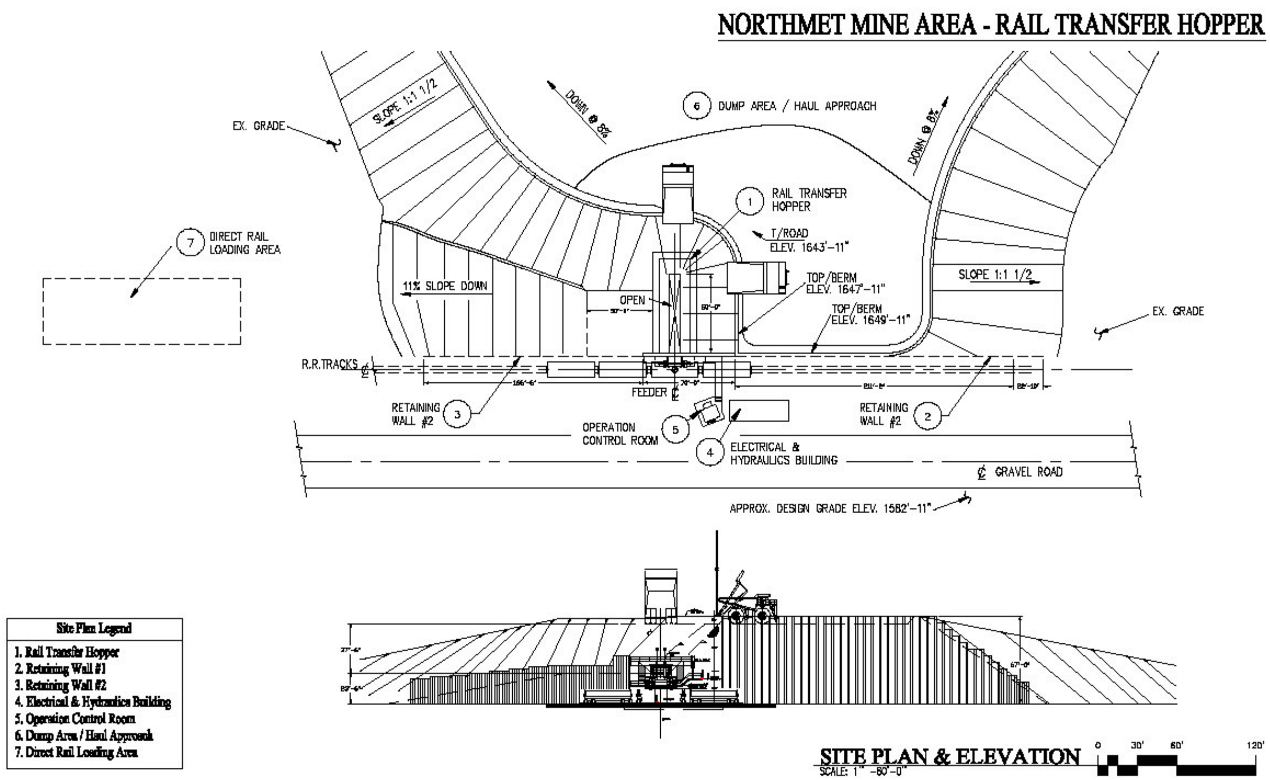
Figure 20-1 LTVSMC Rail Transfer Hopper in Operation

Figure 20-2 LTVSMC Rail Transfer Hopper Mechanical Equipment before recovery by PolyMet.

To connect the rail transfer hopper to the Primary Crusher, a total of 10,600 feet of new track will be constructed along with installation of 1,600 new ties and 3,000 feet of new rail in existing track. The existing Main Line will not require upgrading as it has remained in irregular service since closure of the LTVSMC facilities. The new sections of track construction will include 5,000 feet of spur line to connect the transfer hopper to the existing main line, and 5,600 feet of new track to connect the mainline and existing track running to the Primary Crusher. Much of the latter will utilise former rail bed from which ties and rail were removed prior to acquisition by PolyMet. Design includes provision adjacent to the rail transfer hopper for direct loading of railcars using front end loaders in the event of hopper breakdown or non-availability.

Figure 20-3, below, shows the design of the rail transfer hopper to be constructed on the southern side of the NorthMet pit. This design incorporates many features of the former LTVSMC loading hoppers which operated very successfully until closure.

Figure 20-3   NorthMet Rail Transfer Hopper Design



The rail infrastructure will be refurbished to safely meet operational requirements at minimal capital cost with periodic rail and tie replacement during mine life to maintain serviceability.

PolyMet is acquiring from Cleveland-Cliffs 120 side dumping, 100 ton capacity DIFCO railcars formerly used by LTVSMC to transport run of mine taconite to the Primary Crusher. These rail cars, which are not self-dumping, are very robust and have been inspected by KOA who have developed an estimate to restore the fleet to operational condition. The strategy is to initially restore the fleet to safe and reliable operating condition at minimal cost. Once the mine is operational and generating cashflow, rolling stock will undergo progressive restoration/rebuilding as required to minimise operating costs for the remaining mine life.

Drawing on the experience of LTVSMC railroad operations, haulage simulations were used to model the PolyMet operation and determine optimum equipment requirements and cycle times to ensure an integrated mining, rail haulage and ore beneficiation operation. The optimum configuration is to use 30 car trains powered by two 3,000 horsepower diesel electric locomotives. One locomotive will be placed at the front and rear of each train in a manner known as distributed power. Railroad operating experience elsewhere has shown that distributed power reduces drawbar loadings, improves operational flexibility and is preferred for safety reasons. Under full production conditions, the railroad will operate two 12 hour shifts per day, 365 days per year. Locomotives will be equipped with remote controls to allow one-man crew operation including loading at the rail transfer hopper. Additional resources will be engaged in track and rolling stock maintenance. As is common practice in the United States, PolyMet will lease locomotives on the basis of engine running hours with the lease fee including allowance for maintenance but not major overhauls or re-builds; PolyMet will retain responsibility for routine service and minor maintenance. Locomotives will be replaced before major overhauls are due.

Table 20-3 below, shows how the railroad ore haulage operation will ramp up during the plant commissioning and ramp-up period. Initially the railroad operations will be single shift, increasing to two 12 hour shifts per day as plant utilisation increases. Although the rail haulage system has capacity to meet plant feed requirements working round the clock on 5 days per week, buffer storage capacity within the crushing system ahead of the mill feed bins is limited and insufficient to enable a complete weekend shut-down of rail haulage. Railroad operations will, therefore, continue 7 days per week, 365 days per year.

Table 20-3 Train Fleet Buildup (30 cars/train)

% Full Production	Round Trips/day	No. of operational cars required (incl. spares)	Operating Crews/day
50	6	70	3
70	8	100	4
85	10	100	5
100	12	100	6

20.4 Stockpiles and Waste Stockpiles Design

Although waste rock contains only very small to trace amounts of sulphur, mostly in the form of sulphides, characterisation testwork carried out as part of the Environmental Impact Statement (EIS) review work has indicated a possibility that base metals may leach from waste rock when exposed to precipitation. To protect against possible environmental degradation and to ensure the operation complies with water quality discharge standards, waste rock will be categorised according to its sulphur content and placed selectively on waste stockpiles designed specifically for each of the waste categories. Waste categories on which stockpile designs are based are as follows:

- Category 1 – Sulphur content is less than 0.12%S and between 0.12% and 0.31% if Cu/S is more than 0.3. This material is considered to be non-reactive and will not generate acid or leach metals if the water flow path is kept to a maximum of 30 feet. This category currently comprises approximately 77.5% of the total waste rock volume.
- Category 2 – Sulphur content is less than 0.12%S and between 0.12% and 0.31% if Cu/S ratio is less than 0.3. This material is considered to have low reactivity when the flow path exceeds 30 feet. Effluent from a Category 2 stockpile will not be acid but may contain minor amounts of leached base metals and will require treatment before discharge. Category 1 waste rock becomes Category 2 if the flow path is greater than 30 feet.
- Category 3 (Lean Ore) - Sulphur content is greater than 0.12% but less than 1.0%S with a Cu/S ratio more than 3. This material has marginal value at current metal prices but will be segregated to allow for future recovery should economic conditions change. Considered as being of medium reactivity, this category comprises 14.5% of the total waste rock. Stockpile effluent may be acidic and may require treatment before discharge.
- Category 3 - Sulphur content is greater than 0.31% but less than 1.0%S with a Cu/S ratio of less than 3. Considered as being of medium reactivity, this category comprises 4% of the total waste rock. Stockpile effluent may be acidic and may require treatment before discharge.
- Category 4 – Sulphur content is greater than 1.0%S plus all Virginia Formation rock. Considered to have high reactivity, this category comprises 3% of the total waste rock mined and effluent may require treatment to meet regulated discharge standards.
- Glacial till is non-reactive and require no special stockpile arrangements. Because they are inert, till and overburden will be used as required for mine site civil works and for construction of waste stockpile grading layers and water impounding dikes.

To comply with the Minnesota mining regulations, waste stockpile design will utilise a base or foundation liner and a cover system to encapsulate the contained waste rock and restrict entry of precipitation and snow melt.

- Category 1 material will require no liner provided flow paths are restricted to no more than 30 feet.
- The liner system for Category 1 / 2 stockpiles will consist of compacted till soils with a maximum permeability of $5x10^{-7}$ centimetres per second (cm/sec) and an overliner drainage layer to provide for a calculated leakage rate of 462 gallons per acre per day (gal/acre/day).
- Category 3 stockpiles will have a sub-grade of compacted local till soils, with a maximum permeability of $1x10^{-5}$ cm/sec, overlain by a geomembrane liner and drainage layer to provide for a calculated leakage rate of 8 gallons per acre per day (gal/acre/day).
- For high reactivity Category 4 waste rock and the Ore Stockpile, the liner system will consist of a minimum of one foot of compacted till soil with a maximum permeability of $1x10^{-6}$ cm/sec

overlain by a geomembrane liner and drainage layer to provide for a calculated leakage rate of 1 gallon per acre per day (gal/acre/day).

Each liner system will be constructed on a geotechnically stable foundation (to minimise the risk of liner distortion and rupture) and graded to provide effluent collection points around the periphery of the stockpile. Effluent will be channelled or pumped from these collection points to a central sump for pumping to the water treatment plant. To minimise earthmoving during construction and to facilitate effluent drainage collection, waste stockpiles will be preferentially constructed on areas of higher ground where the depth of surficial till and soil covering is minimal.

Stockpile design provides for an average waste rock porosity of 30% and a minimum 100-foot setback from property boundaries, with additional setback provided for critical corridor areas. A minimum crest width of 200-ft has been used for efficient mine haulage traffic. The final profiles of out-of-pit waste stockpiles were designed in accordance with the Minnesota Department of Natural Resources, Rules for Nonferrous Metallic Mineral Mining. After placement of an initial grading layer made up of non-reactive waste rock and an impermeable solution collection membrane/liner, stockpiles will be formed in 40 foot high lifts with a 30 foot wide berm on the outside face at the base of each lift. Batter slopes between the berms will be at an angle of 1.0 vertical to 2.5 horizontal.

Table 20-4 below shows, in acres, how the footprint of each waste stockpile will develop over time. This development schedule allows for backfilling the East Pit with Category 1 and 2 material after the pit is depleted in production year 11. These footprints are dynamic because of ongoing waste characterization but are not expected to change significantly. Backfilling the mined void will reduce haulage costs and contribute to reducing the cost of final site closure and rehabilitation.

Table 20-4 Waste Stockpile Footprint Areal Development Over Time (Acres)

Stockpile	Year 1	Year 5	Year 10	Year 15	Year 20
Category 1/2 East	34	144	212	285	313
Category 1/2 West	65	176	340	432	466
Category 3	9	29	42	49	57
Category 3 Lean Ore	18	42	69	76	89
Category 4	9	34	42	44	49

As a general strategy, stockpiles will be capped and re-vegetated progressively through mine life to minimise precipitation inflow. An evapo-transpiration cover system is considered the most effective means of preventing meteoric water inflow, for controlling run-off and for promoting eventual site re-vegetation and will be used on all stockpiles, regardless of classification.

Glacial till and surficial overburden will be removed ahead of mining on an 'as-required' basis and will be either stockpiled or, after screening out boulders, used for mine site civil construction or for capping waste rock stockpiles. Overburden and till stockpiled during operations will be extensively re-used towards the end of mine life to cap waste rock stockpiles, for re-contouring disturbed landforms and for general site rehabilitation.

20.5 Mine Site Infrastructure and Electrical Power Reticulation

20.5.1 Mine Site Facilities

Apart from the rail loading hopper, facilities at the NorthMet mine site will be kept to a minimum. A covered field service and refuelling facility with temporary storage tanks will be set up in the vicinity of the rail transfer hopper. As is common at taconite mining operations in the area, fuel oil will be supplied direct to the end-user by a local supplier who will also be responsible for its storage and distribution. In much the same way, a local supplier of explosives and blasting accessories will provide an 'in-hole' service delivering and placing explosives directly into blast holes. The supplier will be responsible for storing and delivering explosives and hence no onsite explosives magazine will be required.

20.5.2 Mine and Railroad Offices and Staff Facilities – Area 2 Workshops & Offices

Offices and change-house facilities for mine and railroad operating and technical personnel will be provided by refurbishing existing facilities located adjacent to the railroad and about two miles east of the Primary Crusher. Known as the Area 2 Shop, this facility includes a large building which will house the refurbished offices and personnel facilities as well as a workshop, complete with overhead crane that will be set up for railroad rolling stock maintenance.

20.5.3 Mine Mobile Equipment Maintenance Facility – Area 1 Truck Shop

This study assumes the mining contractor will be responsible for equipment fleet maintenance and that all associated costs are included in the contract rates used to develop mine operating costs. PolyMet now owns the former LTVSMC mine mobile equipment maintenance complex known as the Area 1 Truck Shop. This will be refurbished and reactivated for use by the mining contractor. Area 1 Truck Shop is a purpose-built, fully enclosed, winterised, heavy mobile equipment maintenance facility located about one mile west of the process plant site. Comprising six truck bays (capable of accommodating haul trucks up to 240 ton payload class), three miscellaneous heavy equipment bays, a two-stall, enclosed truck wash down bay and associated shops, lunch room, offices, storage capacity, change house and ablution amenities, this facility is ideal for maintaining the mining equipment fleet. Although it is located about nine miles from the mine site, access between the two will be in part via the existing, upgraded Dunka Road and in part through former LTVSMC mine areas (now inactive) to avoid mixing light and heavy vehicular traffic in the vicinity of the Area 2 Offices. The minor inconvenience of having to move equipment between the mine and the workshops is off-set by having a ready-made, comprehensive maintenance facility available at very low capital cost.

Figure 20-5 and 20-6 show the Area 1 Truck Shop as it is at present.

Figure 20-5 Area 1 Truck Shop from the north west showing the six haul truck maintenance bays and truck wash down bay (nearest camera).

Figure 20-6 Area 1 Truck Shop viewed from the south east showing the tracked equipment bays and tyre shop.

20.5.4 Mine Site Electrical Power Distribution

Electrical power for the major items of mining equipment (excavators, blast hole drills, dewatering pumps, powering the rail transfer hopper facility and for ancillary services) will come from the nearby

138kV transmission line owned and operated by local power utility, Minnesota Power (MP). For cost estimation purposes it has been assumed that the power utility will provide the main step down transformer at the mine site as well as the connection from the 138kV transmission line. From there power will be distributed around the open pit by means of a single circuit line suspended from wooden poles. This supply line will be extended periodically as required by the changing nature of the ongoing mining operation. PolyMet has already acquired sufficient 4,160 volt, skid-mounted substations to meet the start-up requirements of the mining fleet though it is anticipated that additional substations and extension of the in-pit power line will be required in years 6 and 12.

20.6 Mine Site Water Management

Mining activities will alter the land use characteristics of the mine site, which in turn, will impact the quantity and quality of water that leaves the site. In addition to excavation of the open pit, the mine site, which is currently forested, will be modified by construction of access roads, haul roads, rock stockpiles, water collection sumps, ponds, pipes, ditches, and mine infrastructure, all of which will affect the area's water balance. Moreover, water that contacts ore or waste rock may have dissolved metals and must be collected and possibly treated before it can be discharged; however, water that collects on undisturbed portions of the mine site can be routed into its natural drainage system, the Partridge River. Site water management takes into consideration variations in the water balance during mine life as the shape and size of the open pit changes and as reclamation progresses.

Surface drainage onto or off the mine site will generally be controlled by the construction of dikes around the perimeter of the site or by natural drainage area divides. These dikes, which also serve to prevent dewatering of adjacent wetlands, will be constructed during the mine pre-production development phase. Typically constructed of overburden material excavated ahead of mining, these dikes and ditches will be located along the mining area perimeter and along the rim of the open pits as needed to minimize surface runoff into the pit.

During the normal course of mining operations, temporary sumps will be routinely excavated at suitable low points to collect run-off and to assist with pit de-watering. Depending on location and operational considerations, electrical or diesel pumps will be used to pump water to collection sumps at original ground level near the pit rim prior to further treatment. Groundwater flow rates from the Duluth Complex and the Virginia Formation will be variable and are expected to increase with time as the pits expand both laterally and vertically. Although inflows from precipitation will vary depending on the time of year, not all of precipitation within the pit perimeter will make it to the pumping area. Some will be held in depressions and either infiltrate the rock mass or evaporate. Based on pumping tests and hydrogeologic modeling the expected inflows to the pits from groundwater and precipitation runoff will increase to a maximum in year 11 as the pits get larger and deeper, with maximum inflows ranging from 1,100 – 2,000 acre-feet/year. Thereafter, total inflow will decrease as groundwater inflows decrease after mining and dewatering in the East Pit cease and the East Pit is backfilled with waste rock.

To keep pumping equipment costs and sump sizes to reasonable levels, sump and pump capacities will be designed to handle average predicted water inflows rather than maximum design storm event (100 year storm event) inflows. In the event that a storm temporarily exceeds the design capacity of the pumping system overflow will be directed to the open pit where the lowest levels will be used as an emergency sump with mining operations re-scheduled to other parts of the mine until water levels are pumped down to normal.

Drainage water accumulations from stockpile liners, the rail transfer hopper, and road runoff from within the active mine area will also be controlled because they may contain dissolved metals. Runoff from each of these areas along with all water that has contacted ore or reactive waste rock will be collected in a sump from where it will be pumped to a Central Pumping Station (CPS) located in the south western corner of the mine area. From there, mine waste water will be pumped to a Waste Water Treatment Plant (WWTP) located at the Area 2 Workshops.

The CPS will be a lined pond that will temporarily store mine water prior to being pumped to the Waste WWTP where, if necessary, it will be treated for dissolved metals prior to being directed to the tailings basin. If no treatment is necessary, mine site water will bypass the WWTP and discharge directly to the tailing basin. Under normal operating conditions the metallurgical process is a net water consumer and plant make-up water will be drawn from either the tailings basin or from Colby Lake, if required. Water balance modeling shows that under anticipated conditions recycling mine water to the process plant allows fresh water make-up from Colby Lake to be minimised. If, however, mine waste water volumes exceed expected levels, the tailings basin will be used to provide buffer storage capacity and plant fresh water make-up from Colby Lake can be further reduced accordingly.

Water quality data from test bores indicate that groundwater inflow to the open pit, which represents approximately 80 percent of the yearly average flow from the mine site, will not contain significant concentrations of metal contaminants. Data from kinetic weathering tests (designed to simulate the effects of long term exposure of NorthMet material to weathering) show that contact water may contain dissolved copper and nickel and that, from certain rock types, concentrations of these metals may exceed appropriate water quality discharge limits for these parameters. At these levels mine waste water will need treatment before being pumped into the tailings basin. Only about 23% of total waste rock contains sufficient sulphur to generate acid and approximately two thirds of this is lean ore.

Water management options at the mine site such as isolating runoff water from reactive waste rock and ore stockpiles may alleviate the need to treat all of the waste water streams generated at the mine site. However, since water quality predictions are based on limited data and more reliable information will become available as mining progresses, DFS design and costing have been based on conservative quality and flow rate assumptions.

A variety of possible treatment technologies including chemical precipitation, reverse osmosis, and ion exchange were examined for suitability to remove metals. Conventional chemical precipitation was selected and a bench-scale treatability testing program was developed to evaluate its effectiveness.

Because there are no stockpiles of weathered NorthMet material to provide useable quantities of stockpile seepage water for testing, an alternative source was required. Effluent samples taken from stockpiles of Duluth Complex material in the near-by, inactive Dunka taconite open pit were analysed for metals and other parameters and their quality was compared to predicted NorthMet effluent quality. Dunka effluent was found to be at least as “bad” as that predicted from NorthMet material and, with MDNR approval, was used as a surrogate for NorthMet effluent in treatability tests. (Effluent from the Dunka Pit (inactive taconite pit) Duluth Complex stockpiles has long been recognised as problematic and has been extensively studied by the MDNR and others for more than 17 years. The MDNR therefore has a large amount of data on the weathering characteristics of Duluth Complex material, albeit with higher sulphur content than NorthMet material).

Testwork conducted by Canadian Environmental & Metallurgical Incorporated (CEMI) located in Burnaby, British Columbia showed that using conventional chemical precipitation at a lime dosage rate of approximately 400mg/L, copper and nickel content could be reduced from 0.281 mg/L Cu and 1.48 mg/L Ni to 0.0054 mg/L and 0.0076 mg/L, respectively. These levels will meet appropriate surface water discharge quality limits.

The range of anticipated wastewater flow rates to the treatment plant will vary according to the time of year with the maximum flow of over 4,000 gpm occurring during spring snowmelt. The average annual flow is estimated to be approximately 2,000 gpm. To handle these variations two 2,000-gpm treatment lines will be constructed with the second line operated as and when required to deal with increased flow rates. Principal reagents will be lime and carbon dioxide (as a by-product of the Oxygen plant) for pH control. The resultant sludge will be dewatered and returned to the hydrometallurgical process for recovery of contained metals.

20.7 Existing beneficiation Plant and equipment

20.7.1 Assessment Methodology and Engineering Philosophy

At closure, the former LTVSMC facilities were a fully operational, well maintained, going concern. Shut down had been systematic and there was an expectation that the plant would be re-started at some point in the future. Prior to the start of the DFS preliminary engineering studies by Optimum Project Services Ltd., Penguin Automated Systems, Inc. and Bateman assessed the major elements of the crushing plant, milling and tailings disposal facilities and determined they were fit for the purpose of crushing and milling NorthMet ore. The exception was the original taconite flotation equipment which is to be removed and replaced with larger capacity, state-of-the-art flotation equipment engineered specifically for NorthMet ore. It was PolyMet’s expectation, therefore, that much of the plant could be reactivated at minimal cost, with up-grades restricted to areas such as environmental controls and dust extraction where stringent compliance standards are expected.

To assess the condition of existing equipment and hence to determine the risks and costs associated with re-starting it, detailed site inspections were carried out by qualified individuals who had previously worked at and knew the plant intimately. In addition to drawing on the personal experience and knowledge of former LTVSMC employees, detailed and pertinent operating data, maintenance records and reports, and supervisors' shift logs were reviewed to provide a detailed picture of the condition of the plant at closure. During July and August 2006, a number of motors including those for a crusher, a rod mill, a ball mill, feeders and various drives were successfully test-started to confirm reactivation assumptions. Existing instrumentation was also reviewed to confirm the extent to which it could be reactivated. The number of test failures was minimal thereby adding confidence that the selected plant can be re-started with limited refurbishment. Appropriate allowances are made in the Capital and Operating cost estimate for refurbishment prior to restarting equipment and subsequent staged maintenance.

Because the original LTVSMC plant had a capacity (90,000 long tons per day) nearly three times larger than that required by PolyMet, part of the design and commissioning philosophy assumed reactivation of sufficient plant and equipment to meet the expected ramp-up schedule with subsequent reactivation of additional equipment to provide spare capacity when major scheduled overhauls or maintenance work is required.

Another aspect of design philosophy relates to the use of spare equipment. There is a large amount of equipment available to PolyMet which does not need to be immediately reactivated. Therefore, PolyMet intends to refurbish some of this surplus equipment progressively to provide spares in the event of breakdown, or additional capacity in the event that some existing equipment does not perform as expected.

20.7.2 Requirements for Re-commissioning Existing Plant Facilities

Based on detailed plant condition assessments, the following activities will be necessary to refurbish and reactive the ore beneficiation facilities.

- The existing plant facilities will be cleaned up and made safe ahead of refurbishment work. This work will include removal of debris as well as asbestos removal and mitigation.
- Buildings are structurally in very good condition and need only minor repairs including some minor roof patching and drain pipe replacement due to freezing damage.
- The existing plant includes two parallel coarse crushing trains, each consisting of one 60 inch primary gyratory crusher and four 36 inch secondary crushers. To meet the planned 32,000 short tons per day throughput rate, PolyMet plans to refurbish and activate only the southerly primary plus three secondary crushers. (It is not planned to reactive the northern line at this stage but it could be refurbished at any time in the future, as required). Linking the coarse and fine crushing facilities is a conveyor system comprising two parallel belt conveyors for transporting primary crushed ore to the coarse ore bins above the Fine Crushers housed in an adjacent structure. It is proposed to immediately reactivate the west side conveyors 1A and 2A along with their ancillary equipment followed by the east side conveyors 1B and 2B as stand-by.
- Crusher maintenance records were used to determine remaining wear life and to plan and schedule subsequent maintenance. Liners and wear materials will be replaced where remaining life was identified as less than 25% original or where obviously required. Other items needing attention in the Coarse Crushing facility include the rebuild of an existing Pioneer feeder and replacement of one METSO apron feeder.
- In the Fine Crushing facility, equipment from four of the original seven lines was sold and removed prior to acquisition by PolyMet. The planned production rate requires only three fine crushing lines, each line consisting of one 7 foot standard tertiary cone crusher in series with two 7 foot quaternary shortheads. These will be arranged so as to maximise live storage capacity in the overhead coarse ore bin. There are also a variety of spare crusher frames, bowls, mantles, drive motors, conveyors and feeders, which will be refurbished for use as spares. Using LTVSMC maintenance records verified by field inspection, it was determined that only one of the three tertiary crushers requires new liners and a frame repair. The six quaternary crushers have good liners in place and will only require servicing prior to start-up. The six existing single

deck screens between the tertiary and quaternary crushers will be replaced with new double deck screens for increased screening efficiency.

- Conveyors 3A, 4B and 5N will be reactivated to transport fine crushed ore to the ore beneficiation building storage bins. As elsewhere, maintenance records were used to determine the condition of conveyor drives, bearings, trippers, feeders and related components. Visual inspection of conveyor idlers indicated about 10% would need replacement prior to start-up. Chute work will be replaced where worn.

Of the 34 original rod/ball mill grinding lines, four were sold before the facilities and equipment were acquired by PolyMet. Including spares, a total of thirty-three rod mills and thirty-two ball mills along with thirty rod mill feed conveyors and three hundred and sixty vibrating feeders remain available for PolyMet's use. However, to meet planned production it will be necessary to refurbish only twelve rod mill/ball mill grinding lines plus their associated feeders and mill feed conveyors. Mill lines 1-N to 12-N inclusive will be reactivated, though it is proposed to use and relocate the mills with the most remaining liner life.

Figure 20-7 is a photograph of the Concentrator interior showing some of the rod mills owned by PolyMet. Figure 20-8 shows one of the rod mills in more detail and illustrates its good condition.

Figure 20-7 Concentrator interior showing some of the 33 rod mills owned by PolyMet

Figure 20-8 Rod Mill – detail

There are also three 12 ft 2 in by 23 ft 4 in, 1500 hp regrind mills, one of which will be reactivated to regrind scavenger concentrate, while regrind mill 3S will be used to produce a limestone slurry for acid neutralisation in the hydrometallurgical plant. The third mill will be available as stand-by.

- The concentrator upper bay is equipped with two overhead cranes, one 200 tons capacity and one 25 tons capacity, which range over the full length and breadth of the milling level. These cranes are functional and will require only inspection and re-certification before reactivating. These cranes also provide tremendous operational and maintenance flexibility as they have sufficient lifting capacity to pick up and move a mill shell (rod or ball) complete with media charge to a central maintenance area.
- Based on mill throughput records and maintenance records, a liner replacement schedule was developed which optimises remaining liner life and forms the basis of mill capital and operating cost estimates.
- The large number of redundant mills and associated feed equipment will allow PolyMet to progressively refurbish units as required for spares. Moreover, in the unlikely event that existing equipment does not perform as expected, additional milling capacity can be brought on line quickly and cheaply.
- The ball mill feed cyclones will be re-used and elevated to allow gravity flow of cyclone overflow to the rougher flotation cells. Two cyclone feed pumps with variable speed drives were successfully test run.

- Dust collector scrubbers, bag houses and associated fans throughout the facility will be replaced with new, high efficiency units meeting current Minnesota Pollution Control Agency (MPCA) standards.
- A new sulphide flotation circuit will be installed comprising two 250 m^3 rougher flotation cells, ten 250 m^3 scavenger flotation cells and ten 17 m^3 cleaner flotation cells. To provide space within the Concentrator it will be necessary to remove remaining taconite magnetic separators, relocate some electrical gear, remove some steel work and demolish some existing concrete equipment pedestals. Despite the cost of demolition, installing new flotation equipment within the existing structure was significantly less expensive than constructing a new purpose-built facility. A feature of flotation system design was the use of gravity feed wherever possible to minimise pumping.
- Extending across the concentrator lower bay are two 10 ton overhead cranes, which range over the full length and breadth of the proposed flotation system. These cranes are functional and will require only inspection and re-certification before reactivating. As elsewhere throughout these facilities, overhead cranes will provide tremendous operational and maintenance flexibility.
- A number of possible alternative tailings pumping configurations were studied and evaluated. The most economic option requires direct pumping of un-thickened tailings to the tailings basin using existing, refurbished tailings pumps and new steel pipe.
- A 2,000 hp/ 1,500 kW IsaMill will be installed in the Concentrator building on Ball Mill 1S foundations to grind final flotation concentrate to P_{80} of 15µm prior to pumping to the hydrometallurgical plant. To control the water content of concentrate fed into the autoclaves, a new concentrate thickener will be installed outside on the west side of the Concentrator between the Concentrator and the hydrometallurgical facility.
- The existing raw, domestic, mill, service and fire water systems will be reactivated with only limited refurbishment necessary. Approximately 300 feet of buried fresh water pipeline from Colby Lake will require sleeving along with repair to associated vacuum breaker inspection pits. Elsewhere miscellaneous valves and fittings will need replacement or refurbishment. Pipe wall thicknesses through the high volume, high head mill water feed system have been measured to confirm that its condition is adequate for system pressures.
- To reduce the amount of pipe reconditioning required, two existing plant air compressors will be relocated to the Concentrator building and piped into the existing air header system.
- The original facilities made extensive use of pumped hot water and steam for plant heating; however, to avoid costly overhaul of this system, new gas-fired heating equipment will be installed and, where necessary, existing gas-fired equipment will be reconditioned. (The plant site is served by a natural gas pipeline which supplied LTVSMC with up to 13,000 M cu ft/day of natural gas at 125 psi which far in excess of PolyMet's consumption estimates.
- The original sewage treatment plant was sized for a workforce of over 2,000 employees and even if refurbished would not function adequately with the reduced bio-mass produced by PolyMet's smaller workforce. The existing plant will therefore be replaced by a new packaged system sized for PolyMet's requirements.

- The primary substation was operated continuously with a power draw of 130 MW and since LTVSMC closure parts of this substation have been kept in operation, albeit at reduced load. PolyMet will own the sub-station and intends to re-commission it to service the existing plant site facilities, the new hydrometallurgical plant facilities and the new mine service area. Substation equipment will be tested and a detailed power flow and fault current analysis will be performed as part of the re-commissioning work.
- Included in PolyMet's plant acquisition deal with Cleveland Cliffs are large numbers of spare electric motors of all sizes, MG sets, electrical switching gear, starters, motor controls and associated electrical gear. All electrical equipment will be cleaned, inspected, and tested before being re-started with ample spares available in the event that some items need to be replaced. During July and August 2006 representative samples of electrical equipment throughout the crushing and milling facilities were inspected and tested and a number of crusher and mill motors were successfully started under no-load conditions. PolyMet's expectation is that most electrical equipment will re-start satisfactorily after appropriate cleaning and lubrication.
- With few exceptions, the existing Foxboro distributed control system is in place and intact and will be recommissioned with assistance from vendors' representatives. This system has been successfully restarted with minimal failure of individual modules. The system will be upgraded with the replacement of processors and power supplies. This will allow the existing input-output modules to communicate to the new Foxboro system that will be installed in the Hydrometallurgical Plant.

20.7.3 Support Facilities and Plant Infrastructure

One of the key elements of this project is that infrastructure is well established, generally in good condition and, in most cases, requires only minor modification to accommodate new installation. Existing infrastructure and services include:

- Incoming HV power (138 kV) from the Minnesota Power grid;
- Power distribution within and around the existing facilities;
- Water supply and distribution;
- Sewage collection system (though treatment plant must be replaced);
- Guard house and related security facilities;
- Offices, changing rooms, meeting rooms, lunch rooms;
- Sample preparation and analytical laboratories;
- Warehouses and storage facilities;
- Road and on-site railroad system;
- Railroad connection to common carrier rail network;
- Workshops;
- Natural Gas Supply;

- Communications;
- Mine railroad and locomotive services and refuelling facilities; and
- Tailings disposal facilities.

All the above were evaluated in detail to determine their suitability and cost effectiveness of re-use and cost estimates have been included in this study to refurbish the existing facilities and return them to a condition suitable for safe re-use by PolyMet. Figure 20-9 shows part of the 138kV electrical switch yard and HV substation situated on the east side of the Concentrator.

In addition to the various and extensive offices available at the Coarse Crushing facility, the Fine Crushing building, in the Concentrator, associated with the General Workshop, the Unit Rebuild Workshop and the warehouse complex, PolyMet is also acquiring the former LTVSMC Administration Building located away from the main industrial area on the public road from Hoyt Lakes. This building previously housed 150-200 administration staff. PolyMet intends to use this building during the construction phase to accommodate engineering and construction management staff. Existing telecommunications, networking and fibre optic connections within the building are functional and can be fully reactivated at minimal cost.

Historically the Mesabi Iron Range has been the centre of a very large and extensive iron ore mining industry with six world-class taconite iron ore mining operations in production at this time. To support this mining activity, the area has a very well developed infrastructure which includes excellent roads, extensive railroads, access to ocean shipping via the nearby ports of Duluth/Superior, reliable grid power, engineering support services and service providers as well as a significant pool of skilled labour for construction as well as operations. PolyMet will benefit from the existence of this infrastructure which will facilitate construction and provide simplified and reliable shipping logistics for equipment, parts, consumables and product export.

Figure 20-9 Part of the 138kV electrical switch yard and HV sub-station

20.8 Permitting & Environmental

This section describes the environmental review process to which this project is subject. Details are provided regarding the Environmental Impact Statement (EIS) process under the Minnesota Environmental Policy Act (MEPA) and the National Environmental Policy Act (NEPA). This section also outlines the environmental permitting process and identifies major environmental issues and PolyMet's plans to deal with them. Finally, a summary of the current status of the environmental review and permitting process is provided.

20.8.1 Environmental Review Process

There are four major steps to the environmental review process. These include;

- Scoping of the Environmental Impact Statement (EIS),
- preparation of the draft EIS,
- preparation of the final EIS, and
- documenting the Record of Decision (ROD) regarding the adequacy of the EIS.

Interested citizens may participate in the environmental review of the project including by way of offering opinions on the Draft EIS, the adequacy of the Final EIS and suggesting ways in which potential problems can be resolved.

An EIS identifies the likely environmental impacts of the project along with alternatives that may lessen or mitigate adverse impacts. In 2005, the U.S. Army Corps of Engineers (USACE) and the Minnesota Department of Natural Resources (MDNR) signed a Memorandum of Understanding (MOU) to be the lead agencies in preparing a single joint EIS. (Absent such an MOU, PolyMet would have been obliged to prepare separate state and Federal EISs.) The MDNR will assume primary responsibility for managing the EIS preparation and review.

As envisaged in the joint MOU, the scope of the joint EIS was determined and agreed in 2005. The Scoping Environmental Assessment Worksheet (EAW) and Draft Scoping Decision Document were published in June 2005 and the USACE issued a Notice of Intent (NOI) to prepare an EIS in July 2005. The MDNR and USACE issued press releases concerning the availability of the Scoping EAW, Draft Scoping Decision Document, and the mandated public meeting which was held on 29 June 2005 in Hoyt Lakes, Minnesota. During the public comment period which ended 6 July, 2005, a total of 29 written and two verbal comments were received. As required by law, the MDNR and USACE considered all comments that were received and revised the Draft Scoping Decision Document accordingly. A Final Scoping Decision Document and Response to Public Scoping Comments were issued in October 2005.

In October 2005, the MDNR issued a Request for Proposal to consulting firms for the preparation of the EIS. The selection process was completed in March of 2006, when the MDNR selected the team of Environmental Resources Management (ERM) and Knight Píesold Consulting. Both firms have considerable experience in working with environmental matters related to the non-ferrous mining industry with Knight Píesold having specific experience with sulphide mining.

Although PolyMet as project proponent is responsible for funding the work, PolyMet had no input to the selection of the EIS Contractor which was carried out according to standard MDNR selection criteria. The EIS Contractor formally started work on 24 April 2006.

No state or federal permits may be issued until the state issues an EIS Adequacy Decision and publication of the federal ROD. Permit issue will lag behind the State EIS Adequacy Decision and publication of the federal ROD in order to allow for mandated administrative actions and comment periods. Permit issue could also lag because the law allows for challenge to permit issuance initially through an administrative process and ultimately through the courts.

20.8.2 Environmental Permitting Process

Table 20-5 provides a list of the permits required for the project. Major permits are described below.

The environmental permitting process is underway for the USACE Section 404 wetlands permit, Minnesota Pollution Control Agency (MPCA) air quality and water quality permits, and MDNR permit to mine. A preliminary application for the Section 404 permit was submitted in July 2004 and will be kept updated with current design information. An air toxics risk assessment (AERA) was submitted to MPCA in February 2005. Air quality modelling and water quality modelling studies are currently being conducted. It is expected that all applications for permits will be submitted to the appropriate agencies in time to allow the draft permit applications to be considered by the EIS contractor in preparing the draft EIS. While the majority of permits required to operate the mine have been identified, there remain a number (included in Table 20-5 below) the requirement for which remains uncertain. The reason for this uncertainty is primarily that ongoing test work and/or evaluation have not yet determined unequivocally that a permit will not be required.

PolyMet is currently in discussions with regulators regarding the EIS and permitting schedule. Although those discussions are not yet fully resolved, PolyMet expects some delay to the original schedule with submission of major permit applications by the 1st Quarter 2007, adoption of the State EIS adequacy decision by about the 3rd Quarter 2007 and issue of major permits by the 4th Quarter 2007.

Table 20-5 Required Environmental Permits and Approvals

Responsible Government Unit	Type of Application	Comments and Status
MDNR	Permit to Mine	In preparation
MDNR	Water Appropriations permit for pits and tailings basins, and mine dewatering	In preparation
MDNR	Dam Safety Permit Amendment for tailings basin for dikes at mine	Existing Cliffs Erie Permit will be transferred. To be applied for if needed
MDNR	Permit for work in protected waters, possible modifications and diversions of local streams	Application will be made if deemed necessary
MDNR	Permit for wetlands modifications under Wetland Conservation Act (as part of Permit to Mine)	In preparation
MDNR	Burning Permit (possibly needed for construction or land clearing)	Application will be made if deemed necessary
MDNR	Permit for taking of threatened or endangered species	Determination is pending whether permit required.
MPCA	Minnesota Air Emissions Permit	In preparation
MPCA	NPDES and/or SDS permit for discharge of mine dewatering water	Applications will be made if deemed necessary – current design is based on pumping mine water to a water treatment plant at the plant site, hence no permit required.
MPCA	NPDES and/or SDS permit for discharge to tailings basins	Permit application will be required although design is based on no discharge of treated waters.
MPCA	NPDES and/or SDS permit for discharge of sanitary wastewater at processing plant	To be applied for
MPCA	NPDES permit for storm water discharge	To be applied for
MPCA	Minnesota Waste Tire Storage Permit	To be applied for

Responsible Government Unit	Type of Application	Comments and Status
MPCA	General Storage Tank Permit (fuel tanks)	To be applied for
Minnesota Department of Health	Radioactive Material Registration (for low-level radioactive materials in measuring instruments)	To be applied for
USACE	Section 404 Permit for Wetland Impacts	Application being prepared
Minnesota Department of Health	Permit for Non-Community Public Water Supply System (serving an average of at least twenty-five individuals daily at least 60 days out of the year) and wellhead protection plan	To be applied for, if needed - but may not be required
Minnesota Department of Health	Notification of Water Supply Well Construction	To be provided when well(s) constructed
Minnesota Department of Health	Permit for Public Onsite Sewage Disposal System	To be applied for, if needed
City of Babbitt	Building Permit for buildings at mine site	To be applied for

Note: NPDES/SDS stands for National Pollutant Discharge Elimination System and State Disposal System.

MDNR Permit to Mine: One of the goals of this permit is the control of possible adverse environmental effects of nonferrous metallic-mineral mining and the preservation of natural resources. To accomplish this, it is MDNR policy that mining be conducted in a manner that will reduce impacts to the extent practicable, mitigate unavoidable impacts, and ensure that the mining area is left in a condition that protects natural resources and minimises, to the extent practicable, the need for maintenance. This is to be accomplished through the use of appropriate mining methods and implementing mine-waste management and passive-reclamation procedures that maximise physical, chemical, and biological stabilisation of areas disturbed by mining.

MPCA Water Quality Permits: Industrial wastewater and storm water discharges are regulated under the National Pollutant Discharge Elimination System (NPDES) permit program. The goal of this program is restoring and maintaining the chemical, physical, and biological integrity of the nation's waters so that they can support 'the protection and propagation of fish, shellfish, and wildlife and recreation in and on the water.' In Minnesota, the MPCA administers this program with USEPA oversight. The rules require that all industrial direct discharges from a discernable, confined, and discrete conveyance be allowed only by permit. The permit provides the basis for the implementation and enforcement of surface water quality standards and the level of pollution control for discharges.

MPCA Air Quality Permits There are several air quality permitting programs potentially applicable to this project. MPCA has been delegated authority to administer the federal requirements and has an approved air permitting program. Therefore, MPCA will administer the permit issue process with USEPA oversight.

The general goal of the air permitting program is to ensure that economic growth will occur in harmony with the preservation of areas with clean air and to preserve and protect the air quality in areas of special value (e.g. nearby national parks and wilderness areas). The program is also intended to help ensure compliance with applicable regulations by imposing appropriate monitoring and by requiring regular reports of deviations from permit conditions and compliance certifications.

Section 404 Wetlands Permit: Section 404 requires that the USEPA and the USACE regulate discharges of dredged or fill material to US jurisdictional waters. This includes regulation of impacts to wetlands and requires compensation where impacts are unavoidable. The Minnesota Wetlands Conservation Act (MWCA) also requires that a state permit be issued for disruption of wetlands. The application for the MWCA permit will be prepared as part of the mining and reclamation plan and will be submitted to the MDNR for review and approval. PolyMet will be required to construct, restore, or enhance wetlands as replacement for affected wetlands.

20.8.3 Principal Environmental Issues & their Mitigation

PolyMet has been engaged in ongoing dialog with all of the agencies responsible for permitting the project since January 2004. During this period PolyMet has worked closely with the MDNR, MPCA and USACE to identify potential environmental issues associated with the NorthMet project such that acceptable mitigation strategies can be developed. The issues that have been identified fall into four categories:

- Water Quality;
- Potential for 'Fibres' in Air and Water;
- Wetland Loss and Replacement; and
- Financial Assurance.

Water Quality. Review of the PolyMet project has indicated the potential environmental impacts from the direct discharge of project-related waters to surface water. PolyMet plans to manage water and aqueous effluent flows around the project site to avoid these discharges and the following provides an overview of these plans.

<u>Tailings Basin Discharge</u> - NorthMet ore typically contains sulphides that, when exposed to water and oxygen, can generate acid and dissolved metals. However, the flotation process is designed to maximise recovery of all sulphide minerals. Moreover, gangue minerals have acid buffering capacity and waste characterisation studies show that the flotation tailings will not contain a sufficient mass of sulphides to generate acid or produce concentrations of dissolved metals in excess of surface water quality standards. However, seepage from the existing taconite tailings basin currently does exceed MPCA water quality standards in respect of hardness and specific conductance. To address this PolyMet plans to install a tailings basin seepage-recovery system to reduce seepage losses from the basin. The recovered seepage will be added back into the basin for use as process make-up water. This approach is expected to eliminate the need for an NPDES direct discharge permit, eliminate the existing non-compliance issues, and provide control of any future NorthMet tailings seepage.

Process Residues Discharge - The economic success of the project relies upon the oxidation of sulphides to recover metals. Accordingly, the metallurgical process has been designed to maximise sulphide recovery and allow as little sulphide as possible to escape to residue. In addition, the hydrometallurgical process includes washing and neutralisation of residues to minimise the loss of acid and metals in solution to the residue. Finally, the residue will be placed in engineered and lined disposal cells constructed on the taconite tailings basin. These residue cells will be part of a closed process water circuit which has a negative water balance where supernatant will be recovered from the lined cells and recycled back into the hydrometallurgical process where some fresh make-up water is required. Thus, there will be no direct discharge of the supernatant to surface waters and no need for a NPDES direct discharge permit. When full each cell will be closed, sealed with an impermeable capping layer and re-vegetated to avoid infiltration of meteoric water and snow melt.

Acid Mine Drainage - NorthMet waste rock typically contains sulphides that, when exposed to water and oxygen, may generate acid and may leach metals such as copper and nickel. Accordingly, management procedures will be required not only for waste rock but also for lean ore that may be stockpiled for extended periods and ore which may be stockpiled for only very short periods.

PolyMet has developed a MDNR approved waste characterisation program based on the concept that sulphur content will determine waste rock classification. The program includes kinetic weathering tests (humidity cell tests) on 98 samples representing different waste, ore and lean ore rock types contained within the proposed mining envelope. Stockpile drainage water quality was estimated on the basis of the results of this ongoing program using a characterisation model that includes factors for infiltration rate, the length of the water flow path through a stockpile (i.e. height of the rock column which represents the rock waste stockpile height), the effect of particle size and temperature. Calculations showed that leaching of copper was the most significant parameter relative to the surface water quality standards, and that sulphur content and flow path were important variables. When sulphur content of waste rock was less than 0.12%S or less than 0.31% with a Cu/S ratio of less than 0.3 and the water flow path was less than 30 feet, there was no acid generation and copper and nickel concentrations were below the appropriate surface water quality discharge limit for those metals. At those sulphur contents, nickel concentrations were generally below the surface water standard for all anticipated flow paths.

Based on these calculations, four categories of rock were defined.

- Category 1 – material with less than 0.12%S and material between 0.12% and 0.31% with a Cu/S ratio of less than 0.3 which is non-reactive when water flow path is less than 30 ft.;
- Category 2 – material with less than 0.12%S and material between 0.12% and 0.31% with a Cu/S ratio of less than 0.3 but for which water flow path is greater than 30 ft. This will not generate acid but may leach metals. (Category 1 becomes Category 2 when flow path length exceeds 30 ft.);
- Category 3 (Lean Ore) – material with greater than 0.12%S but less than 1.0%S with a Cu/S ratio of more than 0.3. This material may generate acid and may leach metals; and

- Category 3 – material with greater than 0.31%S but less than 1.0%S with a Cu/S ratio of less than 0.3. This material may generate acid and may leach metals; and
- Category 4 – material with greater than 1.0%S and all Virginia Formation rock which will generate acid and leach metals.

Using this rock classification, PolyMet currently projects that 77.5% of the NorthMet waste rock would be Category 1 and could be used for construction purposes if the height of construction was less than 30 feet. Similarly, this material could be placed in single waste lifts of less than 30 feet without generating acid or drainage water that would require treatment for heavy metals. Other rock categories, which comprise about 22.5% of total mined waste rock, may generate acid rock drainage or concentrations of metals above water quality standards.

PolyMet plans to manage effluent and run-off from waste rock, ore and lean ore by placing them on engineered stockpiles constructed with impermeable bases designed to collect effluent and run-off. When closed, stockpiles will be covered to minimise infiltration of meteoric water. PolyMet plans to tailor stockpile liner and cover system designs to each rock category so as to provide appropriate environmental control. All precipitation run-off water that has contacted ore or reactive waste rock as well as stockpile drainage water and mine pit water will be collected to a central sump prior to being pumped via pipeline to the plant area for use as process make-up water. PolyMet plans to treat this water for metals removal if necessary before adding it to the tailings basin as make-up water. This method of water management effectively eliminates direct discharges of water that has contacted NorthMet waste rock, eliminates the need for a NPDES direct discharge permit, and eliminates surface water quality compliance issues at the mine site.

<u>Great Lakes Initiative - Mercury Standard Compliance</u> - Under a long term, federal program known as the Great Lakes Initiative (GLI) which is intended to improve water quality in the Great Lakes, strict discharge quality standards apply to discharges to surface waters within a watershed draining into any of the Great Lakes. Both the NorthMet mine and Erie Plant are within the Lake Superior watershed. Hence, a discharge water quality standard for mercury of 1.3 parts per trillion of Hg would apply to any discharge from the project site. Further, federal and state regulations prohibit new or expanded discharges, which would cause or contribute to the impairment of surface waters. In the event that a direct discharge was required, the MPCA would regulate this discharge in terms of an NPDES direct discharge permit which requires (a) compliance with the GLI mercury standard at the point of discharge and (b) that the mass of mercury reporting to affected waters (listed as impaired) is less under the proposed discharge than it was prior to the discharge; i.e. that the project would result in a net reduction in the mass of mercury.

To increase the certainty of a positive permitting outcome, PolyMet plans to eliminate the need for a NPDES direct discharge permit as follows;

- Surface run off and water that has come into contact with ore, lean ore and sulphide-bearing waste rock at the mine site will be collected and pumped to a water treatment plant at the Area 2 Shops from where it will be pumped to the tailings basin for use as make up water;
- The metallurgical operations will be managed to ensure they remain net consumers of water, with no discharge requirement;

- Treated mine waste water will be used as process make-up water with the objective of minimising fresh water make up from Colby Lake;
- Plant site run-off will be collected and channelled to the tailings basin; and,
- Advantage will be taken of the demonstrated capacity of taconite tailings to sequester heavy metals and, in particular, mercury to minimize the potential impacts of any seepage from the tailings basin.

Potential for 'Fibres' in Air and Water. In the 1970s, federal and state courts used the potential health risk of 'asbestos-like fibres' in drinking water as the legal means to force a Minnesota taconite company to discontinue the discharge of taconite tailings into Lake Superior. In that case, 'fibres' were defined as particles with a 3-to-1 aspect ratio though there was no standard for such fibres in drinking water at the time. Although the company ultimately lost a long and contentious court case, 'fibres' continue to be an issue.

Taconite mines across the Iron Range extract the Biwabik Iron Formation which contains amphibole minerals. (The Biwabik occurs in the footwall of the NorthMet Deposit and will not be mined). It is now generally agreed that east of a north - south line that runs approximately through the centre of the LTVSMC property those amphibole minerals can exist in fibrous form which, when processed, can form mineral cleavage fragments with an aspect ratio of approximately 3-to-1. In response to concerns about potential impacts, an NPDES permit was applied to LTVSMC that specified a discharge limit for asbestiform fibres consistent with the USEPA's drinking-water standard and which defined asbestiform fibres as a particle that is at least ten microns long. LTVSMC operated successfully under those conditions while the Northshore open pit, which is the source of the ore that caused the concerns in the first place, continues to operate satisfactorily.

During this DFS a number of samples of ore, tailings and process water were collected from NorthMet pilot plant testing and were submitted for fibres analysis. To date, detailed microscopy has not identified any amphibole asbestos in samples collected from the deposit or from the processing of the NorthMet ore; however, short (less than 10 microns) amphibole cleavage fragments were identified in some samples.

While PolyMet does not expect NorthMet ore to generate fibres that could be a concern, PolyMet recognises that providing workers with a safe working environment is a duty of care. As a matter of course, therefore, standard practices and operating procedures will be designed to ensure provision of a safe working environment for employees. These will include;

- Use of air conditioners with fine particulate matter filters on mine and other equipment, and particularly blast hole drill rigs, excavators, haultrucks, locomotives, front end loaders, graders and dozers;
- Dust suppression on haul roads and at material handling and discharge points around the mine;
- High efficiency dust extractors at points throughout the beneficiation plant where materials handling may cause dust;
- Surface treatment and re-vegetation of areas such as the tailings basin where fugitive dust lift-off may occur; and

- Provision to employees of appropriate personal protective equipment.

Wetland Loss and Replacement. The NorthMet mining operation may ultimately impact an area of approximately 3,000 acres of which about 1,200 acres are wetlands. Under state and federal regulations any form of wetlands disturbance must be mitigated in terms of a wetlands replacement plan that must be approved by the USACE and the MDNR as part of the Section 404 and MWCA permitting process.

Every aspect of project design and planning including development of the mine plan and layout of mine site infrastructure has been focussed on minimising wetlands disturbance. In the mine this includes use of in-pit waste disposal where possible and construction of mine site infrastructure and placement of waste rock stockpiles in such a way that wetlands are avoided wherever possible. Precautions will be taken to avoid draining of wetlands into the mine not only to conserve wetlands but also to promote efficient mining operations. Surface runoff and mine water containment systems will be designed and used to avoid uncontrolled releases to the environment in general and to wetlands in particular. Moreover, the ultimate closure plan for the tailings basin and parts of the mine site incorporates practical and realistic opportunities for significant wetlands creation.

Regulations require approval of a wetlands impacts mitigation plan prior to permit issue and PolyMet is working with the USACE, MDNR and other agencies to meet this requirement. While a life of mine mitigation plan is required, the USACE has indicated that it is acceptable to submit a detailed plan for the first 5 years of operations with less detailed but viable mitigation plans for the remainder of mine life. Currently PolyMet is actively pursuing a number of options with a view to firming up an acceptable plan to create or upgrade at least enough wetlands to secure sufficient wetlands credits to cover the area that will be disturbed during the first 5-6 years of production. One of these options includes a plan, agreed to in principle with St. Louis County, for restoring wetlands on state administered tax forfeit lands and sharing the resultant wetlands credits with the County. Other options include acquisition of degraded lands and conversion to wetlands. The total amount of potential wetland credits contained within the various options being pursued are in excess of the expected total life of mine wetlands disturbance.

Financial Assurance. Minnesota Rules require that, in addition to a detailed closure plan, anyone intending to conduct a mining operation must submit, as a part of the application for a permit to mine, a contingency plan and estimate of the costs of closure, reclamation, and post-closure maintenance on the assumption that closure will occur the 'next year.' Financial assurance in the amount equal to the contingency reclamation cost estimate must be provided to the commissioner before a permit to mine can be issued. Once lodged, the financial assurance must be continuously maintained and is adjusted annually to account for new disturbances and progressive reclamation and rehabilitation. Although final terms and conditions have not yet been negotiated, PolyMet plans to provide this assurance by way of an insurance policy for which the initial premium is estimated to be US$10.0M payable prior to issue of the Permit to Mine.

20.8.4 Current Status of Project and Schedules

Environmental review and permitting activities have been in progress since 2004. Table 20-6 provides a listing of baseline and environmental engineering studies that have been prepared or are under preparation to support the EIS and permit applications as of September 2006.

Table 20-6 Baseline Environmental and Environmental Engineering Studies

Baseline Environmental Study	Status
Baseline environmental studies completed or near completion:	
Winter Wildlife & Wildlife Habitat Survey	Completed
Summer Wildlife & Wildlife Survey	Completed
Wetland Delineation and Classification Survey	General Mapping Completed, 5-Year Operating Plan Delineation Completed
Threatened & Endangered Plant Species Surveys	Completed
Canada Lynx Study	Completed
Stream and Wetland Biological Surveys (fish and aquatic marco-invertebrates)	Completed
Stream Classification of Partridge River and Trimble Creek	Completed
Freshwater Mussel Survey in Trimble Creek and Embarrass Rivers	Completed
Soil Mapping	Completed
Background Surface Water Quality Monitoring in Partridge and Embarrass Rivers	Phase I Completed; Phase II ongoing indefinitely
Compilation of Existing Surface Water Quality Data	Completed
Hydrogeologic Investigation for the PolyMet – NorthMet Mine Site	Phase I Completed; Phase II Completed; Phase III to be completed 4th Quarter 2006.
Scoping Cultural Resources Assessment	Completed
Phase I Archaeological Survey	Completed
Wetland Hydrology Study	Work Plan Approved, monitoring initiated summer 2005, ongoing monitoring thereafter
Environmental engineering studies completed and in final implementation phases:	
Flotation Pilot Plant Products Environmental Investigation and Air Testing	Completed
Hydrometallurgical Pilot Plant Products Environmental Investigation and Air Testing	Completed
Air Emission Risk Assessment (AERA) Study	Completed, will be updated in late 2006
Geotechnical Pit Slope Analysis Study	Completed

Baseline Environmental Study	Status
Tailings Basin Geotechnical Study	Completed
Tailings Basin Water Balance Modeling Study	Completed
Mine Site Water Balance Modeling Study	Pending, to be completed 1st Quarter 2007
Mine Site Storm Water Management Plan	Pending, to be completed 1st Quarter 2007
Pilot Plant Process Optimisation Environmental Testing Study	Completed
Mine Waste Management Plan & Stockpile Design	Pending, to be completed by 4th Quarter 2006
Wetlands Mitigation Plan	Several options identified and being assessed, draft plan being revised, finalisation pending completion of regulatory review
Mine Closure Plan	Pending, to be completed 1st Quarter 2007
Mine Waste Characterisation Studies	Ongoing, preliminary report completed, final report pending , to be completed 4th Quarter 2006
Mine Wastewater Characterisation Studies	Pending, to be completed by 4th Quarter 2006
Mine Water Treatment Studies	Ongoing, to be completed by 4th Quarter 2006
Process Waste Characterisation Studies	Ongoing, preliminary report completed, final report pending, to be completed by 4th Quarter 2006
Process Wastewater Characterisation Study	Ongoing, to be completed 4th Quarter 2006

20.9 Capital Cost Estimate

Capital cost estimates were generated to an overall level of accuracy of -5% to +15% in order to provide a confident basis for project financing decisions. The following section summarises the basis and methodology for developing capital cost estimates to the required level of accuracy and confidence. Capital cost estimates are prepared with an April 2006 cost base without application of escalation and exclude Minnesota state sales tax.

20.9.1 Basis of Capital Cost Estimate

The capital cost estimate was developed on the basis of frozen design criteria and flowsheets and includes an initial and sustaining life of mine capital schedule. Components of the capital cost include:

- Initial capital is that required during the pre-production construction period necessary to bring the operation into production and includes EPCM, owners costs, first fills, insurance, and commissioning costs;
- Sustaining capital includes replacement of capital plant and equipment and expansion or extension of facilities required to maintain operations e.g. progressive construction of additional

hydrometallurgical residue cells, major rail replacement programs, extension of the impermeable base of waste rock stockpiles etc.

The capital estimate is broken down by facilities, equipment items, freight, direct labour, construction, contractors costs, and spares. Most of the equipment, services and materials will be sourced within the USA and therefore foreign exchange rate variations are unlikely to be significant.

Contingency was assessed by Bateman using a sophisticated Monte Carlo risk assessment method that analysed key areas of the cost estimate separately and allocated contingency according to assessed risk and commensurate with estimate accuracy.

State sales tax was excluded on the assumption that it would be recoverable.

The following summarises the basis on which the major components of the capital cost estimate were prepared.

- Mine Pre-production Costs: An estimate was developed by PolyMet from written quotes from four prospective mining contractors. Pre-production mining costs included mobilisation, preparation of site access and construction of initial haul roads, pre-stripping and initial waste removal in preparation for ramping-up to full mill production during year 1. Material movement quantities were based on a production schedule developed by AMDAD.
- Waste Rock Stockpile Construction: In the absence of close spaced overburden drilling and sampling, excavation and fill volumes were estimated from an overburden thickness model based on drill hole logs, geophysical soundings and a limited number of test pits which provided the basis for assumptions relating to soil types and characterisation. For environmental reasons waste rock stockpiles are required to be constructed with impermeable bases the construction costs of which were estimated from a combination of local contract earthmoving rates and recent project experience elsewhere.
- Mine Power Supply: For costing purposes it was assumed the power utility will provide at no cost the tap and connection to 138kV transmission line and the main mine site step down transformer. The cost of constructing and periodically extending the 4,160v mine site wooden pole mounted, power reticulation line was based on a written quote from local power utility, Minnesota Power.
- Railroad: Railroad costs were estimated by Duluth-based KOA who specialise in railroad engineering and, therefore, were able to call upon reliable, recent local costs of services, construction and materials (rail, ties etc.) Refurbishment costs for existing track were based on a detailed survey of its condition using recent local rates for similar work elsewhere.
- Rail Transfer Hopper: Design by KOA was closely based on two approximately similar loading hoppers built for LTVSMC in the mid- to late- 1990's. Current Iron Range construction labour rates were used with materials costs estimated against an engineered materials take-off. Costs for overhauling and refurbishing salvaged mechanical and hydraulic equipment were provided by original equipment manufacturers.
- Mine and Railroad Infrastructure: Refurbishment costs were based on preliminary architectural and engineering drawings with application of standard unit rates for refurbishment of offices, change houses and personnel facilities. Reactivation costs of Area 1 (mine equipment) and Area

2 Shops (railroad rolling stock maintenance) workshops were estimated from a combination of vendor/supplier quotes, allowances and standard rates for similar work elsewhere.

- Mine to Waste Water Treatment Plant (WWTP) Pipeline: Capital cost was developed from a quote for spiral-wound, steel pipe laid above ground with a factored allowance for installation. Costs for refurbishing existing pumps were supplied by a pump vendor.
- HV Electrical Sub-Station: Although parts of the sub-station remained active since closure, re-activation costs were based on LTVSMC operating and maintenance records, inspections by heavy current electrical contractors and engineers of the local electrical supply utility, Minnesota Power.
- Ore Beneficiation Plant: Reactivation costs were based on;
    - Detailed plant condition surveys;
    - Assessment of operating and maintenance records to determine remaining life in crusher and mill wear materials and liners;
    - Vendor assessment of process control system hardware and I/O points;
    - Vendor quotes for dust extraction system equipment;
    - Vendor quotes for flotation equipment; and
    - Test starting of selected, representative electric motors to confirm re-start and start-up failure assumptions.
- Hydrometallurgical Plant: Table 20-7 below summarises the basis of new plant capital cost estimates.

Table 20-7 Basis of New Plant DFS Capital Estimates

Process	**Requirements**
Process flowsheets	Optimised
Bench scale tests	Essential
Pilot scale tests	Recommended and completed
Energy and material balances	Optimised
Equipment List	Finalised
Facilities Design	
Plant capacity	Optimised
Equipment selection	Optimised
General arrangements - mechanical	Preliminary
General arrangements - structural	Preliminary
General arrangements - other	Outline
Piping	Based on single line drawings
Electrical	Based on single line drawings

Process	Requirements
Specifications	General
Basis for Capital Cost Estimate	
Vendor quotations	Multiple, preferably written
Civils	Derived from drawings
Mechanical and piping	Approximate quantities
Structural work	Derived from material take-off
Instrumentation	Derived from material take-off
Electrical work	Derived from material take-off
Indirect costs	Calculated
Project program/schedule	Critical path network
Expected contingency range	10-15%

- Flotation Tailings Basin – Seepage Recovery System Upgrade: Capital estimate was developed by Barr Engineering and based on recent, similar project experience and standard unit costs for pipe and earthworks. Tailings piping will consist of a combination of new and salvaged steel pipe and refurbishment costs of existing tailings pumps were provided by a local pump vendor.
- Hydromet Residue Cells: Excavation costs were developed by PolyMet from local earthmoving contractor unit rates with liner acquisition and placement costs derived from recent, local experience of constructing land fill and taconite tailings disposal facilities.
- Limestone Stockpiling and Handling System: cost based on preliminary engineering and materials take-off. Allowances for re-use of some components were also included.
- Fresh Water Reticulation System: Costs were based on field examination and engineered estimates of refurbishment requirements. In the case of the fresh water pipeline from Colby Lake, historical maintenance records were used to estimate the amount of plastic, internal re-sleeving required to return the pipeline to operable condition.
- Plant Site Infrastructure: Costs were based on field inspections and an assessment of historical maintenance and operating records. Where equipment or component refurbishment or replacement was necessary costs were derived from vendor and original equipment manufacturers quotes.

The capital cost schedule contains estimates for all environmental aspects of the study that resulted from technical evaluations and studies undertaken by Barr Engineering, SRK, Golder Associates and others.

The overall estimated initial and sustaining capital cost for developing the project is shown in Table 20-8 below. Costs are estimated at a base date of April 2006 and exclude escalation. Equipment import duties, freight and insurance are included where appropriate but state sales tax (at 6%), which is recoverable, is excluded.

Table 20-8 Summary of Initial & Sustaining Capital Costs

	US$'000	
	Initial	**Sustaining**
Direct Costs		
Mining & Mine site Infrastructure	18,489	24,354
Railroad	8,464	33,344
Erie Plant Beneficiation Plant	62,992	0
Hydrometallurgical Process Plant	191,996	3,170
Tailings & Residue Disposal	3,134	7,949
Total Direct Costs	**285,075**	**68,817**
Contingency	27,070	
Indirect Costs	67,495	2,970
Total Project Capital	**379,640**	**71,787**

20.10 Operating Cost Estimates

20.10.1 Operating Expenditure

A steady state operating cost model has been developed to an accuracy of –5% to +15% and includes manpower resources, power consumption, reagents, consumables and maintenance material and spare parts and contract services. Once the operation reaches steady state production the majority of operating costs will be fixed with only mining, rail ore haulage and ore beneficiation costs being significantly tonnage variable. As with capital, costs are estimated with a base date of April 2006 and exclude escalation. Table 20-9 below shows the estimated annual operating cost averaged over the first 10 years of full rate, steady state operation starting in year 2. Table 20-10 below at the end of this section shows the annual life of mine operating cost schedule.

Table 20-9 Average Annual Operating Cost Summary – Years 2 – 11 inclusive.

Cost Element	Annual Cost US$ ('000)	Average unit cost US$/ton milled
Mining & Mining Infrastructure	38,039	3.26
Railroad Operations	1,656	0.14
Metallurgical Operations	56,274	4.82
Technical Support, Management, G&A	5,927	0.51
Power	26,752	2.29
Total	**128,648**	**11.02**

20.10.2 Basis of Operating Cost Estimation

Control Philosophy

Control philosophy will determine operating manpower requirements, particularly in the metallurgical plant.

The mining and rail ore haulage operations will be managed as an integrated system by means of a state of the art GPS-based dispatch system interfacing with the beneficiation plant control system at the primary crusher. The dispatch system will incorporate real time monitoring of loading and truck hauling as well as destination tracking of ore and waste to ensure waste rock is correctly placed on the designated waste rock stockpile and all ore goes to the rail transfer hopper or an ore stockpile. Mine equipment will be fitted with state of the art condition monitoring equipment to facilitate maintenance and maximise availability. GPS-based surveying and equipment positioning systems will minimise the need for field technicians (surveyors and grade control technicians) to walk around the open pit where they would be exposed to weather, dust and heavy equipment movements.

Process control of the beneficiation and hydrometallurgical plants is via a state of the art distributed control system (DCS) designed to monitor and sequence all functionality. Four control room operators (CRO) per continuous shift will have full control over the entire plant including access to process configuration, motor stopping and starting, alarm handling and emergency shutdown (ESD) management via a personal computer interface. The CROs will also interface with the mine-railroad dispatch system. A high degree of automation is designed into the overall process to limit personnel numbers. Automation includes valve actuation, process instrumentation and remote operation of equipment from the control room.

Equipment functionality is supervised by the DCS via sequence blocks. Interlock logic is set up to prevent improper operation and preserve equipment in the event of process upsets. The plant is designed to fail safe and an ESD management system is included to protect the plant in case of catastrophic failure of equipment, fire or loss of power. The plant can be restarted using the automated sequences in the DCS.

Field operator intervention will be required from time-to-time and the operating expenditure model allows for a “roving” crew to monitor equipment status, perform routine checks, take samples for analysis, report on maintenance requirements and maintain housekeeping to a high standard. Similarly, a small permanent crew of maintenance technicians including fitters, boilermakers and instrument electricians will support operations personnel on a twenty four hour basis.

Plant information is captured and stored in the DCS and archived by data historian software to facilitate metallurgical accounting and preparation of routine metallurgical reporting. Configurable process trends are displayed in real time to allow the CRO to monitor and manipulate conditions to achieve safe production targets.

Manpower Resources

Manpower requirements are based on an organisational structure developed by Bateman and PolyMet for steady state operating conditions. The organisation includes mine, railroad and metallurgical plant operating manpower, general management, technical services, administration and accounting, support services (safety, environmental and human resources), purchasing, warehousing and product marketing. The basis for the human resource cost estimate is the following;

- Mining and mineral processing operations will operate 365 days per year, 24 hours per day with three 8-hour shifts. Operations supervision and essential support services will be provided round the clock on a continuous basis with technical and general support, and general management services operating on day shift only Monday to Friday, excluding statutory holidays. Laboratory services will be provided on a continuous basis. There is no presumption as to the use of union or non-union labour.

- Operating labour costs are based on a current taconite industry labour agreement;
- Management, supervisory, technical and non-operating support staff costs were derived from current rates applicable at a number of other local taconite producers; and,
- Labour burden and on-costs are based on labour legislation and current practice at other taconite producers.

Including loading for leave, absenteeism and training, total management, operating, maintenance and support personnel numbers will be in the range 400-450, excluding out-sourced major equipment maintenance contractors.

Prospective mining contractors were reluctant to provide manpower estimates and for costing purposes PolyMet developed an assumed mine manning structure based on the assumed equipment fleet and including supervisors, equipment operators and maintenance personnel. Mine planning, geology, grade control, dispatch system technicians and mine management are considered owners staff.

Power Consumption

Electrical power consumption was determined for each motor identified on equipment lists, using vendor information and industry typical parameters for load factors, motor efficiencies and plant availability. Equipment utilisation was based on design parameters. Where not specified by equipment vendors and manufacturers, motor sizes were selected using engineering calculations and by using industry standards for items such as feeders, flotation agitators, tank agitators, fans and blowers.

Electrical energy will be supplied by local, regulated electrical power supplier, Minnesota Power (MP), and imported from the existing grid at a cost of US$0.045 /kWh. This tariff is similar to that of other large power customers in the region and excludes a package of up to 25MW of low cost power offered at incentive rates and for which PolyMet is potentially eligible.

Reagents and Consumables

Reagent and consumable unit costs are based, in most cases, on written quotes which generally included transportation and shipping. Some suppliers of process reagents offered to provide or construct on-site storage facilities (tanks) at their initial cost with a capital recovery component added to the unit reagent cost. Where economically justified, this option was adopted as a means of minimising initial capital cost.

In the case of diesel fuel oil and explosives, it is common practice on the taconite mines for the vendor to be responsible for storage and distribution to the point of end use. For costing purposes, it was assumed the fuel supplier will provide, man and operate in-field refuelling equipment and facilities. A unit rate of US$2.45/gallon delivered was used which includes an assumed vendors premium of US$0.05/gal for storage, delivery and field re-fuelling services. Fuel oil consumption by mobile equipment was estimated on the basis of manufacturer’s specific fuel consumption rates and estimated engine running hours.

Similarly, for costing purposes, it was assumed the explosives vendor will be responsible for transporting explosives and blasting accessories to the mine and placing them in blast holes under PolyMet supervision. This fully integrated "in hole" service has the added advantage that no on-site explosives storage is necessary as the vendor is entirely responsible for explosives storage in off-site magazines. Explosives volumes were estimated on the basis of a blasting study carried out by Golder Associates and which included definition of ore and waste drilling patterns and calculation of blasting powder factors from standard laboratory tests of drill core samples.

Reagent and oxygen consumption rates were determined from Metsim modelling and were optimised during the various pilot-scale test programs carried out at Lakefield. Wear materials and grinding media consumption rates were estimated from Bond work and abrasion indices calculated from standard laboratory tests of NorthMet material derived from drilling.

Maintenance Spares and Materials

Provision of maintenance spares and materials will be governed by PolyMet's operating philosophy which requires PolyMet personnel to carry out routine preventative and day to day plant maintenance at site with all major maintenance, overhaul and repair work out-sourced to the several providers of maintenance services that currently serve the taconite mines.

<u>Mine equipment</u>: maintenance requirements were included in the contract mining unit cost rates and are not separately identified.

<u>Railroad equipment</u>: locomotives will be leased and typically lease fees are structured to include preventative maintenance costs but not major component overhaul or re-builds. PolyMet is assumed to carry the cost of routine service and maintenance and, of course, fuels and lubricants. The condition of each item of rolling stock acquired from Cliffs Erie was determined by field examination by KOA. KOA is a Duluth-based engineering company which, among other things, specialises in providing railroad engineering services to most of the taconite producers and railroad operations in the area. Consequently KOA has extensive experience and an up to date database of railroad operating information which together with historical LTVSMC maintenance records was used to estimate rolling stock and track maintenance requirements.

<u>Beneficiation Plant</u>: maintenance requirements of the crushing and milling sections were largely based on historical information (LTVSMC maintenance records and information from former employees). Where appropriate, adjustments were made for differences in the characteristics of taconite and NorthMet ore. Maintenance requirements for the new flotation section were based on a combination of vendor recommendations and typical requirements for flotation plant elsewhere. Tailings slurry pump maintenance requirements were based on LTVSMC historical performance with adjustments for the less abrasive nature of NorthMet ore and gangue minerals. Development of beneficiation plant maintenance spares and materials requirements took into consideration the significant amount of redundant plant and equipment which will be available for refurbishment and reactivation at minimal cost in the event that the originally selected item fails to re-start or breaks down prematurely.

<u>Hydrometallurgical plant</u>: as with the other parts of the plant, a preventative maintenance philosophy will be adopted to maximise availability. This will be particularly important for the autoclaves and associated equipment and in areas where conditions will be corrosive or chemically aggressive. Maintenance costs were initially developed from a maintenance schedule developed by Bateman which

included provision for periodic closure for statutory inspection of pressure vessels. Bateman's maintenance schedule was drawn largely from its own construction and operating experience and was based on the performance and maintenance history of hydrometallurgical plants elsewhere. To avoid possible cost anomalies, the ratio of maintenance costs derived from this schedule and direct equipment capital costs was also benchmarked against similar ratios at a variety of recently constructed process plants elsewhere. Copper solvent extraction and electrowinning technology is well known and widely used and because PolyMet's plant will be similar to many other SX-EW plants, maintenance requirements were reliably determined on the basis of benchmarking against known plant performance elsewhere.

20.10.3 Life of Mine Operating Cost Schedule

Figure 20-11 which appears at the end of this section shows the life of mine operating cost schedule. Increases in mining costs over time as a result of progressive deepening of the open pit will be, to some extent, offset from Year 11 onwards by a reduction in some haul distances as the East Pit is backfilled with waste from the West Pit. Unit costs will remain relatively constant in all other cost centres as these are either fixed or constant relative to process plant throughput.

20.10.4 Operating Cost Summary Description

The following section summarises the composition of the major operating cost components.

<u>Mining</u>

Four prospective mining contractors; two local and two larger, integrated organisations with operations elsewhere in North America provided written quotes in response to an enquiry and detailed scope of work prepared by jointly by PolyMet and AMDAD. For purposes of their estimates, contractors were asked to assume that;

- excavators, blast hole drill rigs and main pit pumps will be electrical and that power will be supplied by PolyMet at no cost to contractor;
- explosives would be provided, delivered to site and placed in blast holes by the local vendor. (This type of "in hole" service arrangement is used by several local taconite operations);
- fuel oil will be supplied and delivered to the point of use by the local fuel oil vendor at an assumed premium (US$0.05/gal), which includes manning a field re-fuelling station and in-field re-fuelling of equipment such as dozers and diesel pumps;
- the mining contractor will be responsible for equipment maintenance for which purpose PolyMet will make available the Area 1 Mine Equipment workshop facility which is located to the west of the Erie Plant.

Contractors were also requested to provide estimates for "wet" hire of mobile equipment for unspecified day works. In addition, contractors were also requested to provide estimates for pre-split drilling and smooth wall blasting though, based on a preliminary rock slope assessment by Golder Associates, these were not used for the operating cost estimate.

Annual operating costs were estimated by putting unit rates provided by the prospective mining contractors against the production schedule prepared by AMDAD. The production schedule included estimates of overburden and till removal and took into consideration variations over time of haul distances to waste rock stockpiles, the ore stockpile and the rail transfer hopper. It was assumed that PolyMet personnel will carry out grade control using a dedicated reverse circulation (RC) drilling rig with sample analysis in PolyMet's plant site laboratory. RC drilling costs are based on escalated historical costs and typical actual costs elsewhere from AMDAD's data base.

The costs of crushing and screening waste rock for blast hole stemming and for road construction and gritting during winter were based on a quote from a local contractor for mobilisation and operation of a demountable crushing and screening plant. Crushing and screening operations will carried out annually on a campaign basis to build up a stockpile of roadstone and aggregate for use throughout the year.

Railroad Operations

Railroad operations were assumed to be carried out by PolyMet personnel using leased locomotives and rolling stock (100 ton capacity side dumping cars) acquired from Cliffs Erie. As indicated above, rolling stock and track maintenance costs were estimated by railroad specialists from KOA. For cost purposes, it was assumed that specialist track maintenance work would be outsourced to one of several local specialist service providers although these costs were estimated on the basis of experience at other railroad operations and were not supported by specific quotes.

Fuel oil and lubricants consumption rates were also calculated by KOA from specific consumption rates provided by equipment manufacturers and cross checked against typical performance statistics from other operations using similar locomotives.

Mine Site Infrastructure

- Area 1 Truck Shop. Routine mine mobile equipment maintenance work will be carried out at the refurbished Area 1 Truck Shop facility by the mining contractor or a sub-contractor. Major repair and overhaul work will be outsourced. Operating costs for this area shown in the operating cost schedule apply only to maintenance of the buildings and their associated infrastructure and for the cost of lighting, heating and power for tools. All tools and maintenance equipment will be provided by the mining contractor or a sub-contractor.

- Area 2 Offices & Rail Car Maintenance Facility. Operating costs for this area represent an allowance for the cost of building maintenance, power and services.

- Field re-fuelling & Service Bay. PolyMet has acquired two steel structures from Cliffs Erie which were used by LTVSMC for re-fuelling and daily servicing of mine mobile equipment and haul trucks. After re-erection of one of these refurbished structures at the NorthMet mine site, an operating cost provision was made for maintenance of the structure and for lighting.

Metallurgical Operations

PolyMet personnel will operate and maintain both the Beneficiation and the Hydrometallurgical plants along with their associated infrastructure and flotation tailings and hydromet residue disposal. Manning of these facilities is based on the control philosophy described at the beginning of this section. Routine maintenance and repair work will be carried out by PolyMet personnel with major scheduled maintenance work outsourced.

The basis of estimation of reagents, consumables and maintenance materials costs is described above.

The POX process is a major consumer of oxygen which will be provided by way of an “over-the-fence” supply agreement from an on-site cryogenic plant. Negotiations with potential suppliers are ongoing and hence commercially sensitive; however, for cost estimation purposes it was assumed that an oxygen production facility will be constructed and operated on the Erie Plant site by others, independent of the PolyMet operation. Part of the construction cost will be covered as an initial capital charge and subsequent capital cost recovery by means of a surcharge on the unit cost of oxygen supplied. No offset has been included for sale to third parties of surplus nitrogen and/or argon though this remains a possibility.

Analysis of mine drilling and process control samples is an essential part of an efficient operation and accordingly an operating cost provision is included for laboratory consumables. The cost of laboratory staff is included in Operations and Maintenance Staff.

PolyMet Supervisory and Technical Staff

Costs in this area are based on PolyMet’s organisational structure and includes all the management, technical support, general support, administration and engineering functions not included in mining and metallurgical operations. The remuneration and employment burden rates used are based on current local practice and are discussed in more detail above.

Power Consumption

Estimates of power cost by area are based on a combination of electrical equipment lists, engineering calculations, manufacturers’ data, industry standards, assumed utilisation and target availabilities.

Table 20-10 Life of Mine Operating Cost Schedule

Year	0	1	2	3	4	5	6	7	8	9	10	11
Mining Operations		24,862,547	33,802,911	38,986,510	39,660,582	36,834,479	34,807,617	36,819,471	37,308,943	39,248,330	41,298,870	37,147,324
Load and Haul Till		2,022,740	30,399	712,963	1,779,648	2,330,321	1,954,155	611,801	0	0	0	2,287,037
Grade Control		129,962	239,900	283,573	212,288	252,765	252,976	253,461	197,299	223,242	279,551	223,938
Drill Ore		701,792	1,295,462	1,531,292	1,146,355	1,364,933	1,366,071	1,368,688	1,065,414	1,205,509	1,509,576	1,209,265
Blast Ore		779,769	1,439,402	1,701,435	1,273,728	1,516,592	1,517,856	1,520,765	1,183,794	1,339,455	1,677,306	1,343,627
Drill Waste		1,704,000	2,174,019	2,247,774	2,428,381	2,036,548	1,948,133	2,199,889	2,519,827	2,509,212	2,216,407	2,019,083
Blast Waste		1,441,846	1,839,555	1,901,963	2,054,784	1,723,233	1,648,420	1,861,445	2,132,161	2,123,179	1,875,421	1,708,454
Load and Haul Ore		5,117,551	9,541,483	12,423,607	9,929,262	10,665,211	10,761,478	11,483,121	8,964,825	9,773,780	13,458,000	10,556,051
Load and Haul Waste		12,743,824	17,042,691	17,983,903	20,091,413	16,744,876	15,158,527	17,320,301	20,235,029	21,603,841	20,082,608	17,348,209
Stockpile Reclaim		21,062	0	0	544,722	0	0	0	810,594	270,112	0	251,660
Dayworks		200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000
Mine Area		204,700	391,300	387,900	391,500	391,500	391,500	391,500	391,500	391,500	391,500	391,500
Aggregate crush & screening - Contractor mobilization		8,600	8,600	8,600	8,600	8,600	8,600	8,600	8,600	8,600	8,600	8,600
Drill stemming		11,800	14,100	10,700	14,300	14,300	14,300	14,300	14,300	14,300	14,300	14,300
Road gritting (winter)		184,300	368,600	368,600	368,600	368,600	368,600	368,600	368,600	368,600	368,600	368,600
			0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03
Railroad operation		977,400	1,656,000	1,656,000	1,656,000	1,656,000	1,656,000	1,656,000	1,656,000	1,656,000	1,656,000	1,656,000
Fuel & lubes		839,400	1,399,000	1,399,000	1,399,000	1,399,000	1,399,000	1,399,000	1,399,000	1,399,000	1,399,000	1,399,000
Materials		38,000	76,000	76,000	76,000	76,000	76,000	76,000	76,000	76,000	76,000	76,000
Track maintenance - consumables		20,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000
Track maintenance - out-sourced services		36,000	55,000	55,000	55,000	55,000	55,000	55,000	55,000	55,000	55,000	55,000
Rolling stock - maintenance & parts		20,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000
Loading pocket maintenance, parts, lubricants, consumables		24,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000
Mine Site Infrastructure		56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000
Area 1 Truck Shop - maintenance		20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000
Area 2 Offices & Maintenance facility		24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000
Field re-fuelling & service bay		12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000
Metallurgical Operations	2,482,000	33,782,000	56,274,000	56,274,000	56,274,000	56,274,000	56,274,000	56,274,000	56,274,000	56,274,000	56,274,000	56,274,000
Operations and Maintenance Staff	2,482,000	12,320,000	13,129,000	13,129,000	13,129,000	13,129,000	13,129,000	13,129,000	13,129,000	13,129,000	13,129,000	13,129,000
Contract Services	0	0	3,148,000	3,148,000	3,148,000	3,148,000	3,148,000	3,148,000	3,148,000	3,148,000	3,148,000	3,148,000
Reagents & Consumables	0	20,647,000	32,123,000	32,123,000	32,123,000	32,123,000	32,123,000	32,123,000	32,123,000	32,123,000	32,123,000	32,123,000
Maintenance materials	0	538,000	7,410,000	7,410,000	7,410,000	7,410,000	7,410,000	7,410,000	7,410,000	7,410,000	7,410,000	7,410,000
Assay laboratory - consumables	0	277,000	464,000	464,000	464,000	464,000	464,000	464,000	464,000	464,000	464,000	464,000
PolyMet Supervisory & Technical Staff		5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300
PolyMet - Management & Admin		2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900
PolyMet - Mine & Railroad Supervision, technical & labor		3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400
Power Consumption		24,137,800	26,752,200	26,752,200	26,752,200	26,752,200	26,752,200	26,752,200	26,752,200	26,752,200	26,752,200	26,752,200
Mining equipment		505,100	841,900	841,900	841,900	841,900	841,900	841,900	841,900	841,900	841,900	841,900
Mine site including pit dewatering		57,400	95,600	95,600	95,600	95,600	95,600	95,600	95,600	95,600	95,600	95,600
Rail Loading Hopper		85,000	141,700	141,700	141,700	141,700	141,700	141,700	141,700	141,700	141,700	141,700
Waste water pumping - mine to WWTP		206,400	344,100	344,100	344,100	344,100	344,100	344,100	344,100	344,100	344,100	344,100
Process Plant (Ore beneficiation & Hydromet)	1,000	23,131,000	25,074,000	25,074,000	25,074,000	25,074,000	25,074,000	25,074,000	25,074,000	25,074,000	25,074,000	25,074,000
Area 1 Shop		92,200	153,700	153,700	153,700	153,700	153,700	153,700	153,700	153,700	153,700	153,700
Area 2 Shop & offices		30,700	51,200	51,200	51,200	51,200	51,200	51,200	51,200	51,200	51,200	51,200
Waste water treatment plant (WWTP)		30,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000
TOTAL (US$)	2,482,500	89,947,747	124,859,711	130,039,910	130,717,582	127,891,479	125,864,617	127,876,471	128,365,943	130,305,330	132,355,870	128,204,324

Year	12	13	14	15	16	17	18	19	20	21	22
Mining Operations	38,985,679	36,395,613	33,861,236	35,437,769	37,614,421	40,705,356	34,164,505	37,139,910	34,425,755	35,338,373	6,748,038
Load and Haul Till	0	0	2,747,669	48,628	0	0	0	0	0	0	0
Grade Control	258,061	249,221	224,318	138,317	231,306	319,376	235,257	236,543	146,940	252,461	71,021
Drill Ore	1,393,530	1,345,796	1,211,316	746,913	1,249,051	1,724,628	1,270,388	1,277,330	793,478	1,363,291	383,514
Blast Ore	1,548,367	1,495,328	1,345,906	829,904	1,387,834	1,916,253	1,411,542	1,419,256	881,642	1,514,768	426,127
Drill Waste	2,321,855	1,941,695	1,570,267	2,732,782	2,249,897	1,791,934	1,583,285	1,645,400	1,853,212	1,086,243	78,619
Blast Waste	1,964,646	1,642,972	1,328,688	2,312,354	1,903,759	1,516,252	1,339,702	1,392,262	1,568,103	919,128	66,524
Load and Haul Ore	11,265,308	11,497,109	10,894,018	6,214,988	10,679,575	15,429,131	12,339,708	12,996,364	8,129,776	15,398,092	4,614,611
Load and Haul Waste	20,033,911	18,023,492	14,155,334	22,203,383	19,712,998	16,886,932	15,784,622	17,292,354	20,852,604	13,218,390	907,622
Stockpile Reclaim	0	0	183,721	10,500	0	920,850	0	680,400	0	1,386,000	0
Dayworks	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000
Mine Area	391,500	391,500	391,500	391,500	391,500	391,500	391,500	391,500	391,500	391,500	0
Aggregate crush & screening - Contractor mobilization	8,600	8,600	8,600	8,600	8,600	8,600	8,600	8,600	8,600	8,600	0
Drill stemming	14,300	14,300	14,300	14,300	14,300	14,300	14,300	14,300	14,300	14,300	0
Road gritting (winter)	368,600	368,600	368,600	368,600	368,600	368,600	368,600	368,600	368,600	368,600	0
	0.03	0.03	0.03								
Railroad operation	1,656,000	1,656,000	1,656,000	1,324,800	1,324,800	1,324,800	1,324,800	1,159,200	1,159,200	1,159,200	993,600
Fuel & lubes	1,399,000	1,399,000	1,399,000	1,119,200	1,119,200	1,119,200	1,119,200	979,300	979,300	979,300	839,400
Materials	76,000	76,000	76,000	60,800	60,800	60,800	60,800	53,200	53,200	53,200	45,600
Track maintenance - consumables	60,000	60,000	60,000	48,000	48,000	48,000	48,000	42,000	42,000	42,000	36,000
Track maintenance - out-sourced services	55,000	55,000	55,000	44,000	44,000	44,000	44,000	38,500	38,500	38,500	33,000
Rolling stock - maintenance & parts	30,000	30,000	30,000	24,000	24,000	24,000	24,000	21,000	21,000	21,000	18,000
Loading pocket maintenance, parts, lubricants, consumables	36,000	36,000	36,000	28,800	28,800	28,800	28,800	25,200	25,200	25,200	21,600
Mine Site Infrastructure	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000	56,000
Area 1 Truck Shop - maintenance	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000
Area 2 Offices & Maintenance facility	24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000	24,000
Field re-fuelling & service bay	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000
Metallurgical Operations	56,274,000	56,274,000	56,274,000	45,019,200	45,019,200	45,019,200	45,019,200	39,391,800	39,391,800	39,391,800	33,764,400
Operations and Maintenance Staff	13,129,000	13,129,000	13,129,000	10,503,200	10,503,200	10,503,200	10,503,200	9,190,300	9,190,300	9,190,300	7,877,400
Contract Services	3,148,000	3,148,000	3,148,000	2,518,400	2,518,400	2,518,400	2,518,400	2,203,600	2,203,600	2,203,600	1,888,800
Reagents & Consumables	32,123,000	32,123,000	32,123,000	25,698,400	25,698,400	25,698,400	25,698,400	22,486,100	22,486,100	22,486,100	19,273,800
Maintenance materials	7,410,000	7,410,000	7,410,000	5,928,000	5,928,000	5,928,000	5,928,000	5,187,000	5,187,000	5,187,000	4,446,000
Assay laboratory - consumables	464,000	464,000	464,000	371,200	371,200	371,200	371,200	324,800	324,800	324,800	278,400
PolyMet Supervisory & Technical Staff	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300	5,927,300
PolyMet - Management & Admin	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900	2,586,900
PolyMet - Mine & Railroad Supervision, technical & labor	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400	3,340,400
Power Consumption	26,752,200	26,752,200	26,752,200	21,709,060	21,709,060	21,709,060	21,709,060	19,187,490	19,187,490	19,187,490	16,680,090
Mining equipment	841,900	841,900	841,900	841,900	841,900	841,900	841,900	841,900	841,900	841,900	841,900
Mine site including pit dewatering	95,600	95,600	95,600	95,600	95,600	95,600	95,600	95,600	95,600	95,600	95,600
Rail Loading Hopper	141,700	141,700	141,700	113,360	113,360	113,360	113,360	99,190	99,190	99,190	99,190
Waste water pumping - mine to WWTP	344,100	344,100	344,100	344,100	344,100	344,100	344,100	344,100	344,100	344,100	344,100
Process Plant (Ore beneficiation & Hydromet)	25,074,000	25,074,000	25,074,000	20,059,200	20,059,200	20,059,200	20,059,200	17,551,800	17,551,800	17,551,800	15,044,400
Area 1 Shop	153,700	153,700	153,700	153,700	153,700	153,700	153,700	153,700	153,700	153,700	153,700
Area 2 Shop & offices	51,200	51,200	51,200	51,200	51,200	51,200	51,200	51,200	51,200	51,200	51,200
Waste water treatment plant (WWTP)	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000
TOTAL (US$)	130,042,680	127,452,613	124,918,236	109,865,629	112,042,281	115,133,216	108,592,365	103,253,200	100,539,045	101,451,663	64,169,428

20.11 Project Implementation

20.11.1 Introduction

The Implementation Plan details all of the activities that are on the critical path and the manner in which these will be executed ahead of the critical dates, and addresses the sourcing and organization of all of the resources needed for implementation of the project. This covers the Owner's team, engineering and construction contractors and vendors. The Implementation Plan also takes account of the following strategic considerations:

- Provision by Bateman of process performance guarantees.
- Project implementation timing which will be governed by completion of the EIS and the environmental permitting process since no construction activities may begin before the appropriate permits have been issued.
- Identification of long lead items and their impact on procurement effort and the construction schedule.
- The autoclaves are among the longest lead items for procurement (up to 12 months from the completion of shop fabrication drawings). For US based suppliers, purchase orders would have to be placed by the end of the first quarter of 2007. Preceding activities to placing the purchase order are all on the critical path and will be commenced in late 2006. A similar approach will be taken on the other long lead items.
- Project scheduling requires that local climatic conditions are taken into consideration. Winters are severe and extreme cold requires appropriate planning provisions are made; namely,
    - The closure of the Great Lakes shipping season from early December to about April. This affects equipment deliveries and bulk consumables such as limestone.
    - While road and rail transport operate throughout the winter, the Spring thaw impacts both (but particularly road transport) when axle weight restrictions are applied. Road weight restrictions generally extend between March and April though duration is variable and dependent on local conditions.
    - In this part of Minnesota in winter, extreme cold and snow make construction significantly more difficult and expensive than in summer. For this reason the implementation schedule aims to have the major new buildings completed before winter to enable equipment installation and other work to progress within the enclosed structures.
- Currently (and for the foreseeable future) construction resources in the US are in high demand due to natural disaster reconstruction, major projects in the energy industry and a generally high level of natural resource development activity. The resulting competition for skilled labor and engineering capability will be a consideration in the quality and cost of these resources. Strategies are being prepared that will ensure that contractors selected for the project will be able to secure quality labor at competitive rates.

20.11.2 Critical Path Activities

Since neither construction nor any form of ground-breaking activity can begin until permits has been issued, PolyMet plans to use the intervening time between completion of DFS and issue of permits to progress detail engineering of long lead items such as the autoclaves, electrical transformers, and heavy mining equipment. The objective is to place conditional orders, secure positions in manufacturing schedules and/or obtain vendor drawings before the start of construction for items that have long lead times or are on the critical path.

The pre-permitting engineering phase will focus on:

- Detailed engineering for long-lead items;
- Preparation of equipment specifications;
- Obtaining competitive bids for equipment and material;
- Placement of orders and/or securing manufacturing slots in fabrication schedules for critical long lead items;
- Securing supply of materials of construction with long delivery;
- Purchase of certified drawings from vendors; and
- Preparation of bid packages for concrete and steel.

Immediately after the permits have been issued, the civil/concrete contractor will move onto site to commence earthworks and foundation construction.

Pre-permitting work in the crushing and grinding areas will involve making the area safe, collection and cataloguing of surplus equipment, spares and materials and repairs (such as to roofing, siding and sump pumping arrangements) to prevent deterioration of the assets.

20.11.3 Contracting Strategy - Construction

Labor

The Project is located in a region that has a strong tradition of organised labor. Considering the expected demand and competition for labor, PolyMet has initiated negotiations with local construction trades unions with the objective of reaching a labor agreement prior to the start of construction. The objective is to establish rates and working conditions that will be attractive to quality craftsmen while at the same time providing a degree of certainty with respect to construction outcomes. Construction contractors will source their labor through the trades unions.

Principal Contracts

The Contracting Strategy is to award several major contracts for construction (as opposed to appointing a General Contractor for the complete project). Lump Sum pricing will be obtained wherever possible when design is at an advanced stage. The following are the principal contracts to be let:

- Mine pre-production development;
- Mine infrastructure construction;
- Rail refurbishment and construction of a rail transfer hopper at the mine site;
- Refurbishment of the comminution equipment, crushing, milling and piping;
- Removal of surplus equipment notably in the flotation and reagent handling areas;
- Concrete construction in all areas other than the mine and for the railroad;
- Structural, mechanical and piping (SMP) installation for all new construction;
- Electrical installation in all areas;
- Instrumentation and control systems in all areas;
- Insulation and heat trace (design, supply and install);

- Fire control systems;
- Lined impoundments and water recovery systems;
- Cladding (design, supply and erect);
- HVAC (design, supply and install to be considered); and,
- Communications

Other than mine pre-development, the control and coordination of these contracts will be performed by the selected EPCM Company.

Pre-permitting Phase

During the pre-permitting phase, all the basic and detail engineering and procurement will be completed to allow the award of civil, concrete and structural erection contracts in time for mobilization as soon as permits are issued.

20.11.4 Organization

The implementation strategy is based on a work breakdown structure (WBS) that organizes the full scope of work into a number of major packages that are defined by distinct areas of operations and disciplines of engineering within the existing and new plant areas. The major WBS areas are:

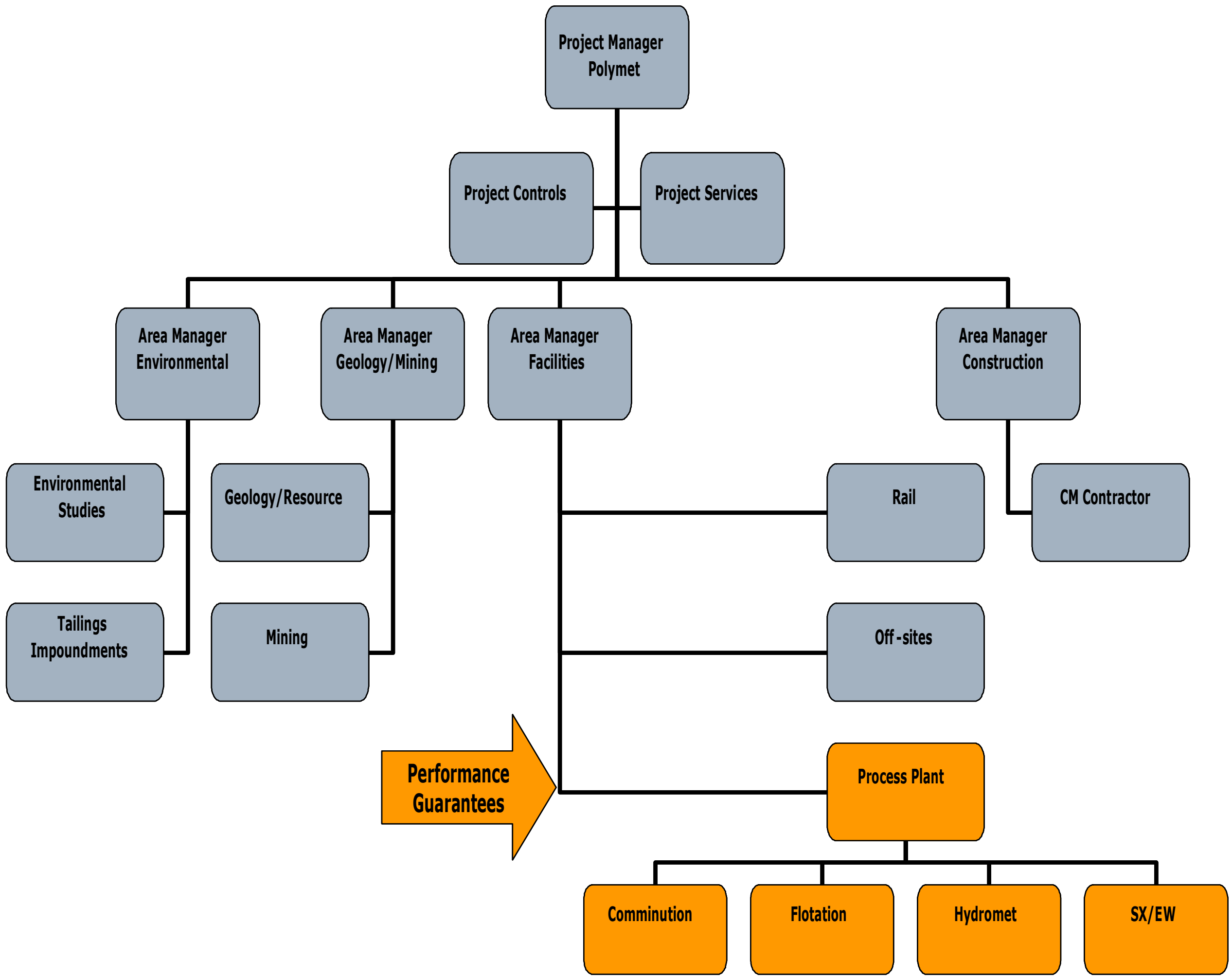
- Mine design and development
- Rail design, refurbishment and construction
- Refurbishment of the existing comminution facilities and infrastructure
- Design and construct the new process plant and infrastructure
- Existing and new tailings facilities for hydrometallurgical residue disposal and management

Local engineering and contracting companies have been identified that will be used for the rail, comminution facilities and existing infrastructure scopes of work. These companies have had previous involvement with the Erie Plant and contributed to the DFS.

Bateman will provide the Technology Package for the process plant along with certain process performance guarantees and will have oversight of the engineering companies selected to provide detail engineering and procurement services for the process plant. The nature and scope of these performance guarantees will be more clearly defined when the engineering packages are bid.

The overall organisation structure for the implementation phase is shown in the following Figure 20-10.

Figure 20-10 Organisation Structure for Project Implementation



21. INTERPRETATION AND CONCLUSION

21.1 Drill Hole Database

When work started on the DFS there already existed a large electronic drill hole data base from various campaigns of drilling going back to 1969. Because the resource model, which forms the fundamental basis of the project, is built upon this data base, it is absolutely essential that it is free from errors and can be relied upon. Accordingly, PolyMet's geologists spent considerable time and effort in the latter part of 2004 and early 2005 meticulously checking and validating the existing data. Among other things this process included ensuring any assay values that could not be supported by an assay certificate or which appeared inconsistent were discarded. Numerous checks were also carried out on drill core to ensure that core samples, sample position and length, core logs and related assay certificates corresponded. Field checks were carried out to confirm the position and elevation of drill hole collars as recorded in the data base. However, because of the passage of time, not all pre-2004 drill hole collars could be located, in which case the recorded coordinates and elevation were compared with available plans and known hole positions to avoid gross locational errors. Prior to starting resource estimation work, Dr. Phillip Hellman, carried out his own independent checks on the data base.

Having validated the historical data base, PolyMet geologists, lead by Mr. Richard Patelke, developed protocols and detailed quality assurance procedures for handling drill core, for core logging and sample preparation, record keeping and for adding or amending drill hole data. Special attention was paid to issues of data security and how new data could be added to the data base to protect against data corruption. As a result of these efforts, PolyMet is confident that the NorthMet drill hole data base is of the highest quality and can be relied upon as a basis for resource modelling and for the purposes of the Mineral Resource estimate.

21.2 Mineral Resource and Mineral Reserve Estimates

Metal price assumptions used for resource estimation are well below current and projected future metal prices, implying that the reported resource estimate is conservative. While project economics can only be based on Measured and Indicated resource that has been converted to Proved and Probable reserves, the resource model and current knowledge of the geology of the deposit suggest there is significant potential to upgrade the existing resource by additional drilling. The expectation is that additional, targeted drilling will result in some Inferred material being upgraded to Indicated, some Indicated to Measured and some material that currently can only be considered as potential may be upgraded to Inferred category. Assuming drilling results in significant upgrading of the resource estimate, additional mine planning will be required to extract the additional, potential value of that upgrade.

21.3 Metallurgical Testwork

The DFS included various programs of both bench and pilot-scale metallurgical testwork. All the DFS work as well as all PolyMet's 2001 pre-feasibility study testwork was carried out by the same team of Lakefield scientists and metallurgical engineers. This ensured good continuity and consistency between the testwork programs. Each phase of testwork was designed to build upon the results of previous testwork. The DFS work confirmed the design parameters determined during pre-feasibility the and thereby provides confidence in the DFS results.

21.4 Environmental

Environmental work started in early 2004 and has continued since then in parallel with the DFS but essentially as a separate yet directly inter-related activity. The objective of this environmental work is to identify potential impacts and to develop ways to minimize or mitigate those impacts. This includes development of a closure plan along with an estimate of the cost of carrying out that closure plan including any long term, on-going post-closure management, monitoring and treatment activities.

Environmental work can be broadly divided into two related areas of activity; environmental impact review and permitting. To assist their own specialists with review and permitting, federal and state regulators have retained top level specialist consultants acknowledged by industry as expert in their various fields and disciplines. From the outset, PolyMet and PolyMet's environmental consultants have worked closely with the regulators team to ensure that all aspects of the relevant regulations are adequately addressed and that information and data is provided to the regulators and their nominated consultants in sufficient detail for the required review and assessment process.

PolyMet's underlining project approach has been to ensure that potential environmental impacts are avoided where possible and, where this was not possible, that impacts were minimized and mitigation measures are defined. Design of mitigation measures has required quantification of potential impacts and, where there has been uncertainty, engineering and design work have been based on worst case scenarios. Having worked closely with federal and state regulators, PolyMet is now confident that all potential environmental impacts arising from implementation of the project have been identified and can be responsibly managed.

Although final air modelling still has to be completed, air emissions are expected to be minimal. Waste characterisation has progressed to the point where a waste rock management plan has been developed. Surface and ground water management at the mine site have been integrated with the waste rock management plan that requires waste stockpile effluent and "contact" run-off to be collected and pumped to the tailings basin for treatment and re-use in the process plant. Although mineral processing releases negligible amounts of mercury, compliance with GLI mercury discharge limits is potentially problematic because of mercury contamination of precipitation. However, this issue has essentially been avoided by collecting mine "contact" water and preventing untreated water discharge at the mine site.

As a result of extensive testwork and engineering, PolyMet is now confident that technically acceptable means of impact mitigation have been developed that will satisfy all relevant regulations. Moreover, the costs of environmental impact mitigation and management have been included in DFS cost estimates along with an estimate of the cost of the mandated closure bond.

The ultimate objective of ongoing environmental work is to obtain permits to mine; these permits are a prerequisite for project financing and construction.

21.5 DFS Risk Assessment

A DFS of this scope and scale would typically absorb between 25,000 – 35,000 manhours to complete. Due, in part, to the level of engineering definition required for provision of process performance guarantees, for assessment of the existing facilities and for environmental review and permitting, Bateman and its associated consultants have actually spent approximately 48,000 man hours to completion of the DFS. This total, however, does not include consultants engaged directly by PolyMet for environmental related investigations. Despite the overall, high level of engineering and design definition implied by this level of effort, Bateman, as project engineer, has independently carried out a detailed and sophisticated risk assessment of the DFS as a whole and of the metallurgical process design and engineering work, in particular. The purpose of this risk assessment was (a) to determine a realistic level of project contingency for purposes of capital cost estimation, and (b) to provide a basis for future definition of process guarantees, the scope and extent of which will be defined at a later stage of engineering. Bateman's risk assessment, which was done using its own proprietary methodology based on Monte Carlo simulation techniques, assessed project contingency at 9.9% of directs overall (rounded to 10%). This relatively low level of contingency reflects the high level of confidence in DFS engineering and cost estimation implied by the level of invested effort.

Bateman's DFS risk assessment included consideration of the ore beneficiation which, in large part, will involve refurbishment and reactivation of existing crushing and milling facilities. Because the plant has been inactive since 2001, there was concern regarding the cost of reactivation. To better quantify these costs Bateman engaged experienced local engineers to carry out detailed examination and assessment of the condition of the plant. Many of these engineers had worked at the former LTVSMC plant and therefore had direct knowledge of the plant. In addition to careful field examinations and careful research of maintenance and operating records, a number of electrical equipment items, including crusher motors, mill motors and numerous feeders, were test started under no-load conditions. The objective was to develop an indication of likely failure and mortality rates when reactivating idled plant. Of 49 items tested, none failed to start thereby adding confidence to assumptions about reactivation and hence the capital cost.

21.5 Conclusions

The following conclusions have been drawn from the results of the DFS.

- The drill hole database forms a fully validated, reliable basis for Resource estimation.
- Additional drilling has potential to upgrade the resource which, in turn, will require further iteration of the mine plan to provide an improved production schedule.
- Metallurgical testwork completed to date provides a reliable basis for process engineering and, with the benefit of Bateman's process design experience, will enable confident scale-up from pilot- to commercial-scale operations.
- Environmental work has identified and quantified all potential environmental impacts and practical, technically appropriate means to mitigate impacts have been developed. Realistic costs of impact mitigation have been incorporated into project economics.
- Risk assessment has shown that in the areas of process design and engineering, sufficient work has been done to reduce technical risks to a lower level than is often the case for a DFS.

- Economic evaluation indicates that, based on DFS operating performance assumptions, the project can provide attractive returns at a range of metal prices that are lower than current prices.

22. RECOMMENDATIONS

The following recommendations are based on DFS results;

- Although drilling to date has defined a resource adequate to support the proposed mine development, an additional program of diamond drilling is recommended to further upgrade the resource and promote to Indicated material currently categorized as Inferred, thereby extracting additional value from the deposit. The additional drilling will also assist with detailed planning of pre-production mine development. Recommended drilling will total approximately 30,000 feet (NTW or NQ2 size core) in 110 holes although actual drilling be modified depending on drilling results. Estimated budget for this drilling program is US$1.4 million.
- Once drilling results are available, update the resource model and resource estimate and carry out a further mine planning iteration with an estimated budget cost of US$35,000.
- Commence detail engineering work with the object of acquiring vendor certified drawings or placing conditional orders on crucial materials and items of equipment identified as having long lead times that could impact the construction schedule. It is further recommended to progress discussions and negotiation with engineering and construction management companies and organisations with the object of having contracts in place so as to be able to start construction work as soon after issue of permits as possible.
- Advance project financing and metal off-take negotiations with a view to finalization as soon as permits are issued.
- Conclude negotiations with construction labor unions to assure an adequate supply of skilled construction labor.
- Continue with environmental engineering, EIS related work and complete permit applications.

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24. DATE

The date of this report is 25th October 2006.

25. ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES

25.1 Marketing

25.1.1 Introduction

The NorthMet project is principally a copper project with on-site production of London Metal Exchange (LME) Grade copper cathode. It also has important revenue streams coming from two by-products in high grade intermediate form. Namely:

- Nickel/cobalt (as a mixed hydroxide),
- Precious metals in a base of copper sulphide- this diluted concentrate containing the platinum group metals (platinum, palladium), gold and some silver.

Table 25-1 below shows the revenue streams from these 3 product lines, on a recovered level (paid revenue to PolyMet).

Table 25-1 Metal Production and Revenue Streams

Metal	Base Case		Production	Contribution to net revenue
	US$/lb or oz.	Recovery %	M lbs or oz.	%
Copper	2.25	92.3	72,060	51.8
Nickel	7.80	70.3	15,400	30.7
Cobalt	16.34	40.7	0.727	3.0
Palladium	274	75.2	76,000	6.3
Platinum	1,040	72.7	20,500	6.8
Gold	540	67.0	9,500	1.6

Copper cathode is the main revenue generator and is considered to be a highly fungible product and treatment of the copper off-take may be guided by the principal lender to the project. The nickel/cobalt hydroxide concentrate is in a form which is readily useable by a number of nickel refiners. This concentrate, when dried, amounts to approximately 20,000 tons per year and will be shipped off-site in containers. The precious metals will be in the form of a dried cake amounting to 3,300 tons per year and will be shipped off-site for refining.

PolyMet will be a small to medium scale producer. The process allows one step recovery of the valuable products and the project's geographical location in the safe jurisdiction of the USA is highly advantageous. The history of mining and metal production in Minnesota with its excellent rail and lake shipping facilities and proximity to major consumer areas makes this project ideally suited to be a long term supplier.

25.1.2 Copper

World copper production in the early 1900's was less than 0.5 million tonnes. Today it is over 15 million metric tonnes. The growth of the copper industry is intimately linked with the growth of the electrical industry and copper ranks third in world metal consumption after steel and aluminium. While some substitution of copper by other metals is possible, such change is extremely slow and difficult to implement.

Some of the present tightness in copper supply is artificial because of copper stockpiled in warehouse locations far away from the consumer (fabricators).

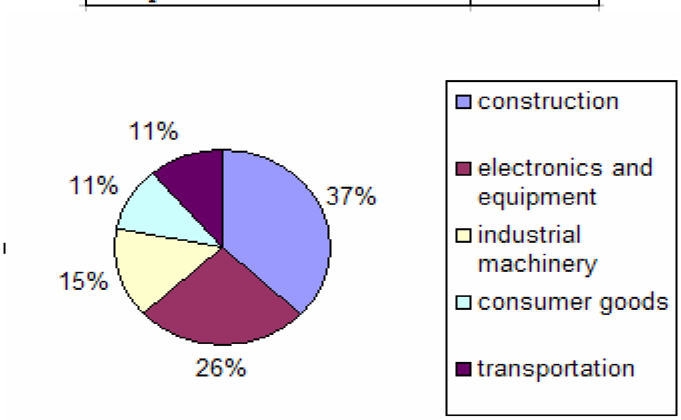
PolyMet will produce approximately 36,000 annual tons of a high quality cathode. The most likely buyers will be in neighbouring states.

Applications of copper
The world copper mine production forecast for 2006 (source International Copper Study Group) is 15.48 million metric tonnes, an increase of 606,000 metric tonnes (+4.1 %) over 2005. For 2007, an increase by 816,000 metric tonnes (+5.3 %) to 16.3 million metric tonnes is expected.

Based on 2003 figures, the copper consumption by Industry sector is shown in Figure 25-1 below.

Figure 25-1 Copper Consumption by Industry Sector – 2003 Statistics

construction	37%
electronics and equipment	26%
industrial machinery	15%
consumer goods	11%
transportation	11%



Global production - copper
Factors influencing copper production globally include the following;
- Present consumption is close to production capacity/availability.
- The top three major suppliers/producers are Codelco (11 %), BHP Billiton (7%), Phelps Dodge (6 %).

- The main source of supply is copper in concentrate to be treated by smelters for subsequent refining. Smelter capacity as of June 2006 exceeds worldwide concentrate availability by 2 million metric tons of copper contained.
- New projects tend to be more complex than in the past and with lower grades.
- Current mines have an average reserve grade of 0.82 % Cu while new "green field" projects have an average grade of 0.58 %Cu.
- With a tight supply situation anticipated until 2010, the mining industry is trying to develop new copper mines; however, with strong demand for construction materials and skilled labour from all sectors, the industry is experiencing unprecedented restrictions to growth.

Challenges for new projects and expansions of existing mines

Of the known advanced projects some are challenged by technical and political problems which hamper a fast development. Some examples:

- Water: mines in the high Andes have a basic shortage and others such as the underground Konkola mine in Zambia suffer from very high ground water inflows - with large pumping costs.
- Depth: the high costs of developing deeper underground mines such as Resolution in Arizona.
- Remote sites and poor infrastructure: the high cost of establishing access and developing infrastructure affects large deposits such as Oyu Togloi in Mongolia and El Arco in Mexico.
- Lack of community support: this is becoming prevalent in many parts of South America such as Peru and Argentina. It is also beginning to affect major producers such as Freeport McMoRan in Indonesia.

The industry has published updated costs which indicate that, depending on location, the price for copper has to exceed US$ 0.80 to1.00/lb Cu to warrant new development.

PolyMet's project in Minnesota is well located next to a well established mining infrastructure which also enjoys the benefits of abundant water supplies, excellent road and railroad access, and significant support from local communities. The open pit mining operation will use similar methods and technology and will be of approximately similar scale as the neighbouring taconite iron ore mines.

Payment premiums based on quality and savings on shipping, off-take

The proximity of PolyMet to US markets is expected to fetch premium payment.

- Quality: it is anticipated that commercial production from NorthMet will yield a high quality copper cathode, low in sulphur, lead and trace elements make it a premium product which could be sold direct to end users.
- Shipping: proximity to consumers in North America will provide certain benefits by avoiding shipping to and deposit into LME warehouses.
- Warrant fees: By selling direct to end users LME warrant fees can be avoided.
- Shipping charges and swap costs: LME related charges and costs of shipping from an LME warehouse can be avoided.

Copper supply / demand

Table 25-2 shows historical and projected future supply, demand and inventory positions to the end of 2007.

Table 25-2 Copper Supply & Demand Forecast 2006 - 2007

Copper Supply and Demand Forecast									
		2000 (act)	2001 (act)	2002 (act)	2003 (act)	2004 (act)	2005 (est)	2006 (proj)	2007 (proj)
Supply									
Mine Production	mt	13,211	13,626	13,579	13,675	14,523	15,033	16,025	16,850
Capacity Utilization	*%*	*93%*	*94%*	*90%*	*90%*	*92%*	*91%*	*89%*	*90%*
Secondary	mt	2,193	1,926	1,575	1,781	2,049	2,113	2,456	2,511
Total refined production	mt	14,828	15,644	14,997	15,237	15,868	16,770	18,080	18,940
Change on previous year	*%*	*1.9%*	*5.5%*	*-4.1%*	*1.6%*	*4.1%*	*5.7%*	*7.8%*	*4.8%*
Demand									
Europe	mt	4,784	4,732	4,636	4,746	4,997	4,903	5,050	5,150
Asia (ex. China)	mt	4,127	3,864	4,196	4,304	4,420	4,531	4,660	4,830
China	mt	1,876	2,357	2,775	3,097	3,420	3,695	3,990	4,310
N. America	mt	3,775	3,310	2,971	2,903	3,090	3,065	3,172	3,230
Rest of world	mt	800	827	779	844	893	919	945	970
World consumption	mt	15,362	15,090	15,357	15,894	16,820	17,113	17,817	18,490
Change on previous year	*%*	*7.5%*	*-1.8%*	*1.8%*	*3.5%*	*5.8%*	*1.7%*	*4.1%*	*3.8%*
Change in inventory	mt	(534)	554	(360)	(657)	(952)	(343)	263	450
Inventory – end of period	mt	1,934	2,488	2,128	1,471	519	176	439	889
Inventory – consumption	*weeks*	*6.5*	*8.6*	*7.2*	*4.8*	*1.6*	*0.5*	*1.3*	*2.5*

Table 25-3 Copper Supply and Demand Balance 2001 - 2007

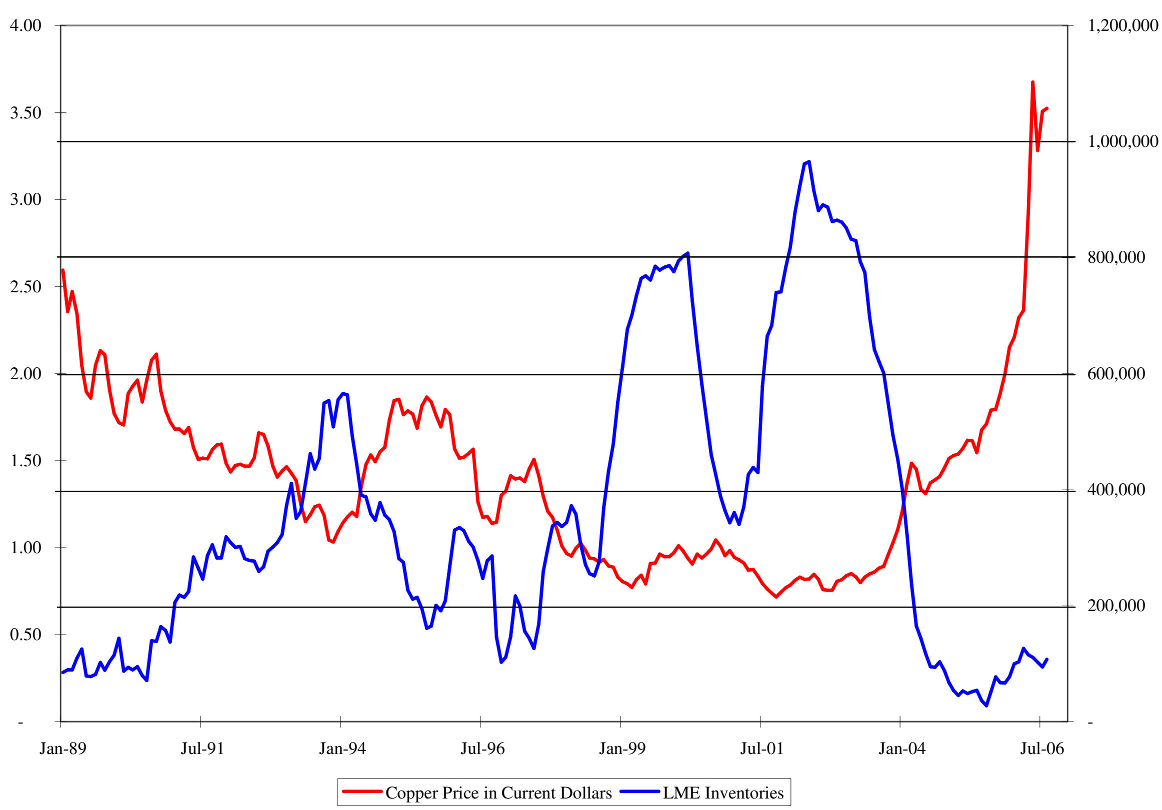
Copper Supply-Demand Balance 2001- 2007 (000 tonnes)							
	2001	**2002**	**2003**	**2004**	**2005**	**2006**	**2007**
World wide mine production	**11,185**	**11,003**	**11,090**	**11,906**	**12,275**	**12,565**	**13,220**
% change. yr-on-yr	*3.0*	*-1.6*	*0.8*	*7.4*	*3.1*	*2.4*	*5.2*
Net exports of concentrates to "Eastern" countries	385	347	448	857	1,400	1,350	1,400
Smelter losses	168	165	166	179	184	188	198
WW primary production	10,874	10,468	10,305	10,363	10,619	10,863	11,250
% change. yr-on-yr	*7.7*	*-3.7*	*-1.6*	*0.6*	*2.5*	*2.3*	*3.6*
Concentrate balance	-241	23	171	507	72	163	372
Secondary production	1,203	1,208	1,110	1,191	1,127	1,225	1,250
World wide refined production	**12,077**	**11,677**	**11,415**	**11,554**	**11,746**	**12,088**	**12,500**
% change. yr-on-yr	*4.7*	*-3.3*	*-2.2*	*1.2*	*1.7*	*2.9*	*3.4*
Net exports of metal to "Western" countries	192	109	-310	-250	-145	0	-50
Chilean stockpile	0	0	-200	200	0	0	0
World wide total supply	**12,269**	**11,786**	**10,905**	**11,504**	**11,601**	**12,088**	**12,450**
% change. yr-on-yr	1.2	-3.9	-7.5	5.5	0.8	4.2	3.0
World wide consumption	**11,810**	**11,565**	**11,640**	**12,289**	**11,729**	**12,089**	**12,270**
% change. yr-on-yr	*-8.5*	*-2.1*	*.06*	*5.6*	*-4.6*	*3.1*	*1.5*
Metal balance	**460**	**221**	**-735**	**-786**	**-128**	**-1**	**180**

Reported stock change	685	51	-267	-861	-6		
Reported stocks							
Producers	603	497	751	615	587		
Consumers	198	209	170	135	130		
Merchants	24	15	19	11	6		
LME	799	856	431	49	92		
Comex	244	362	255	44	6		
SHFE	95	75	121	32	58		
Total Stocks	1,962	2,013	1,746	885	879	878	1,059
Total as No. weeks con	8.6	9.1	7.8	3.7	3.9	3.8	4.5
LME as No. weeks con	3.5	3.8	1.9	0.2	0.4	0.0	0.0
LME cash ($/tonne)	**1,578**	**1,558**	**1,780**	**2,868**	**3,864**	**6,600**	**5,500**
% change. yr-on-yr	*-13.0*	*-1.3*	*14.2*	*61.1*	*34.7*	*7038*	*-16.7*
Global market balance							
Global production	**15,575**	**15,269**	**15,234**	**15,863**	**16,465**	**17,210**	**17,937**
Global consumption	14,896	15,161	15,643	16,693	16,498	17,271	17,821
Global balance	**679**	**109**	**-410**	**-830**	**-33**	**-61**	**116**

Copper price

Figure 25-2 below shows LME copper prices since 1989.

Figure 25-2 Copper price versus LME inventory 1989 - 2006



Emerging Economies

During the past five years, China has emerged as the most significant factor in copper consumption growth. By 2005, China represented 22% of world refined copper consumption, compared with just 4% in 1980. China now consumes more than 1.6 times as much refined copper as the US, and continuation of this growth trend would add 340,000 tonnes, or 2% to world consumption every year.

By contrast, while OECD economies still represent 43% of total refined consumption, that total has grown at a pedestrian 0.7% a year since 1980.

The importance of China and other emerging economies can be seen in the Figure 25-4 below. Between 1985 and 1990, Chinese growth contributed 0.5% to total growth. In the next five years, China contributed 0.6% to total growth, and between 1995 and 2000, China contributed 0.8% to total growth. However, in the first five years of this century, China contributed 2.1% to total growth.

25.1.3 Nickel

The global production of nickel in 2006 is estimated to be about 1.34 million tonnes and about 1.42 million tonnes in 2007. On the demand side, a conservative growth of 2.9% per annum over the long term is expected, which would bring global consumption to 1.5 million tonnes per year by 2010. This may prove to be too conservative. Consumption during 2006 is estimated to be about 1.36 million tonnes while estimates for 2007 indicate global production and consumption will be balanced and hence sensitive to production or supply side disruptions.

Nickel has played a vital role to the iron and steel industry and has been an important contributor to the development of the chemical and aerospace industries. By the late 1990's, stainless steel production accounted for 60 % of the world nickel consumption and stainless steel remains the primary factor for nickel pricing. Nickel has been traded on the LME since 1979.

Nickel prices have recently gone to historically record highs due to a fundamental shortage of supply and the long lead time to bring new projects into production. Most of the large expansions in the development pipeline involve laterite deposits. The exploitation of the laterites is relatively new and it is only since the mid 1990's that new extraction technologies allowed the economic recovery nickel from laterite ores.

Stainless steel which typically contains about 8% nickel is by far the biggest user of that metal. Other major users are the chemical industry, foundries, nickel platers and super alloys industry (aircraft engines, and turbines).

PolyMet is one of the few sulphidic ore bodies currently being developed and, when in production, will produce approximately 7,500 annual tons of nickel contained in a mixed hydroxide (with cobalt) for further processing off-site at a nickel refinery. The off-take agreements for the mixed hydroxide will be completed post DFS. Representative payment terms were used for pricing revenues in the DFS.

Sources for nickel

The three principal sources of nickel metal are the following;

- Class I nickel is obtained by electrolytic refining of concentrates obtained mainly from sulphide ores and more recently from lateritic ores.
- Class II nickel is a less pure product (containing cobalt, sulphur and other trace elements) obtained from the melting and purification of laterite ores. It can only be used for stainless steel production and some foundry applications.
- Recycling scrap: The global scrap collection system is highly sophisticated with 4 major companies serving approximately 14 major stainless steel mills. It is common to use 75% to 90 % of scrap to produce stainless steel.

Shift from sulphide ore to laterite ore

The traditional method of making nickel is the mining of sulphidic ores, production of flotation concentrates and smelting of those concentrates to make nickel matte with subsequent removal of deleterious impurities. The availability of sulphide ores and environmental pressures on mining of sulphidic ores in general have led the industry to invest heavily in new technology for the development of laterite ores and non-smelting extractive technologies. Traditionally, laterite ores were used to produce ferronickel using pyro-metallurgy. The high level of impurities carried over by this process led to an inferior ferronickel product. A new but expensive technology approach was introduced in the early 1990's using high pressure leaching to separate the pay metals, namely nickel and cobalt.

From the start, the economical and metallurgical challenges of the hydrometallurgical approach have been daunting and adoption of hydrometallurgical technology was not helped by a number of high profile failures. Fifteen years later the technical challenges have been largely overcome but the application for the treatment of laterites still represents a very large capital investment.

The nickel at PolyMet's NorthMet deposit occurs in sulphides which are treated in a pressure oxidation leaching and hydrometallurgical process to recover metal values. The selected process route has been extensively tested and operates at conditions of temperature and pressure significantly lower than those typically required for nickel laterites.

Nickel Price

The nickel price has increased dramatically since 2003. While some correction was expected during the first 6 months of 2006 due to slower demand and possibly stockpiling of finished stainless steel, prices have continued to rise. In the 2nd Quarter of 2006 there have been a number of news items released by industry leaders such as Inco suggesting that the industry is suffering a fundamental shortage of supply.

Nickel pricing is influenced by a variety of factors including the following:

- electrolytic nickel is quoted on the LME.
- ferronickel is not quoted on the LME. The price is referenced to the LME price for class I nickel and not necessarily at a discount.
- nickel in stainless steel scrap is often discounted to the LME price for nickel, depending on supply/demand.

Recent consolidation in the industry

Since 2004 there has been a strong consolidation in the industry. For example:

- Western Mining Corporation was taken over by giant BHPBilliton.
- Inco, Canada, has recently been taken over by CVRD, Brazil.
- Falconbridge, Canada, has also recently been taken over by Xstrata, Switzerland.
- Eramet-SLN, France, has taken over the Weda Bay nickel project in Indonesia.
- CVRD, Brazil, has also acquired controlling interests or outright ownership of some smaller nickel exploration and development projects.

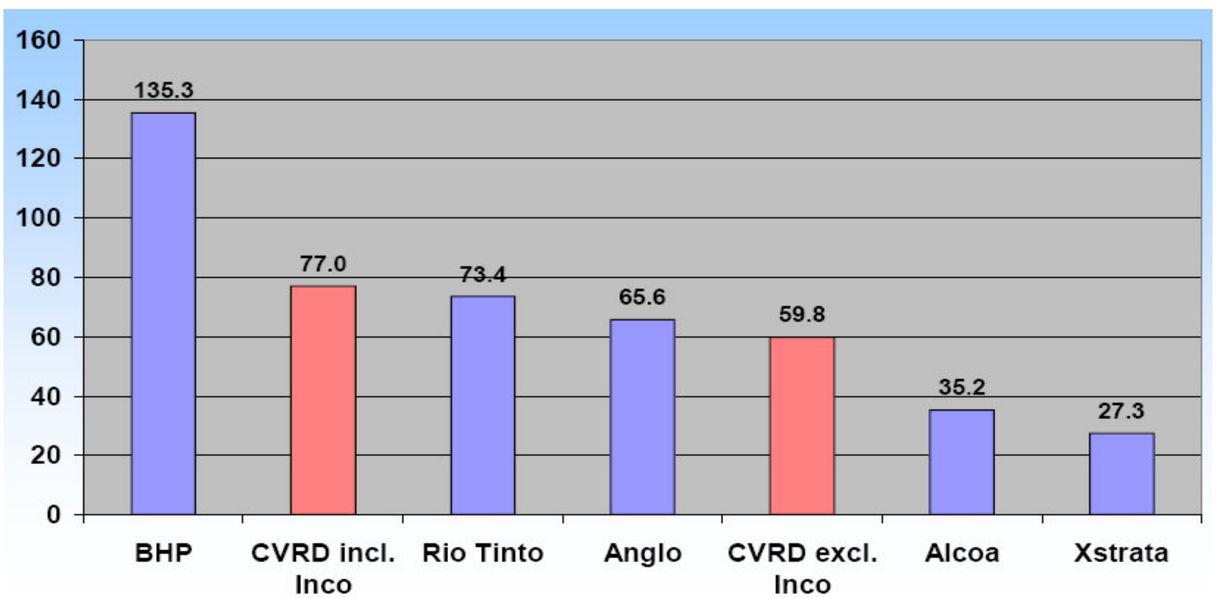
The large and influential Norilsk Kombinat in Russia has until now has not been a take-over target.

In the global arena, five major producers cover approximately 77 % of the nickel industry.

Table 25-4 Major Nickel Producers and their share of total production

CVRD/Inco	31%	(including sales of purchased nickel)
Norilsk	19%	
Falconbridge/Xstrata	11%	
BHPBilliton	10%	(including WMC, after the take-over in 2005)
Eramet	6%	
TOTAL	**77%**	

Figure 25-5 Ranking of the largest mining conglomerates by market capitalisation (in billions of US dollars)



(Source H.Pariser August 2006)

Nickel supply, demand and price forecasting

Table 25-5 Nickel supply and demand forecast

		2000 (act)	2001 (act)	2002 (act)	2003 (act)	2004 (act)	2005 (est)	2006 (proj)	2007 (proj)
Supply									
Mine Production	mt	1,174	1,224	1,248	1,261	1,308	1,363	1,412	1,508
Total refined production	mt	1,083	1,154	1,185	1,201	1,250	1,293	1,349	1,418
Change on previous year	*%*	*5.9%*	*6.6%*	*2.7%*	*1.4%*	*4.1%*	*3.4%*	*4.3%*	*5.1%*
Demand									
Europe	mt	432	461	478	468	465	451	455	465
Asia (ex. China)	mt	401	353	402	423	421	417	429	448
China	mt	63	88	98	130	156	203	230	260
N. America	mt	167	147	134	133	139	141	144	145
Rest of world	mt	59	55	63	74	71	71	78	83
World consumption	mt	1,122	1,104	1,175	1,228	1,252	1,283	1,336	1,401
Change on previous year	*%*	*3.7%*	*-1.6%*	*6.4%*	*4.5%*	*2.0%*	*2.5%*	*4.1%*	*4.9%*
Change in inventory	mt	(39)	50	10	(27)	(2)	10	13	17
Inventory – end of period	mt	67	117	127	100	98	108	121	138
Inventory – weeks' consumption	*weeks*	*3.1*	*5.5*	*5.6*	*4.2*	*4.1*	*4.4*	*4.7*	*5.1*

Table 25-6 Nickel Supply and Demand Price Forecast

		2002	2003	2004	2005	2006E	2007E	2008E
Global IP growth	%	1.9	3.7	6.0	4.5	5.2	4.3	3.6
Rate of growth stainless /IP		0.3	2.0	1.2	1.9	0.8	1.2	1.2
Growth in stainless demand	*%*	*0.6*	*7.3*	*7.3*	*8.7*	*4.0*	*5.5*	*4.0*
Stainless steel demand	mt	**16.5**	**17.7**	**19.0**	**20.7**	**21.5**	**22.7**	**23.6**
austenitic stainless	%	78	78	77	75	75	75	76
nickel in austenitic steel	%	8.6	8.5	8.5	8.4	8.2	8.0	8.0
Primary in melt	%	57	57	55	54	55	54	54
Stainless demand for nickel	kt	800	865	873	832	876	885	932
ss demand as % of total	%	68	70	68	66	65	64	64
Non-stainless demand for nickel	kt	381	378	409	436	467	499	534
Total demand for nickel	kt	1,180	1,242	1,282	1,268	1,342	1,384	1,467
Nickel demand growth	*%*	*4.8*	*5.2*	*3.2*	*-1.0*	*5.8*	*3.1*	*6.0*
Refined nickel production	kt	**1,186**	**1,219**	**1,267**	**1,275**	**1,339**	**1,392**	**1,463**
Nickel production growth	%	2.4	2.8	3.9	0.6	5.0	4.0	5.2
Market balance	kt	**6**	**-23**	**-15**	**7**	**-4**	**8**	**-3**
Global stock consumption ratio	wks	5.3	6.5	5.7	6.1	5.6	5.7	5.3
LME stocks	kt	21	24	21	36	15	20	15
LME price average	US$/lb	**307**	**437**	**628**	**669**	**881**	**1075**	**800**
LME price change year on year	*%*	*13.7*	*42.3*	*43.7*	*6.6*	*31.5*	*22.2*	*-25.6*

Source: Brook Hunt, LME, UBS estimates

Table 25-7 Nickel Supply and Demand Balance Forecast 2001 - 2007

Nickel Supply-Demand Balance 2001 - 2007 (000 tonnes)							
	2001	**2002**	**2003**	**2004**	**2005**	**2006**	**2007**
World wide refined production	**806**	**847**	**836**	**866**	**883**	**912**	**962**
% change. yr-o-yr	*4.6*	*5.2*	*-1.3*	*3.5*	*1.9*	*3.3*	*5.5*
Net exports of metal to "Western" countries	180	250	160	185	165	160	160
Stockpile sales	0	-60	60	10	0	0	0
World wide total supply	**986**	**1,037**	**1,056**	**1,061**	**1,048**	**1,072**	**1,122**
% change. yr-o-yr	1.4	5.2	1.8	0.4	-1.2	2.3	4.7
World wide consumption	**973**	**1,036**	**1,056**	**1,054**	**1,036**	**1,092**	**1,120**
% change. yr-o-yr	*-5.0*	*6.5*	*2.0*	*-0.2*	*-1.8*	*5.4*	*2.6*
Metal balance	**13**	**2**	**0**	**7**	**12**	**-20**	**2**
Reported stock change	9	-3	0	1	9		
Reported stocks							
Country stocks	80.7	75.3	73.4	77.4	71.5		
LME	19.2	22.0	24.1	20.9	36.0		
Total Stocks	99.9	97.3	97.5	98.3	107.5	87.5	89.0
Total as No. weeks con	5.3	4.9	4.8	4.8	5.4	4.2	4.1
LME as No. weeks con	1.0	1.1	1.2	1.0	1.8	0.0	0.0
LME cash ($/tonne)	**5,948**	**6,772**	**9,640**	**13,850**	**14,733**	**19,000**	**16,500**
% change. yr-o-yr	*-31.2*	*13.9*	*42.4*	*43.7*	*6.4*	*29.0*	*-13.2*
Global market balance							
Global production	1,152	1,185	1,207	1,248	1,286	1,344	1,427
Global consumption	1,104	1,177	1,219	1,262	1,282	1,367	1,425
Global balance	**48**	**7**	**-12**	**-14**	**4**	**-24**	**2**

Source: World Bureau of Metal Statistics, INSG & GFMS Metals Consulting

During 2006–2007 the nickel market is expected to remain very tight. Since 2003 there has been a supply deficit and this continues into 2006 with a full year shortage estimated at 16,000 tonnes and a predicted deficit of 10,000 tonnes in 2007. Beyond 2007, the market will continue to be very sensitive to any interruption of production or other supply disruptions. The market looks very encouraging after 2010, mainly due to Chinese requirements, where a 14% increase is expected in 2006. A more conservative increase of 7.5% is estimated in 2007. Demand is expected to remain healthy.

25.1.4 Cobalt

The majority of cobalt is produced as a by-product of copper and nickel mines and there is only one mine, located in Morocco, which produces only cobalt. Present global production and consumption are estimated to be in approximate balance at 60,000 tonnes per year.

There has always been a fine balance between worldwide supply and demand which causes cobalt to experience erratic price changes. The present swing tonnage between supply and demand is approximately 2,000 tonnes and any disruption in the market place has, as evidenced over the last 30 years, a severe effect on the price. Historically the price has ranged from a low of about US$ 6.50/lb to over US$ 30/lb.

Cobalt has a wide range of applications and uses, including in alloys, turbine engines, cemented carbides, and permanent magnets, medical applications such as joint replacement, tool steels, chemical application catalysts, tires, ceramics, animal feed additives, fertilizer and glazing. Cobalt has certain unique qualities such as heat resistance, wear resistance, high strength, and superior magnetic properties and in most applications it cannot be substituted.

In the case of PolyMet, cobalt is a co-product with nickel and therefore its sale and marketing will be closely intertwined with nickel refining. Sale of cobalt from PolyMet's operation is expected to be handled by the buyer of the nickel cobalt mixed hydroxide product and payment terms are expected to be in line with industry norms for this product.

25.1.5 Precious Metals

The precious metals concentrate will be precipitated by copper sulphide and a concentrate holding 4 % precious metals will be shipped off-site for refining in the form of a dried filter cake. Annual production of filter cake will be approximately 3,300 tons.

Typical precious metal concentrate grades will be the following:

- Cu 35.7 %
- Au 56 grams/metric tonne
- Pt 211 grams/metric tonne
- Pd 907 grams/metric tonne

One of the features of precious metals is that forward prices are typically at a premium to the spot price reflecting interest rate – making precious metals potentially an attractive element for hedging or price protection.
The platinum group metals (platinum, palladium, rhodium, ruthenium, iridium, and osmium) are amongst the scarcest of the metallic elements on the planet. Nearly all of the world's supplies of these metals are currently extracted from reef and igneous deposits in the Republic of South Africa, Russia and Canada.

The platinum-group metals have become critical to industry because of their extraordinary physical and chemical properties.

The marketing and sale of the precious metals will be by the concentrate buyer while payment terms will be in line with industry norms for this product.

The following paragraphs describe the uses of the precious metals and some of the factors that influence their price.

Platinum
During 2005, the overall, global demand for platinum grew by 2 % to 6.7 million oz.

- Demand for manufacture of automotive catalysts climbed by 330,000 oz to a record 3.82 million oz., mainly for the light duty diesel sector.
- Jewellery use reduced due to large, demand driven price increases.
- Industrial applications accounted for 1.7 million oz., mainly for LCD glass and computer hard drive discs.
- Spent auto catalyst recovery currently accounts for approximately 10 % of the platinum supply.

Platinum has a fundamental market demand driven by non-cyclical environmental and energy factors. Supply is constrained by scarcity and technical challenges. During 2005 South Africa accounted for 78 % of the global platinum supply with Russia providing about 13 % (mostly all as a by-product of Norilsk's nickel operation).

The price of platinum increased from a US$ 860 – 880/oz. range during the first half of 2005 to well over US$ 1,000/oz. during the first half of 2006 and it is expected that demand for platinum will continue to grow more strongly in the future. Therefore, after a slight deficit in supply versus demand over the last years, a moderate to large deficit is expected from the second half of 2006 onwards.

Palladium
The overall, global demand for palladium increased by 7 % to 7.0 million oz. in 2005.

- Jewellery manufacture demand increased by 54% to 1.4 million oz. largely in response to large cost increases in platinum as consumers substituted palladium for platinum. The main area of jewellery demand growth was in China.
- Auto catalyst demand increased marginally to 3.8 million oz.
- Electronics applications increased by 5 % to 965,000 oz.

Supplies fell by 2 % to 8.39 million oz. due to lower output in North America and reduced sales from Russia. A shift from platinum to palladium auto catalysts started in 2006 in the USA.

Russian palladium sales made up 51 % of palladium supply in 2005. UBS indicates that Norilsk has now exhausted its stockpile pointing to an increase in price. Countering this is increased platinum production in South Africa which will cause the availability of palladium to increase.

Slower sales from Russia, the rapidly expanding jewellery demand, and industrial applications are strong fundamentals that support the palladium price. A steady increase in the price of palladium is expected as the market returns to balance.

Gold
The gold market is perhaps the most complex of all metals because it is more of a financial asset than a commodity. Unlike commodities that are consumed, most of the gold that has ever been mined remains available to the market either in the form of recycled jewellery or, more importantly, bullion holdings of central banks and other investors.

25.2 Economic Evaluation

This economic evaluation is based on proved and probable reserves of 181.7 million tons, a mining rate of 32,000 tons per day (11.68 million tons per annum) and reflects the mine plan described in Section 19.2 above.

All resource and reserve analysis and mine modeling have been based on the following metal prices: Copper - $1.25/lb, Nickel - $5.60 per pound, Cobalt - $15.25/lb, Palladium - $210 per ounce, Platinum - $800 per ounce and Gold - $400 per ounce. This price scenario equates to a NMV of $16.09 per ton.

Table 25-8 Key Statistics

Reserves and Resources			
Measured & Indicated (M+I) Resources[1]	422.1 M tons	Copper equivalent grade	0.86% Cu
Inferred Resources	120.6 M tons	Copper equivalent grade	0.80% Cu
Proved and Probable Reserves	181.7 M tons	Copper equivalent grade	0.96% Cu

Mining			
Life of Mine average total mining rate	81,070 tpd	Plant feed rate	32,000 tpd
Initial mine life (permit application)	20 years		

Production – annual average in 1st five years			
Copper cathode (high grade)	72,057 Mlbs	Precious metals (Pt, Pd, Au)	105,984 oz
Nickel in hydroxide	15,400 Mlbs	Cobalt in hydroxide	0.727 Mlbs

Life-of-Mine operating costs per ton			
Mining cost per ton of rock mined	US$1.14	Processing cost per ton milled	US$6.99
Mining cost per ton of ore mined	US$3.13	General, Admin & other per ton milled	US$0.66

Capital Costs	
Initial Direct Cost	US$285.1M
Contingency	US$27.1M
Total	US$312.1M
Indirect Costs	US$67.5M
Total Initial capital	US$379.6M
Sustaining capital (20-year project)	US$71.8

Economic Summary – NI 43-101 Base Case	
IRR after tax	26.7%
After tax NPV @ 7.5%	US$595.4M.
Average annual EBITDA in first 5 years	US$175.3M

NOTE: 1 – Mineral Resources are estimated with an NMV cut-off of US$7.42/ton and above the 500 feet elevation.

Economic Assumptions:
Metal price assumptions for reserve analysis and pit design are deliberately conservative. The U.S.

Securities and Exchange Commission ("SEC") allows reserves to be estimated using three-year trailing average prices to the date of the reserve report, namely $1.61/lb for copper, 6.52/lb for nickel and $234, $896, and $597 per ounce respectively for palladium, platinum and gold. This price scenario equates to a NMV of $19.55 per ton.

The Base Case for economic modeling in the DFS uses metal prices that are slightly lower than those allowed by the SEC, namely: copper - $1.50/lb, nickel - $ 6.50/lb, palladium - $225/oz, platinum - $900/oz, and gold - $450/oz for a NMV of $18.67 per ton.

These prices are substantially lower than the average in July 2006 of $3.50/lb for copper, $12.06/lb for nickel, and $322, $1,241 and $634 per ounce respectively for palladium, platinum, and gold with a NMV of $36.61 per ton.

As a middle ground, we have used a market-related formula taking the weighted average of the three-year trailing average price at the end of July 31, 2006 (60%) and the average two-year forward price in July, 2006 (40%.) These prices are: $2.25/lb for copper, $7.80/lb for nickel and $274, $1,040, and $540 per ounce respectively for palladium, platinum and gold with a combined NMV of $24.82. This is the price scenario that has been applied to the case referred to herein as the NI 43-101 case.

Key Data and Economic Analysis

The economics reported in the DFS reflect the initial mine plan which in turn is based on the permit application for an ore processing rate of 32,000 tons per day for an initial period of 20 years. As previously described, the pit plan is not fully optimized and the 20-year permit application covers significantly less than half of the measured and indicated resources already defined.

Table 25-11 below sets out DFS Base Case metal price assumptions and process recovery and key operating data for the average of the first five years of full-scale production. These data comprise metal content of the three products described above, the contribution to net revenue after third-party processing costs, estimates of cash costs for each metal using a co-product basis whereby total costs are allocated to each metal according to that metal's contribution to the net revenue, cash costs on a by-product basis whereby revenues from other metals are offset against total costs and those costs divided by production – this analysis is included for copper and for nickel. The final columns show the increase or decrease in the EBITDA with a change in the price of each metal.

Table 25-9 Base Case Price and Operating Assumptions and Key Production Numbers

		Assumptions		Average of First Five Years					
						Cash costs		Sensitivity	
		Base Case $/lb or oz	Metal Recovery %	Production mmlbs or oz	Contribution to net revenue %	co-product $/lb or $/oz	by product $/lb or $/oz	Δ Price $/lb or $/oz	Δ EBITDA $'000
Copper	lb	1.50	92.3%	72.058	46.0%	0.81	0.06	0.10	6,990
Nickel	lb	6.50	70.3%	15.401	34.1%	2.84	(1.46)	0.10	1,195
Cobalt	lb	15.25	40.7%	0.727	3.8%	6.67	n/a	0.10	56
Palladium	oz	225	75.2%	75,995	6.7%	113	n/a	10	737
Platinum	oz	900	72.7%	20,531	7.8%	477	n/a	10	199
Gold	oz	450	67.0%	9,459	1.8%	239	n/a	10	92
Total precious	oz			105,984	16.3%		n/a	10	1,028

The table below sets out key financial statistics – the internal rate of return on the future capital investment and the present value of the future cash flow (including capital costs) using a 5% and 7.5% discount rate on both a pre-tax and an after-tax basis. The bottom section of the table shows the average over the first five years of full-scale production for gross revenue (before royalties and third-party processing fees), net revenues (after those costs) and EBITDA.

The price assumptions include recent actual prices (July 2006), the Base Case and the NI43-101 case described previously. The table shows a sensitivity analysis of a ± 10% change in the Base Case metal price assumptions.

Table 25-10 Economic Projections on a Range of Metal Price Assumptions

		Average July 2006	Price Assumptions			
			Main Cases		Sensitivity	
			Market Case 3-year trailing plus 2-year forward	Base Case	Base -10%	Base +10%
Metal Prices						
Copper	$/lb	3.50	2.25	1.50	1.35	1.65
Nickel	$/lb	12.06	7.80	6.50	5.85	7.15
Cobalt	$/lb	14.52	16.34	15.25	13.73	16.78
Palladium	$/oz	322	274	225	203	248
Platinum	$/oz	1,241	1,040	900	810	990
Gold	$/oz	634	540	450	405	495
Financial Summary						
Pre-tax						
IRR	%	61.0%	34.2%	17.4%	11.4%	22.9%
PV discounted at 5%	$'000	2,606,279	1,210,792	450,643	217,282	684,003
PV discounted at 7.5%	$'000	2,034,062	910,978	298,807	110,911	486,702
Post-tax						
IRR	%	47.4%	26.7%	13.4%	8.6%	17.8%
PV discounted at 5%	$'000	1,931,367	873,022	295,515	117,455	472,983
PV discounted at 7.5%	$'000	1,388,430	595,358	161,924	28,036	295,167
First 5 years:						
Average gross revenue	$'000	504,438	341,417	259,111	233,200	285,022
Average net revenue	$'000	440,257	303,147	228,067	205,091	251,044
Average EBITDA	$'000	312,382	175,273	100,193	77,216	123,169

During the first five years of full-scale production, cash costs of production (excluding amortization of capital) on a co-product basis (allocating costs to each metal according to its contribution to revenue) and using Base Case metal price assumptions are projected at $0.81/lb for copper, $2.84/lb for nickel, and $113, $477, and $239 per ounce respectively for palladium, platinum, and gold.

Alternatively, using the by-product method whereby revenues from other metals are offset against costs of a primary metal, the five-year average cash cost of copper would be $0.06/lb or, if NorthMet were viewed as a nickel mine, nickel costs would be minus $1.46/lb.

After state and federal taxes, the Base Case rate of return is 13.4% and the present value of the future cash flow discounted at 7.5% per annum is $162 million. During the first five years of full-scale operation, EBITDA (Earnings Before Interest, Taxation, Depreciation, and Amortization, or operating cash flow) is projected to average $100 million a year.

A $0.10/lb change in the copper or nickel price would increase or decrease average annual EBITDA during the first five years of full-scale operation by $7.0 million and $1.2 million respectively and a $10/oz change in all of the precious metal prices (palladium, platinum, and gold) would increase or decrease the five-year average annual EBITDA by $1.0 million.

Table 25-11 Economic Summary – First Ten Years

	Year:	0	1	2	3	4	5	6	7	8	9	10
Production												
Copper	mlbs	0.000	36.952	70.658	83.332	78.887	63.135	64.276	66.998	57.985	53.547	64.617
Nickel	mlbs	0.000	8.419	15.041	16.839	15.622	14.686	14.818	14.496	13.219	12.631	13.976
Cobalt	mlbs	0.000	0.391	0.676	0.780	0.749	0.722	0.707	0.688	0.673	0.689	0.696
Palladium	oz	0	43,123	81,791	90,838	74,678	64,952	67,717	72,178	67,267	59,756	67,379
Platinum	oz	0	10,758	22,198	26,782	19,582	16,477	17,614	18,898	17,654	17,079	18,204
Gold	oz	0	4,810	9,914	11,553	9,399	7,992	8,435	8,827	8,074	7,340	8,047
Total precious metals	oz	0	58,691	113,903	129,172	103,659	89,422	93,767	99,903	92,996	84,175	93,630
Pricing												
Copper	$/lb	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25
Nickel	$/lb	7.80	7.80	7.80	7.80	7.80	7.80	7.80	7.80	7.80	7.80	7.80
Cobalt	$/lb	16.34	16.34	16.34	16.34	16.34	16.34	16.34	16.34	16.34	16.34	16.34
Palladium	$/oz	274	274	274	274	274	274	274	274	274	274	274
Platinum	$/oz	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040	1,040
Gold	$/oz	540	540	540	540	540	540	540	540	540	540	540
Revenues												
Gross metal value	$000	-	180,808	338,200	390,569	357,489	307,653	313,173	319,256	285,719	268,355	307,521
Third party processing costs and royalties	$000	-	20,348	36,877	45,339	38,398	35,174	35,559	35,296	32,199	30,698	34,085
Net revenue	$000	-	160,459	301,323	345,230	319,091	272,479	277,614	283,960	253,520	237,657	273,437
Costs (see Table 20-11)												
Total operating costs	$000	-	89,948	124,860	130,040	130,718	127,891	125,865	127,876	128,366	130,305	132,356
Operating cash flow (EBITDA)	$000	-	70,512	176,463	215,190	188,373	144,588	151,749	156,084	125,154	107,352	141,081
Capital costs												
Direct costs	$000	285,075	9,109	6,371	7,428	6,462	8,102	6,295	7,213	6,031	4,817	6,991
Contingency	$000	27,070	0	0	0	0	0	0	0	0	0	0
Indirect and owners costs	$000	67,495	2,128	35	259	54	83	83	83	83	83	83
Total capital costs	$000	379,640	11,238	6,406	7,686	6,515	8,184	6,377	7,296	6,113	4,899	7,073
Working capital												
Total working capital		14,991	20,810	21,673	21,786	21,315	20,977	21,313	21,394	21,718	22,059	21,367
Taxable income		-	22,063	113,818	147,503	122,337	82,448	88,008	91,372	62,677	92,999	135,858
Taxation		-	8,909	36,896	47,205	39,695	27,643	29,284	30,370	21,789	29,468	42,005
Cash flow												
Free before tax		(394,631)	53,456	169,194	207,391	182,329	136,742	145,037	148,706	118,717	102,111	134,699
Free after tax		(394,631)	44,546	132,299	160,186	142,633	109,099	115,753	118,336	96,928	72,642	92,695

26. ILLUSTRATIONS AND TABLES

26.1 FIGURES

26.2 TABLES

Donald J. Hunter

Hunter Mine Engineering Services Pty. Ltd.

9 Wildwood Street

Kenmore Hills

Queensland, Australia

4069

CERTIFICATE OF SUPERVISING REPORT COORDINATOR

I, **Donald J. Hunter**, **BSc (Hons), CP (Mining), C.Eng.** do hereby certify that:

1. I am employed as a consultant by PolyMet Mining Corp.

2. I graduated with a Bachelor of Science with Honours (1st Class) Degree in Mining Engineering from the Royal School of Mines, University of London, London, England in 1973.

3. I am a Fellow of the Australasian Institute of Mining and Metallurgy and a Member of the Institute of Materials, Metallurgy and Mining.

4. I have worked as a Mining Engineer for a total of 33 years since my graduation from university.

5. I have read the definition of a "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), past relevant work experience and knowledge of the project, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for directing and coordinating the team of contributors and for compiling their inputs to the technical report entitled "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" dated October 25th, 2006 (the "Technical Report") relating to the NorthMet Project Property. I have worked on or around the project site for extended periods since July 2004 and I am intimately familiar with the property and surrounding area and infrastructure.

7. I have been involved with the property that is the subject of the Technical Report since July 2004.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am not independent of the issuer applying the tests in section 1.5 of the National Instrument 43-101.

10. I have read the National Instrument 43-101 and the Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 25th day of October, 2006.

Donald J. Hunter B.Sc. CP, C.Eng.

CHARTERED PROFESSIONAL

AusIMM

AUSTRALASIAN INSTITUTE OF MINING & METALLURGY

• MINING •

DONALD J E HUNTER

Richard Patelke
Registered Professional Geologist
3235 West Tischer Road
Duluth Minnesota 55803
USA

CERTIFICATE OF AUTHOR

I, **Richard L. Patelke**, **MSc.,** do hereby certify that:

1. I am a consultant to Poly Met Mining Inc.

2. I graduated with a Masters of Science from University of Minnesota in 1996.

3. I am registered as a Professional Geologist (30080) by Minnesota State Board of Architecture, Engineering, Land Surveying, Landscape Architecture, Geoscience and Interior Design.

4. I have worked as a geologist for a total of 17 years since my graduation from university.

5. I have read the definition of a "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am partially responsible for the preparation of the technical report entitled "Technical Report on the Results of the Definitive Feasibility Study of the NorthMet Project" dated October 25th, 2006 (the "Technical Report") relating to the NorthMet Project Property. I regularly work on the Project site.

7. I have had prior involvement with the property that is the subject of the Technical Report.

9\. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9\. I am not independent of the issuer applying all of the tests in section 1.5 of the National Instrument 43-101.

10. I have read the National Instrument 43-101 and the Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 25th day of October 2006.

Richard L. Patelke MSc.

29th OCTOBER 2006

CERTIFICATE OF QUALIFIED PERSON

I, Kenneth Gordon Baxter do hereby certify that:

1. The engineering and construction company Bateman Engineering Pty Ltd employs me in the capacity of Process Consultant Copper. I am based in the Perth, Western Australian office of Bateman Engineering Pty Ltd located at level 2 15 Ogilvie Re Mt Pleasant Perth W.A. 6153.
2. I am a graduate of the University of Queensland with a Bachelor of Science in 1974 and a Master of Scientific Studies in 1981.
3. I am a member of the Australasian Institute of Mining and Metallurgy.
4. I have been employed in the industry for 24 years both in engineering design construction and commissioning with Signet Engineering and Bateman Engineering and in operational roles in mineral processing plants.
5. I have visited the NorthMet Property in Minnesota.
6. I have been responsible for the process review for the NorthMet Definitive Feasibility Study, and in part for the engineering designs contained therein.
7. I confirm that the NorthMet Definitive Feasibility Study is subject to the terms and conditions agreed between Bateman Engineering Pty Ltd and Polymet Mining Corp.
8. I am not aware of any material fact or material change with respect to the subject matter of the NorthMet Definitive Feasibility Study Report dated October 2006 (the technical report) which is not included in the report, or the omission to disclose said material which makes the report misleading.
9. I confirm that this Certificate is subject to all disclaimers, assumptions and exclusions stated in the NorthMet Definitive Feasibility Study Report dated October 2006 (the technical report).
10. I have read instrument 43-101 and Form 43-101F1 and I confirm that the technical report has been prepared in part in compliance with certain of the requirements of both documents.
11. Nothing herein contained shall be deemed as amendment of, change to or addition to any of the liabilities of Bateman Engineering Pty Ltd as stated in the terms and conditions dated 30 August 2001 and Bateman shall not be held liable by any third party making use of or reliance upon this Certificate for any claim of whatsoever nature howsoever occurred.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

K Baxter 29/10/2006.

Ken Baxter, MSc. Std.
Process Consultant Copper
Bateman Engineering Pty Ltd
Level 2 15 Ogilvie St
Mt Pleasant
PO Box 1001 Canning Bridge 6153
Western Australia
Ph 618 9316 6900
Fax 618 9316 9041

Hellman & Schofield Pty Ltd

Suite 6, 3 Trelawney St

EASTWOOD NSW 2121

+61 2 9858 3863

AUSTRALIA

CERTIFICATE OF AUTHOR

I, **Phillip L Hellman**, **Ph D. FAIG,** do hereby certify that:

I, Phillip Hellman, FAIG, do hereby certify that:

1. I am a Director of:
   Hellman & Schofield Pty Ltd
   Suite 6, 3 Trelawney St,
   EASTWOOD NSW 2119
   AUSTRALIA

2. I graduated with a BSc (Hons) degree in geology from University of Sydney in 1973. In addition I have obtained a PhD in geochemistry and petrology from Macquarie University in 1979 and a Diploma of Education from Sydney University in 1974.

3. I am a Fellow of the Australian Institute of Geoscientists

4. I have worked as a geologist for a total of 33 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am partially responsible for the preparation of the technical report entitled "Technical Report on the Results of the Definitive Feasibility Study of the NorthMet Project" dated October 25, 2006 (the "Technical Report") relating to the NorthMet Project Property. I visited the Property on 4 & 5 June, 1998 for two days, in July 2000 for two days, in January 2003 for two days and for twenty days in September 2004, May & December 2005, and July 2006. I have been remunerated for preparing this report on the basis of a fee for services.

7. I have had an involvement in the Property since June 1998, The nature of this involvement includes resource estimation and general consulting in relation to QA/QC, geological logging and database assembly.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical report has been prepared in compliance with that instrument and form.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 27 October, 2006

Signature of Qualified Person

Phillip L Hellman

Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, **John P. Borovsky,** do hereby certify that:

1. I am a consulting environmental scientist and a Vice President at Barr Engineering Company with an office at 4700 West 77th Street, Minneapolis, Minnesota, 55435-4803..

2. I graduated from University of Minnesota College of Forestry, with a Bachelor of Science Degree in Forest Resources Management in 1971 and Master of Science Degree in Forest Ecology in 1977.

3. I am a member of the Society of American Foresters (SAF), #161003. The SAF does not have authority under statute but does require compliance with professional standards of competence and ethics (SAF Code of Ethics) and has disciplinary powers to censure or expel for violation of the Code.

4. I have worked as an environmental scientist for a total of 29 years since graduation from university/college. I have worked as a manager of environmental projects for the mining industry over the last 25 years during which time I have been involved in the regulation, permitting and environmental review of ferrous and non-ferrous mining operations and mineral processing plants.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101 as related to environmental review and permitting.

6. I am responsible for directing and coordinating other Barr consultants and compiling their inputs to the preparation of the Permitting & Environmental section 20.9 of the of the technical report entitled "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" (the Technical Report"). I have been remunerated for preparing this report on the basis of a fee for services.

7. Since March 2004 I have acted as a consulting environmental scientist to PolyMet Mining Corp of Vancouver, Canada in matters relating to the NorthMet Project. I have visited the NorthMet mine site and plant site on numerous occasions, the most recent of which include: May 23-24, 2006 to participate in site tours for state and federal permitting agencies and their third-party contractors; and July 13, 2006 to participate in a mine site wetland walkover with representatives of an environmental advocacy group.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report which is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical report.

Dated this 30th day of October, 2006.

Signature

Name of Qualified Person
John P. Borovsky

CERTIFICATE OF QUALIFIED PERSON

I, Don **E. Richard**, do hereby certify that:

1. I am a registered professional civil engineer and a Vice President of Barr Engineering Company, with headquarters at 4700 West 77th Street, Minneapolis, MN 55435-4803.

2. I have earned the following undergraduate and graduate degrees:

- Bachelor of Science, Civil Engineering, University of Wyoming, 1986
- Master of Science, Civil Engineering, University of Wyoming, 1988
- Doctor of Philosophy, Civil Engineering, University of Minnesota, 2004

3. I am a registered professional engineer (civil engineering) in the states of Minnesota, Iowa, and Michigan. I am a member of the American Society of Civil Engineers, the Water Environment Federation, and the American Society for Microbiology.

4. I have worked as a consulting engineer for a total of 18 years. My work has included remediation of soil, groundwater and sediments, including the design, installation, and operation of treatment systems for groundwater and industrial wastewaters for primarily the power, mining, and steel manufacturing industries and state and local governments.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of the waste water treatment facility design for the "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" (the "Technical Report"). I have been remunerated for preparing this report on the basis of a fee for services.

7. The waste water treatment system section of the Technical Report contains a conceptual design for a facility that will be used to treat mine process water to appropriate water quality standards prior to re-use within the metal recovery processes. The design of the waste water treatment system relies upon estimates of potential flows to the treatment system that have been developed by others using hydrologic and hydrogeologic modeling of the mine site. The quality of water that will need to be treated has also been estimated by others using humidity cell testing, and geochemical

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and have been prepared the sections of the Technical Report for which I am qualified in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 30th day of October, 2006:

Don E. Richard, Ph.D., P.E.

Licensed Professional Engineer – State of Minnesota License No. 21193

CERTIFICATE OF QUALIFIED PERSON

I, Nancy Johnson Dent, do hereby certify that:

1. I am a civil engineer with an office at 332 West Superior Street, Duluth, Minnesota.

2. I graduated from the University of Minnesota, College of Engineering, with a bachelor of Civil Engineering in 1989.

3. I am a registered Professional Engineer in the State of Minnesota, license #22740

4. I have worked as an engineer for a total of 17 years since graduation from the University of Minnesota. I worked as a water resources engineer for the entire 17 years, during which time I have been involved in the feasibility, design, construction, and operation of water management systems and related infrastructure.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of portions of the technical report entitled "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" (the "Technical Report"). I have been remunerated for preparing this report on the basis of a fee for services.

7. I conducted a site visit in December 2005 to observe the watershed and investigate the location of culverts and bridges at the mine site and the watershed that is tributary to Colby Lake. A site visit was also conducted in May 2006 to view the project site and watershed.

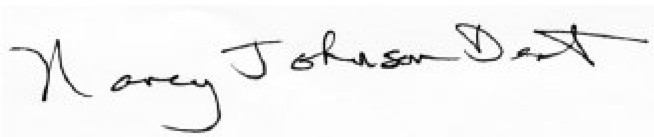
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report which is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical report.

Dated this 30th day of October, 2006.



Nancy Johnson Dent, B.CE.

CERTIFICATE OF QUALIFIED PERSON

I, John E. Quist, do hereby certify that:

1. I am are registered Professional Engineer and a Vice President at Barr Engineering Company, 4700 West 77th Street, Bloomington, Minnesota 55435.

2. I graduated from the University of Minnesota, Institute of Technology, Bachelor of Civil Engineering, 1977

3. I am a Registered Professional Engineer (Civil) in the states of Minnesota, Wisconsin, Michigan, Ohio, Indiana, and Washington.

4. I have worked as a Civil/Structural Engineer at Barr Engineering - for a total of 28 years since graduating from the University of Minnesota, Institute of Technology. I have worked as a design engineer, project engineer, and project manager on rehabilitation, inspection and evaluation, construction and modification of a variety of dam projects for mining, power, paper and municipal industries.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of the Tailings Facility Design and Development section of the "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" (the "Technical Report"). I have been remunerated for preparing this report on the basis of a fee for services.

7. The Tailings Facility Design and Development section of the Definitive Feasibility Study provides conceptual level design of the flotation tailings facility and has been prepared by me or under my direct supervision on the basis of existing geotechnical information, monitoring information, geotechnical exploration data, geotechnical material testing, site topographic survey information, flotation tailings characterization, and METSIM process model output; all associated with or relevant to the development of the tailings facility within existing Cell 1E and Cell 2E of the onsite tailings basin.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report which is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101, Form 43-101F1, and the Technical Report section for which I am the qualified person and it has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical report.

Dated this 30th day of October, 2006.

Signature

John E. Quist
Licensed Professional Engineer – State of Minnesota License No. 15590

CERTIFICATE OF QUALIFIED PERSON

I, Thomas J. Radue, do hereby certify that:

1. I am a registered professional civil engineer and a Vice President at Barr Engineering Co. with an office at 4700 West 77th Street, Minneapolis, Minnesota, 55435-4803.

2. I hold the following undergraduate and graduate degrees:
   - University of Wisconsin, Bachelor of Science – Civil Engineering – 1982
   - University of Wisconsin, Master of Science – Civil and Environmental Engineering – 1985
   - University of Minnesota – Carlson School of Management, Masters of Business Administration – 1999

3. I am an association member in good standing of the Minnesota Society and American Society of Civil Engineers. I am a registered professional engineer (Civil Engineering) in the States of Minnesota, Wisconsin, Michigan, and North Dakota.

4. I have worked as a consulting civil engineer for a total of 21 years since graduating from the University of Wisconsin. I have worked as a designer and manager on environmental permitting, design, and construction oversight of solid waste management facilities in the mining, power, and pulp and paper industry sectors for 20 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101 as related to solid waste management.

6. I am responsible for the preparation of the Hydrometallurgical Residue Disposal section 18.7.2 of the "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" (the "Technical Report"). I have been remunerated for preparing this report on the basis of a fee for services.

7. The Hydrometallurgical Residue Disposal section of the "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" is based on conceptual level design of the reactive-residue facility and has been prepared by me on the basis of site topographic survey information, ground level and aerial site photographs, site geotechnical exploration data, reactive-residue characterization data, and METSIM process model output; all associated with or relevant to development of the reactive-residue facility within Cell 2W of the onsite tailings basin.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report which is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101, Form 43-101F1, and the Technical Report section for which I am the qualified person and it has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical report.

Dated this 30th day of October, 2006.

Thomas J. Radue

Thomas J. Radue, P.E.

Licensed Professional Engineer – State of Minnesota License No. 20951