SNOWDEN

Final

Monument Mining Limited: Mengapur Project
Project No. V1165

Technical Report
November 2011

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This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Monument Mining Limited by Snowden. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Snowden's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended to be used by Monument Mining Limited, subject to the terms and conditions of its contract with Snowden. That contract permits Monument Mining Limited to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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Tables

Figures

1 Summary

Monument Mengapur Sdn Bhd (Monument) engaged Snowden Mining Industry Consultants (Snowden) to prepare a Technical Report on the status of the Mengapur Project, Pahang State, Kuantan district, Malaysia (Figure 1.1) in accordance with the requirements of Canadian National Instrument Form 43-101F (NI 43-101). The information contained within this technical report has been compiled from various other technical reports and documents to disclose relevant information about the Mengapur Project. This report is largely derived from the results of the Mengapur Project Feasibility Study of 1993 (Malaysia Mining Corporation Berhad, 1993). More recent documents are also cited, specifically in sections four, six, and eleven. This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

At least three current land positions totalling approximately 1,000 hectares cover the Mengapur 1990 historical reserve area consisting of the SP6 Design pit. Monument is in the final negotiation phases to acquire the land owned by Malaco Mining Sendiran Berhad (Malaco) referred to as Mining Certificate number PL 1/2006 or Lot 10210 (Hulu Lepar Subdistrict, Kuantan District, Pahang State) that covers approximately 185 ha (457.5 acres) and a majority of the historical reserve (Normet, 1990). The lease holder of the Malaco claim is Cermat Arman Sdn Bhd. (Cermat) which is wholly-owned by Malaco.

The Mengapur polymetallic deposit was discovered in 1979 with anomalous stream sediment samples and later drilled by Malaysia Mining Corporation Berhad (MMC) from 1983 to 1988 with diamond drilling. As part of the Feasibility report (Normet, 1990), James Askew Associates (JAA) (1990) helped determine a Cu-S-Au-Ag sulphide reserve (Table 1.1 on Zone A, and a Cu-S-Au-Ag sulphide and oxide resource (Table 1.2) on Zones A, B, and C that were originally completed by MMC staff with pit optimizations and slope designs completed by Call and Nicholas (Nicholas et al., 1991). The resource and reserve estimate reports are considered relevant because they provide an indication of the mineral potential of the project. In addition, the historical resource and reserve estimates reported in the report (Normet, 1990) use categories other than those set out in NI 43-101 and therefore should not be considered as Mineral Resources and Mineral Reserves as defined in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 2005 guidelines. These reserves and resources do not meet the requirements of the 2005 CIM Guidelines and should only be considered to be historical in nature. The historical resource report does not clearly state if this reserve is included in the resource estimate.

Figure 1.1 Mengapur Project location map

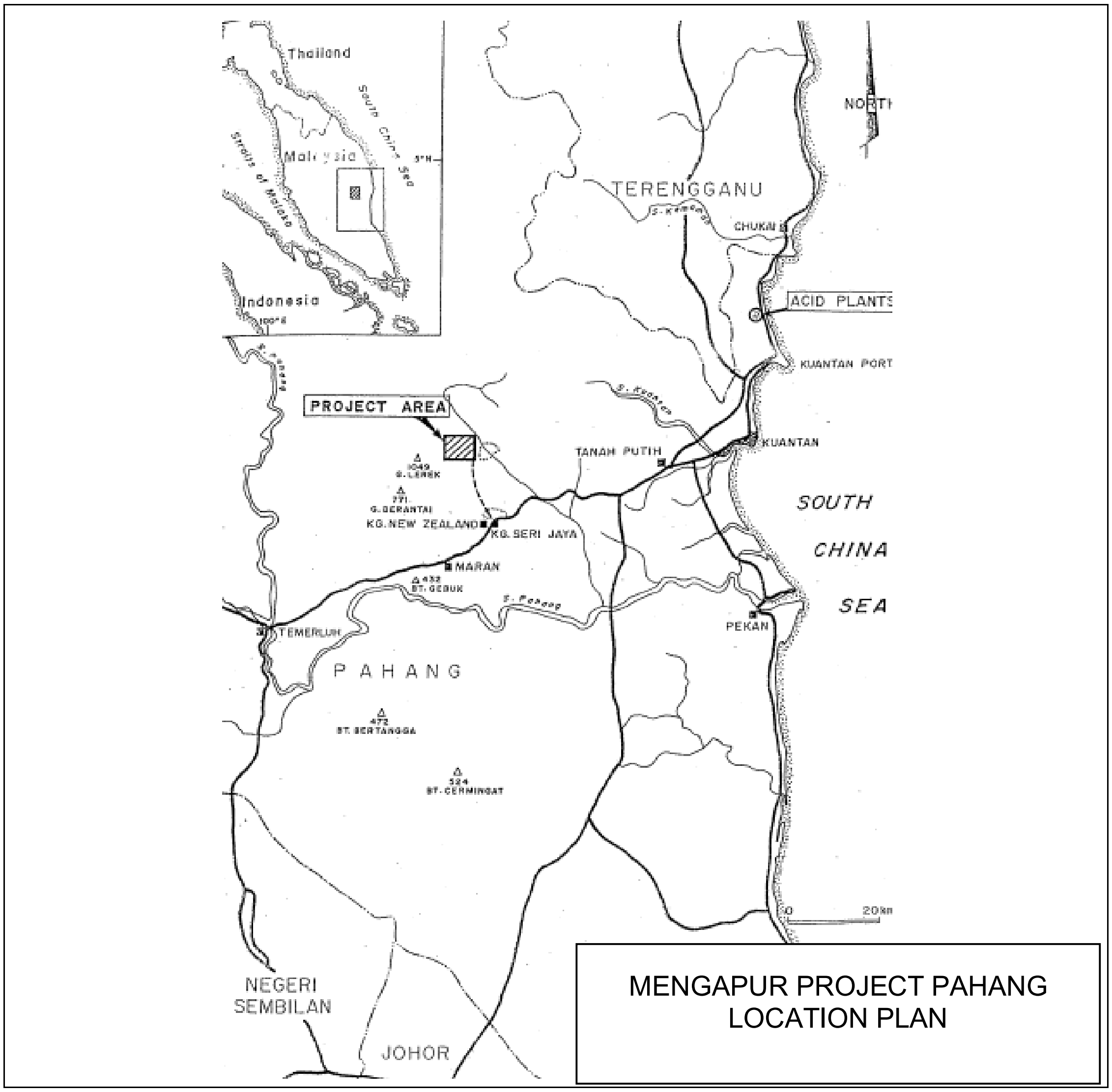


Table 1.1 Mengapur Project historical sulphide Mineral Reserve estimate of October 1990 using a 0.336% equilavent Cu cutoff grade

		Tonnes (Mt)	EQV Cu (%)	S (%)	Cu (%)	Au (g/t)	Ag (g/t)
Sulphide	Proven	26.467	0.803	9.20	0.31	0.25	2.46
	Probable	38.324	0.691	8.23	0.24	0.19	2.68
TOTAL		**64.800**	**0.737**	**8.63**	**0.27**	**0.21**	**2.59**

Notes: Equivalent Cu is based on the following assumptions: Recoveries for Cu, Ag, Au ,and S of 76.6%, 47%, 48%, and 82%, respectively; and commodity prices in US$/kg equal to 1.37 Cu, 4,107 Au; 65 Ag; and 0.09 S and a combined mining and processing cost of US$4.45/t. The historical reserve is based on the A Zone using the SP6 Design pit as defined in the Mengapur 1990 report. The disclosure of historical reserves is not meant to imply that there is any current economic viability. This would require completion of at least a preliminary feasibility study.

Table 1.2 Mengapur Project historical Mineral Resource estimate as of October 1990 using a 0.336% equivalent Cu cutoff grade

		Tonnes (Mt)	EQV Cu (%)	S (%)	Cu (%)	Au (g/t)	Ag (g/t)
Oxide	Measured	4.866	0.419	0	0.47	0.05	27.82
	Indicated	16.406	0.557	0	0.64	0.12	26.45
Subtotal		21.272	0.525	0	0.60	0.10	26.70
Sulphide	Measured	63.438	0.661	7.622	0.25	0.18	3.30
	Indicated	139,699	0.579	7.040	0.19	0.13	3.85
Subtotal		203.137	0.605	7.222	0.21	0.15	3.68
TOTAL		**224.409**	**0.597**	**6.54**	**0.25**	**0.16**	**8.86**

Notes: The same recoveries and commodity prices stated for the reserves in were used for the resources. The resources include Zones A, B, and C.

Copper and iron production has occurred at Mengapur after the 1990 resource and reserve studies by JAA and Normet (1990). A 500,000 tpa used flotation plant was constructed at the site from 2005 to 2007. Total copper production from sulphide skarn rock from October 2008 to June 2009 includes 250 t Cu ore grading 8% to 18% Cu whereas total Fe production from skarn rock from June 2010 to July 2011 totals:

- 26,693 t of iron ore to produce 3,168 t iron (magnetite) fines averaging 63% Fe with high contained sulphur content (3% to 4% S); and
- An additional 24,966 t iron ore lumps averaging 42% Fe by crushing plant.

The iron and copper processed from the copper processing plant at site was mined from mainly one open pit area located in the south-western corner of the Malaco claim.

Total Fe production from oxidized materials from October 2010 to October 2011 totals 2,556,479 t and was mined mostly from two open pits on the Malaco land. This oxidized material was transported off the Malaco claim and processed at facilities owned by another owner.

Mengapur is centred on the Bukit Botak intrusive complex with pyrrhotite-bearing garnet + pyroxene skarn, and hornfels occurring mostly in the adjacent Permian sedimentary rocks at the intrusive rock-sedimentary rock contact zone.

The Cu-S-Au-Ag mineralization is hosted in oxidised and fresh rock. Sulphide mineralogy is dominated by pyrrhotite with lesser arsenopyrite, pyrite, magnetite, chalcopyrite, and molybdenite. Oxide mineralization consists dominantly of hematite, clay, with traces of chalcocite, covellite, digenite, and/or native copper. The oxide mineralization almost always occurs at the surface and overlies the bedrock sulphide skarn mineralization.

The operations plan in the Feasibility study (Normet, 1990) recommended using an 8,500 tpd Cu processing plant operation. Under this plan, the pyrrhotite concentrate was going to be roasted to produce 590,000 tpa of sulphuric acid which would be converted to 203,000 tpa of P_2O_5 in the form of phosphoric acid. This is based on a mining rate of 753,424 tpd (2.75 Mtpa) to produce some 30,500 t of Cu concentrate and about 620,000 t of pyrrhotite concentrate per year over the proposed 23 year mine life.

The historic data compiled in this report indicates the need for more preliminary test work to be completed before the project is ready to move forward. The resource and reserve areas identified in the Feasibility report must be drilled using CIM 2005 standards.

The recommended work plan at Mengapur includes acquiring the land rights to conduct exploration and mine development studies. A first work phase is recommended consisting of due diligence work completed mostly from August 25 to November 25, 2011 at an approximate cost of CAN$788,473. A second work phase includes a 1.6 year drill hole program at an approximate cost of CAN$13,442,222, using three diamond drill rigs and one RC rig to complete a total of 65,980 m of resource conversion and infill drilling (at a 40 m average drill hole spacing for planning purposes). The total work program is estimated to cost CAN$14,230,695 and assumes that the diamond drill production is 30 m per 24-hour work shift. The second phase of work will only be performed if the first due diligence phase is successful.

Included in this 1.6 year drilling program is access road and drill pad construction, metallurgical testwork on the sulphide and oxide ores, consisting of flotation testing, grind testwork for sulphide ores, and leach tests (bottle roll and columns) for oxide ores. Work will also include geological interpretation and mine design modelling, assaying for Au, Cu, Ag, and S along with multi-element ICP, quality assurance and quality control (QAQC) assay program, and contract topographic survey work (air and ground).

1.1 Property description and ownership

The Mengapur deposit was first discovered by the Geological Survey of Malaysia (GSM) from a reconnaissance drilling program carried out in 1979/80. Twelve diamond drill holes were drilled to investigate a geochemical anomaly detected during an earlier survey. Following this, an agreement was signed between the Government of Pahang and Malaysia Mining Corporation Berhad (MMC) on August 16, 1983. Under the terms of the agreement, the State Government agreed to grant MMC and/or the Operating Company, Mining Rights within twelve months after completion of the exploration and prospecting works or studies.

On August 16, 1983, the agreement was signed between MMC, a Malaysian government owned corporation, and the State of Pahang for a 198 km^2 project area at Mengapur. The MMC interest was to be finalized after completion of a positive feasibility study. After completing a drilling program from 1983 to 1988 and a definitive feasibility study in 1990, MMC did not pursue development of Mengapur and the land reverted back to the Government of Pahang sometime after 1993.

Sometime before July 5, 2005, Cermat acquired the mining lease to Lot 10210 in Hulu Lepar Subdistrict, Kuantan District that covers a majority of the historical Proven and Probable reserve outlined in the Feasibility Study. On July 5, 2005, Malaco, a wholly-owned subsidiary of Sumatec Resources Bhd. (Sumatec), purchased 58% of Cermat. Malaco at a later time acquired the remaining 42% of the company. On June 1, 2006, Cermat signed an agreement with the State of Pahang and acquired an Operational Mining Scheme (OMS) for mining lease MC 1/2006 valid for 5 years until May 31, 2011. The OMS has recently been renewed.

On March 17, 2008, Sumatec sold all of its shares in Malaco to Diamond-Hard Mining Sdn Bhd for RM68M (approximately CAD $21.3M).

Announced in a press release on May 31, 2011, Monument entered into an agreement with Malaco to acquire a 70% pre-financing interest in the Mengapur polymetallic open pit project. Cermat is the lease holder of Mining Certificate (MC) number PL 1/2006, which is wholly-owned by Malaco. The acquisition remains subject to due diligence, signing of a Definitive Sale and Purchase Agreement, financing, board and regulatory approval and other conditions (Monument Mining, 2011).

2 Introduction

This Technical Report has been compiled by Snowden for Monument, in compliance with the disclosure requirements of National Instrument 43-101 (NI 43-101), to disclose relevant information about the Mengapur Project. The information contained in this report has resulted from compilation of exploration activities; sample data, mine design analysis, and Mineral Resource estimates obtained from historic documents and the information contained herein have not been verified by Snowden. No attempt has been made to justify historic cost or profit assumptions, and all findings of the 1993 Feasibility and Definitive Feasibility Study require updating to current parameters.

Unless otherwise stated, information and data contained in this report or used in its preparation has been provided by Monument. This Technical Report has been compiled from sources cited in the text by Mr. Walter Dzick, P.Geol, MBA, AIPG, Principal Consultant with Snowden, and Roderick Carlson, BSc, MSc, MAIG, Principal Consultant with Snowden, independent of Monument Mining and are Qualified Persons as defined by NI 43-101. Mr. Carlson visited the Mengapur Project in July 2011. Geological and land tenure status information was written and compiled by Todd Johnson, Vice President Exploration Yukon-Nevada Gold Corp. The responsibilities of each author are detailed in Table 2.1.

This report is intended to be used by Monument subject to the terms and conditions of its contract with Snowden. That contract permits filing this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws any other use of this report by any third party is at that party's sole risk.

Reliance on the report may only be assessed and placed after due consideration of Snowden's scope of work, as described herein. This report is intended to be read as a whole, and sections or parts thereof should therefore not be read or relied upon out of context.

Table 2.1 Responsibilities of each co-author

Author	Responsible for section/s
Rod Carlson	14: Mineral Resource and Mineral Reserve estimates
Todd Johnson	6: History; 7: Geological setting; 8: Deposit types; 9: Mineralisation; 0: Exploration; 11: Drilling;
Walter Dzick	All other sections

3 Reliance on other experts

The qualified persons preparing this technical report have relied on the reports, opinions, and statements of experts whose qualifications are unknown to this author as defined by NI 43-101. Information regarding aspects of the Property has been taken from the Mengapur Project Feasibility Study Report (Ahmad et al., 1993)

In development of the mineral inventory for this assessment Snowden has based its analysis entirely on the Definitive Feasibility Study written in October 1990 by Normet Engineering Pty Ltd. (Normet, 1990), with JAA completing the Ore Reserves and Mineral Resource estimates (Gillett et al., 1990).

4 Property description and location

4.1 Description

Three land positions totalling approximately 1,000 ha cover the Mengapur 1990 historical reserve area consisting of the SP6 Design pit (Figure 4.1). Monument is in the final negotiation phases to acquire the land owned by Malaco referred to as Mining Certificate number PL 1/2006 or Lot 10210 (Hulu Lepar Subdistrict, Kuantan District, Pahang State) that covers approximately 185.10 hectares (457.5 acres) and a majority of the historical 1990 Normet reserve. The lease holder of the Malaco claim is Cermat which is wholly-owned by Malaco.

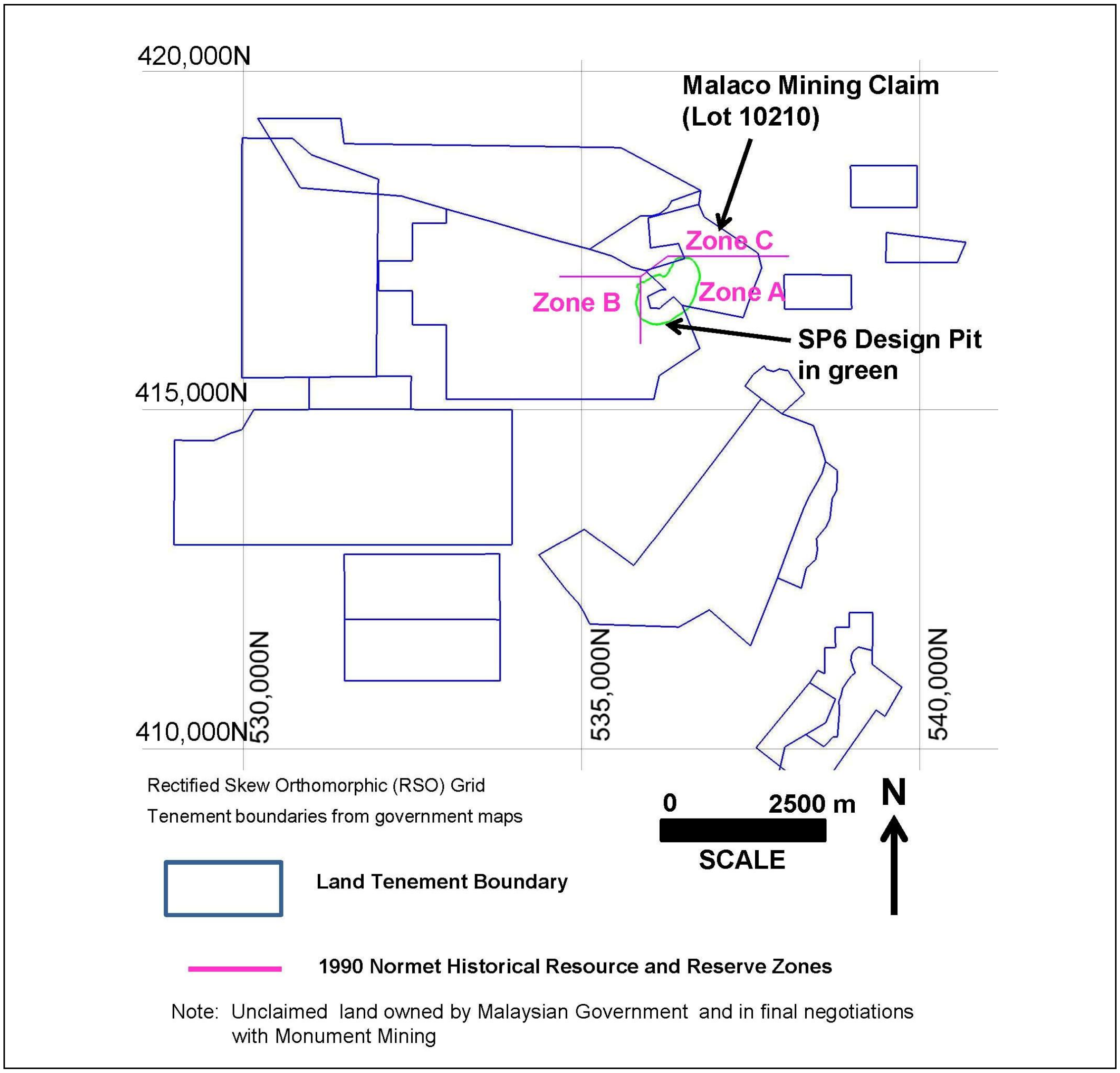
Snowden understands that Monument is currently in final negotiations with other local land holders to obtain access for further exploration and/or mining activities. Unclaimed land around the Mengapur deposit is mostly owned by the Malaysian Government and at the time of writing of this report, Monument is finalizing agreements for these lands too. The author has not reviewed the land tenure situation and has not independently verified the legal status or ownership of the properties or any agreements that pertain to the Mengapur Project.

Malaco has advised Monument and Snowden that Mining Certificate number PL 1/2006 (on Lot 10210) has several encumbrances and/or liabilities associated with it including:

- A current agreement with Zhong Cheng Mining Sendiran Berhad (ZCM) allowing them to open pit mine for Fe in soil down to the sulphide bed rock zone;
- A current agreement with Phoenix Lake Sdn. Bhd. (Phoenix) allowing the company to open pit mine for iron in soil on the same basis (with a processing facility located off of the Malaco claim).

The historic and existing open pits on the Malaco claim, which include those areas being operated by ZCM and Phoenix are described in Section 6 of this report. In addition, Malaco has an outstanding bank loan debt that is being discussed between Malaco's bankers and Monument with a view to it being paid out upon the acquisition by Monument, in the event the transaction closes.

Figure 4.1 Land map of the Mengapur area



4.2 Agreements and royalties

A tribute is payable to the State government for the extraction of minerals/metals from the land covered by mining leases. The exact level of tribute is yet to be negotiated but it is believed that the State will be keen to ensure that the project proceeds and be prepared to accept a moderate tax level. The financial evaluation included in this document assumes that 6% of gold revenue and 5% of non-gold revenue is payable as a tribute.

4.3 Environmental impacts

Monument has represented that no known environmental liabilities exist at this time.

5 Accessibility, climate, local resources, infrastructure and physiography

5.1 Physiography and climate

The Mengapur deposit is located in a complex system of ridges and valleys. The nearest major town to the Project site is Seri Jaya located 17 km to the south. Approximately 5 km west along the main highway from Seri Jaya is another village called Kampung New Zealand. A further 15 km west of Kampung New Zealand is the town of Maran. Maran is the largest populated settlement closest to the mine site.

The project area is covered by secondary jungle surrounded by virgin forest and oil palm plantations. It is situated in an area of dipterocarp forest, the majority of which was logged in 1966. Other sections have been selectively harvested since 1966. Accordingly, the majority of the forest is in a disturbed and altered condition. On the steeper and less accessible lands to the west and northwest of the orebody, primary dipterocarp forest occurs in a virtually undisturbed state.

Topography in the immediate drilling area ranges from a low of 110 m on the southeast corner in the valley to a high of 520 m at the centre of the drill area at the top of Bukit Botak hill. The A Zone reserve area has a pre-mining elevation that ranges from 200 to 320 m above mean sea level.

Mengapur is located in the Sungai Pahang Basin and is drained by a number of low order streams which discharge to the Sungai Lepar. The Sungai Lepar joins the Sungai Pahang about 50 km south-east of the site. Water quality within these streams is good. The concentrations of metals and the values of other physical parameters are all below the minimum desired quality for human consumption.

A shallow groundwater zone occurs in and is hydrologically confined to the immediate area of the proposed pit and discharges to surface streams down gradient of the pit. There is currently no utilisation of this resource.

Existing air quality at Mengapur has been generally inferred on the basis of neighbouring land use. With no existing sources likely to currently exert a major impact on air quality at the Mengapur site, SO_2 and NO_2 levels can be considered representative of ambient conditions.

6 History

6.1 History of Mengapur

Prospecting in the Mengapur area started in the late 1920's when gold was discovered on Sungai Luit draining the south edge of the Megapur area (Lee and Chand, 1981). The placer gold mining continued until the mid-1930's. During the placer mining, several galena (lead) lodes less than 3 meters wide were discovered along the stream beds. The galena was prospected in the area by two different groups in the late 1940's and in 1978 with only minor production.

In 1962 two small Malaysian Companies, the Asia Mining Company and the Jaya Sepakat Mining Company, explored for iron ore over the present Mengapur area (Lee and Chand, 1981). Three areas of skarn-type mineralization were reportedly defined at the time. Several drill holes and trenches defined a small resource of iron ore hosted in near surface soils. As of 1981 the soil-bearing iron ores had not been mined since they contained high base metal content above the marketable limits of the time (Lee and Chand, 1981).

The Mengapur deposit was first identified by the GSM from a reconnaissance drilling program carried out in 1979/80. Twelve diamond drill holes were drilled to investigate a geochemical anomaly detected during an earlier survey. Following this, an agreement was signed between the Government of Pahang and MMC on August 16, 1983. Under the terms of the agreement, the State Government agreed to grant MMC and/or the Operating Company, Mining Rights within twelve months after completion of the exploration and prospecting works or studies, whichever is the later, upon such terms and conditions to be agreed for a 198 square km project area at Mengapur.

The MMC interest was to be finalized after completion of a positive feasibility study. After completing a drilling program from 1983 to 1988 and a definitive feasibility study in 1990, MMC did not pursue development of Mengapur and the land reverted back to the Government of Pahang sometime after 1993.

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On March 17, 2008, Sumatec sold all of its shares in Malaco to Diamond-Hard Mining Sdn Bhd for RM68M (approximately CAD $21.3M).

Announced in a press release on May 31, 2011, Monument entered into an agreement with Malaco to acquire a 70% pre-financing interest in their Mengapur polymetallic open pit project (Monument Mining, 2011). Cermat still remains as the lease holder of Mining Certificate (MC) number PL 1/2006, which is wholly-owned by Malaco. The acquisition remains subject to due diligence, signing of a Definitive Sale and Purchase Agreement, financing, board and regulatory approval and other conditions (Monument Mining, 2011).

6.2 Historic production

In order to provide a more complete update to the historic information in this report the following historic production data is obtained from personal communications with Raymond Quah, General Manager of Malaco (2011) and is not included in the Normet (1993) document.

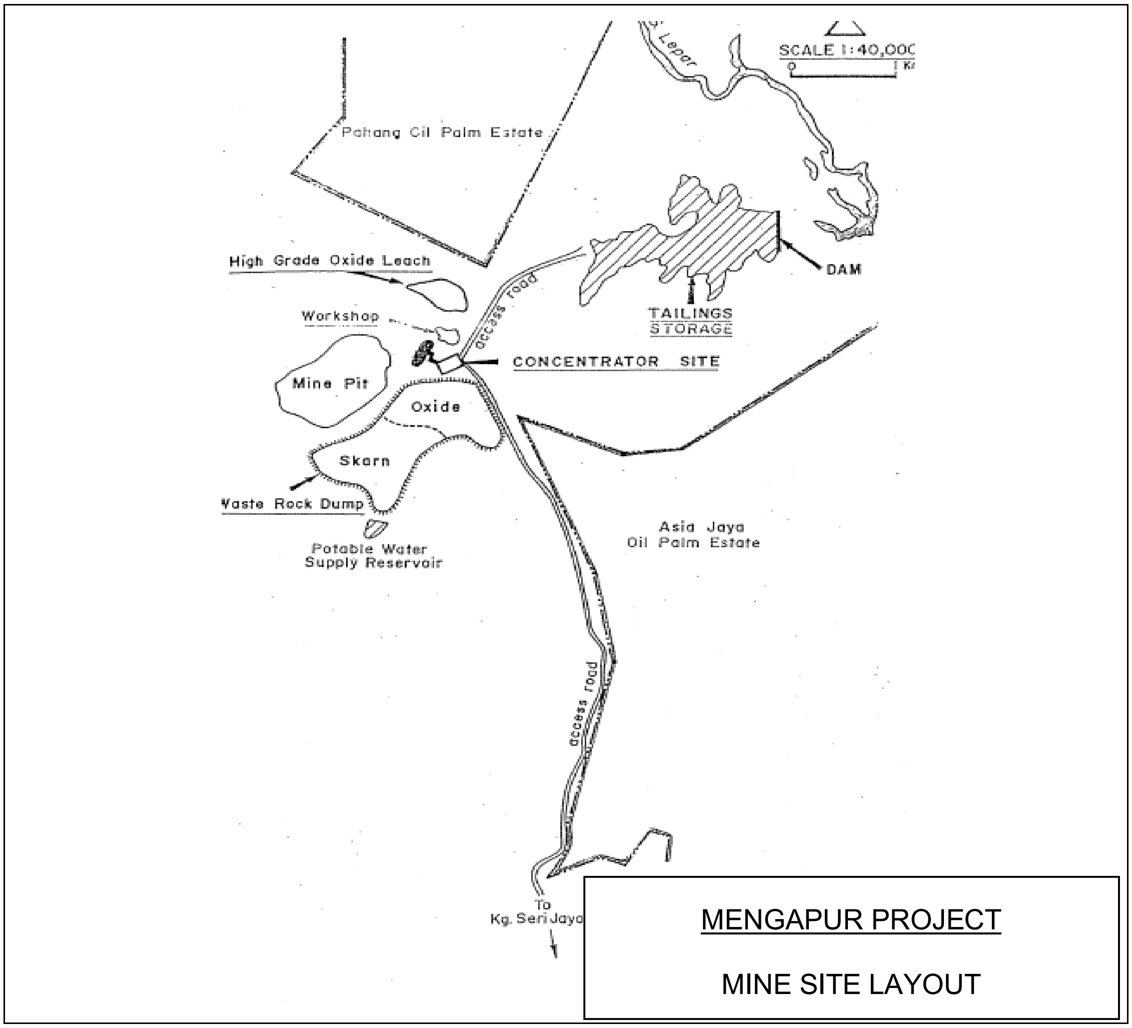
On July 5, 2005, Malaco, a wholly-owned subsidiary of Sumatec, purchased 58% of Cermat. Malaco purchased a ball mill and flotation plant from Benambra, Victoria, Australia, where it was used to process a high grade Cu-Zn deposit with a rated capacity of 500,000 t of ore per annum. The Benambra plant was dismantled, shipped to Malaysia, and reconstructed at Mengapur from 2005 to about the end of 2007. The project encountered some delays during the second half of 2007 as the Mines Department for State of Pahang (Jabatan Mineral dan Geosains (JMG)) insisted on an Environmental Impact Assessment (EIA) for the project before the issue of the annual mining licence (OMS). The first OMS was finally issued by JMG in January 2008. On June 1, 2006, Cermat signed an agreement with the State of Pahang and acquired an Operational Mining Scheme(OMS) for mining lease MC 1/2006 valid for 5 years until May 31, 2011. The OMS has recently been renewed.

In November 2007, Malaco secured a finance facility from Kuwait Finance House (KFH) that enabled him to buy out Sumatec and a year later Cermat. In the meantime Cermat had secured a mining lease (MC 1/2006) for the Mengapur reserve for an area covering 185.1 ha for a five year period from June 1, 2006 to May 31, 2011. This lease was just recently renewed for an additional 5 years up to May 31, 2016.

From January to October 2008 the copper plant construction, commissioning of the plant equipment, setup of power generating station, setup of the crushing plant and complete refurbishment of the Larox Filter Press control circuit were all carried out. The copper plant was finally commissioned on October 16, 2008.

A historic site map of the Mengapur Mine (Figure 6.1) displays the area of historic open pit Cu and Fe mining and stockpiles. Excavation earthwork for the tailings pond to support the Cu mine commenced in August 2007. Upon completion in April 2008, the earthmoving equipment was moved to Bukit Botak hill to develop the mining face. Early development of the mining face centred around drill hole number DDMEN006 where the copper bearing bedrock ore is nearest to the surface. The face was developed in descending benches until about March 2009 and halted due to tight cash flow.

Figure 6.1 Mengapur historic mine site layout (Normet, 1990)



Approximately 1.8 Mt of rock and soil material was open-pit mined from June 2008 to April 2009 to support the Cu processing plant. Production statistics are shown in Table 6.1. Approximately 1.4 Mt of soil, topsoil waste, and magnetite and/or hematite-bearing soil ore were placed in a stockpile/dump located on Lot 10210. The overburden soil covering the underlying Cu-S orebody was known to be iron bearing so the material was stockpiled for further processing in the future.

Table 6.1 Mengapur open pit material movement for southwestern pit on Malaco claim Lot 10210 (Quah, 2011)

<table>
<tr><th rowspan="2">Month-Year</th><th colspan="2">Volume Mined</th></tr>
<tr><th>Soil (m^3)</th><th>Rock (m^3)</th></tr>
<tr><td>Jun-2008</td><td>61,824</td><td></td></tr>
<tr><td>Jul-2008</td><td>69,060</td><td></td></tr>
<tr><td>Aug-2008</td><td>69,297</td><td></td></tr>
<tr><td>Sep-2008</td><td>64,861</td><td>15,086</td></tr>
<tr><td>Oct-2008</td><td>67,923</td><td>41,829</td></tr>
<tr><td>Nov-2008</td><td>55,729</td><td>2,544</td></tr>
<tr><td>Dec-2008</td><td>85,928</td><td rowspan="2">53,154</td></tr>
<tr><td>Jan-2009</td><td>48,989</td></tr>
<tr><td>Feb-2009</td><td>48,783</td><td>15,784</td></tr>
<tr><td>Mar-2009</td><td rowspan="2">53,309</td><td></td></tr>
<tr><td>Apr-2009</td><td></td></tr>
<tr><td>May-2009 to Jul-2010</td><td>Nil</td><td>Nil</td></tr>
<tr><td>Aug-2010</td><td></td><td>7,596</td></tr>
<tr><td>Sep-2010</td><td rowspan="2">5,306</td><td>5,477</td></tr>
<tr><td>Oct-2010</td><td>6,304</td></tr>
<tr><td>Nov-2010 to Dec-2010</td><td>Nil</td><td>Nil</td></tr>
<tr><td>Jan-2011</td><td></td><td>4,464</td></tr>
<tr><td>Feb-2011</td><td></td><td rowspan="2">12,233</td></tr>
<tr><td>Mar-2011</td><td></td></tr>
<tr><td>Total (m3)</td><td>631,008</td><td>164,471</td></tr>
<tr><td>SG</td><td>2.2</td><td>3.2</td></tr>
<tr><td>Total (tonnes)</td><td>1,388,218</td><td>526,308</td></tr>
</table>

A total of 59,887 t of skarn bedrock Cu ore were fed to the Cu processing plant from October 2008 to June 2009 which produced approximately 250 t of copper concentrate grading 8% to 18% Cu (Table 6.2). This ore was not processed for Fe. Teething problems were encountered and the final product did not achieve marketable copper grade. Several changes were then made to the circuit. The fine grain size of the Cu minerals made it difficult to recover Cu with less than 40 microns grind size, as this required re-grinding and re-flotation. This in turn led to higher cost and lower recovery. The ball mill lifter bars were completely worn and there was a waiting period from November 26 to December 14, 2008 for the lifter bars to be supplied from Australia. Generally the plant ran intermittently until June 11, 2009 when the plant was finally stopped due to lack of operational funds.

Table 6.2 Mengapur Cu and Fe crusher and processing plant statistics Oct 2008 to Jul 2011 (Quah, 2011)

MONTH	CRUSHER PLANTS				PROCESSING PLANT			
	For Copper	For Iron			For Copper		For Iron	
	Line 2 (tonnes)	Line 1 (tonnes)	Line 2 (tonnes)	Line 3 (tonnes)	Hrs Run	Tonnes	Hrs Run	Tonnes
Oct-2008	6,860					*3,000*		
Nov-2008	11,970					*4,000*		
Dec-2008	2,450					*5,000*		
Jan-2009	4,200					*4,500*		
Feb-2009	5,740					*4,500*		
Mar-2009	13,930				365	11,220		
Apr-2009	8,820				277	8,587		
May-2009	13,370				360	13,620		
Jun-2009	3,990				156	5,460		
Subtotal	*71,330*					*59,887*		
Jun-2010		3,750						
Jul-2010		29,375						
Aug-2010		26,875						
Sep-2010		22,750	5,436					
Oct-2010		13,640	7,578	2,157				
Nov-2010		5,390	3,780				45	1,875
Dec-2010							104	4,402
Jan-2011							40	1,594
Feb-2011							48	1,711
Mar-2011				17,437			187	7,971
Apr-2011				42,006				
May-2011				29,154				
Jun-2011							162	7,103
Jul-2011							43	2,037
SubTotal		*101,780*	*16,794*	*90,754*			*629*	*26,693*
TOTAL	71,330	209,328				59,887		26,693

Notes: 71,330 t crushed for period October 2008 to June 2009 were for copper processing. Estimated quantity milled is 59,887 t; about 15% (11,443 t) removed at waste belt before the jaw crusher; Average head grade of the ROM feed to Ball Mill is about 0.5 to 0.6% Cu; A lot of the final Cu product was recycled due to low grade; the remaining final Cu product is about 250 t Cu ore grading 8% to 18% Cu; 209,328 t were crushed for iron which produced about 24,966 t iron ore lumps averaging 42% Fe, and 26,693 t were processed for iron fines that produced 3,168 t iron fines averaging 63% Fe; about 161,104 t of non-mag lumps and fines (waste). Italicized figures are estimates. Data from Raymond Quah of Malaco (October, 2011).

By July 2009, Malaco was getting pressure from KFH regarding the repayment of the loan facility. Several potential investors were brought in to take up a stake in the Mengapur operation. In October/November 2009, ZCM collected and shipped approximately 19,190 t of Fe-ore soils from Mengapur to the port at Kuantan for testing. An agreement completed in late 2010 allows ZCM to purchase the raw iron rich soil from Malaco at US$8.75/t with all excavation, loading and hauling costs borne by ZCM. ZCM assumed all financial and monthly payment to KFH. ZCM in turn set up a large washing plant under Phoenix at a neighbouring site to the south in order to process the raw iron rich soil from Malaco. The sale of the raw iron rich soil for processing at the Phoenix mill started in October 2010 and is on-going as of the date of this report (Table 6.3). All of the reported tonnes are based on measured truck weights performed at the weigh bridge located just outside the Malaco gate entrance.

Table 6.3 Sale of iron bearing soil from Malaco claim Lot 10210 (Quah, 2011)

Date	ZCM Minerals (1) (tonnes soil)	Phoenix Lake (2) (tonnes soil)	Total (tonnes soil)
October 2010	6,075	-	6,075
November 2010	18,067	-	18,067
December 2010	30,234	-	30,234
January 2011	30,898	-	30,898
February 2011 (3)	21,743	3,793	25,536
March 2011	44,593	10,247	54,840
April 2011	74,685	-	74,685
May 2011 (4)	65,428	26,253	91,681
June 2011	40,642	65,288	105,930
July 2011	93,948	62,631	156,579
August 2011	42,545	185,042	227,587
September 2011	16,249	370,467	386,716
October 2011	507,231	840,420	1,347,651
Total	**992,338**	**1,564,141**	**2,556,479**

Notes:

(1) ZCM is processing the iron ore at the Kuantan Port at Gebeng, Kuantan

(2) Phoenix is the new iron processing plant at Seri Jaya located approximately 5 km south of the Mengapur Mine office

(3) Stopped delivery to Phoenix due to no Mineral Ore License or Mining License

(4) Phoenix plant commenced operation in mid-May 2011

(5) Sales data from Quah (2011)

Funding availabile in June 2010, allowed the Mengapur copper circuit to be modified and work commenced to set up three crusher lines to produce iron ore lumps for sale to China. The crusher plants operated from June to November 2010 and March to May 2011 to produce iron ore lumps for sale and minus 10 mm ROM feed for the iron plant. Additional small scale open pit mining of 115,436 t of material from the southwestern Mengapur pit on the Malaco claim occurred from August 2010 to July 2011. The iron plant was commissioned in November 2010 and operated until July 2011 with short breaks in January/February 2011 and April 2011 for circuit modification. During this time period, the iron processing plant at Mengapur processed:

- 26,693 t of iron ore to produce 3,168 t iron (magnetite) fines averaging 63% Fe with high contained sulphur content (3% to 4% S); and
- An additional 24,966 t iron ore lumps averaging 42% Fe.

The iron sulphate minerals contain very fine magnetite grains. The removal of the sulphur required re-grinding and re-flotation, which would contribute to higher cost and more capital outlay. The crusher lines were stopped in May 2011, and the iron plant operation was stopped in July 2011 due the lack of operational funds. The crusher lines and the Cu milling plant are currently not operating and are on care and maintenance.

7 Geological Setting

7.1 Regional geology

Peninsular Malaysia forms part of the Sunda Shield and consists of a northerly and north-northwest fold-mountain system that continues and extends from eastern Burma, through Thailand and southeastwards into Indonesian Borneo (Breward et al., 1994). The Mengapur deposit is located regionally within the Central Belt of the Malay Peninsula that is characterized by a predominance of gold and base metal mineralization (Scrivenor, 1928). The Central Belt comprises mainly shallow marine and continental margin sediments of Palaeozoic age and volcanic and volcaniclastic rocks of acid to intermediate composition. The western margin of the belt is defined by the Raub-Bentong suture that is approximately 20 km wide and consists of tectonized metasediments and ultrabasic rocks (melange-type rocks). The nature of the suture, and the tectonic evolution of the Central Belt is still being debated (Williams et al., 1994).

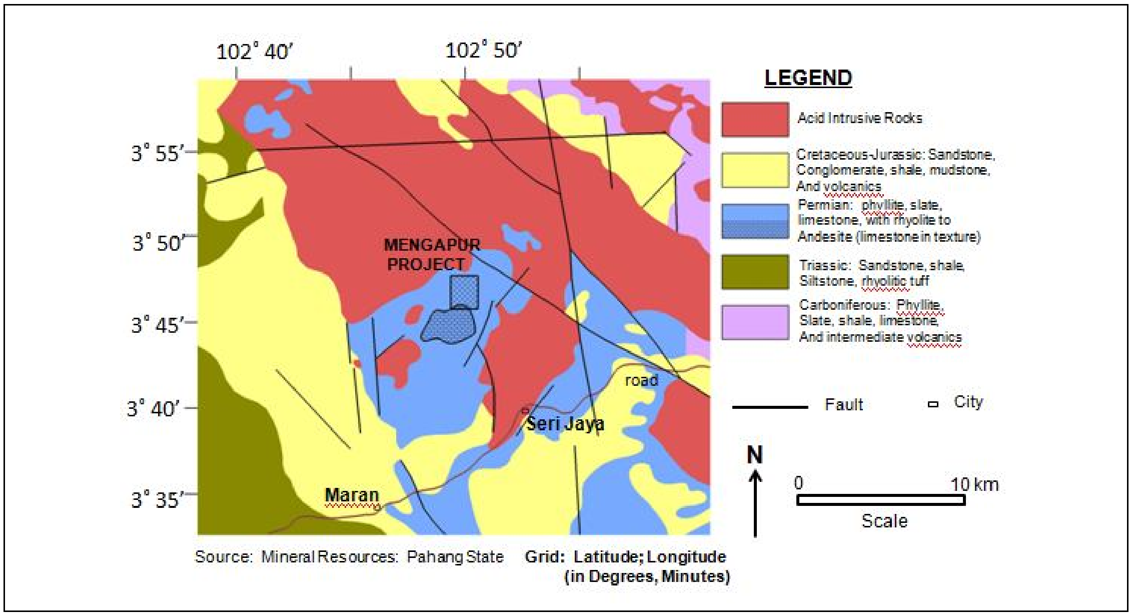
The regional geology of the Mengapur area is shown in Figure 7.1. The oldest rocks in the area are the Kambing beds, a sedimentary formation of early Carboniferous age which crop out in the northeast part of the map area. The Seri Jaya beds, consisting of the Jempul slates and the Mengapur limestones, and the Luit Tuffs unconformably overly the Kambing beds which are a sequence of interbedded argillaceous, calcareous and volcanic rocks of Permian age. The Seri Jaya beds are unconformably overlain by the Buluh sandstones, Tekam and Serentang Tuffs, a sequence of early Triassic arenites and volcanic rocks, and the Semantan Formation that consists of a group of mid-Triassic argillaceous sedimentary and pyroclastic rocks. The Hulu Lepar beds of mid-Triassic to early Cretaceous age, unconformably overly the Semantan Formation and Buluh sandstones and consist of a sequence of rudaceous, arenaceous, and argillaceous sedimentary rocks with minor volcanics.

There are three phases of intrusive rocks in the region:

1) the late Carboniferous/early Permian Dagut Granite that occur in the northwest part of the region,
2) the mid-Triassic Lepar Granodiorite that occurs in the western half of the region that consists mostly of dark gray medium-grained hornblende biotite granodiorite, biotite granodiorite, and quartz monzonite with lesser diorite, granite porphyry, and micrograntie; and
3) the Berkelah Granite that outcrop dominantly in the eastern half of the region (Lee, 1990). Intrusive rocks exposed around the Mengapur area were mapped as the Lepar Granodiorite by previous investigators. No intrusive rock exposures in the immediate area at Mengapur were mapped on the regional map in 1990 by the GSM (Figure 7.1).

Post-Mesozoic uplift, folding, and faulting occurred in the region during the Cenozoic. Faults in the region are either north-south trending or northwest-trending high-angled normal faults, or east-west and NW-SE, or NNE-SSW trending wrench faults. Numerous synclines and lesser anticlines with north-south and north-northeast striking axial planes have been mapped in the region of the Mengapur District (Lee, 1990). Quaternary alluvium consisting of unconsolidated fluviatile clay, silt, sand, gravel, and residual soil is locally abundant in the southern part of the region and covers a majority of the Mengapur Deposit.

Figure 7.1 Regional geologic map Mengapur



7.2 Local geology

The Mengapur deposit is located in the Hulu Lepar area which includes the S. Luit area and has been previously mapped by MMC and the GSM (Normet, 1990), and described by Lee and Chand (1980) and Lee (1990). Rocks in the Mengapur area are dominated by Permian Seri Jaya beds and the Mengapur limestones (Figure 7.2). The Mengapur limestones are typically massive and locally fossiliferous and/or interbedded and can be separated into two distinct facies: a calcareous facies and an argillaceous facies (Lee and Chand, 1980). The younger calcareous facies consists of dark grey carbonaceous limestone locally interbedded with calcareous shale. This unit forms the prominent steep-sided hills in the area. Stylolites have been observed in this unit. The argillaceous facies consists of calcareous shale, graphitic slate, quartz-sericite phyllite, schist, quartzite, and minor interbeds of andesitic, dacitic, and rhyolitic tuff. The sedimentary rocks strike north-northeast and dip steeply to the east-southeast 45˚ to 85˚ based on previous mapping and drill hole information (Figure 7.3).

Figure 7.2 Schematic bedrock geology, Mengapur Project, Malaysia

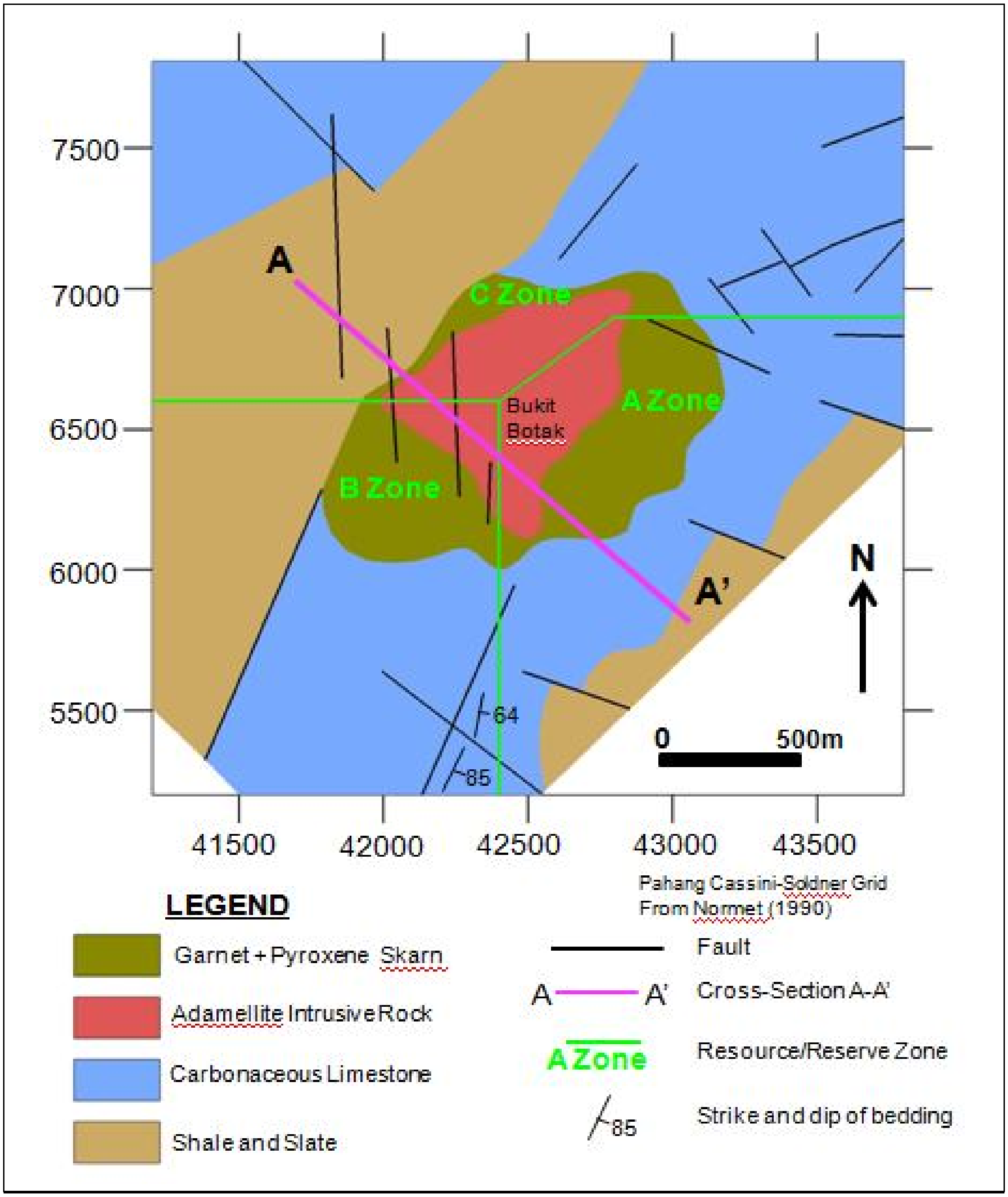
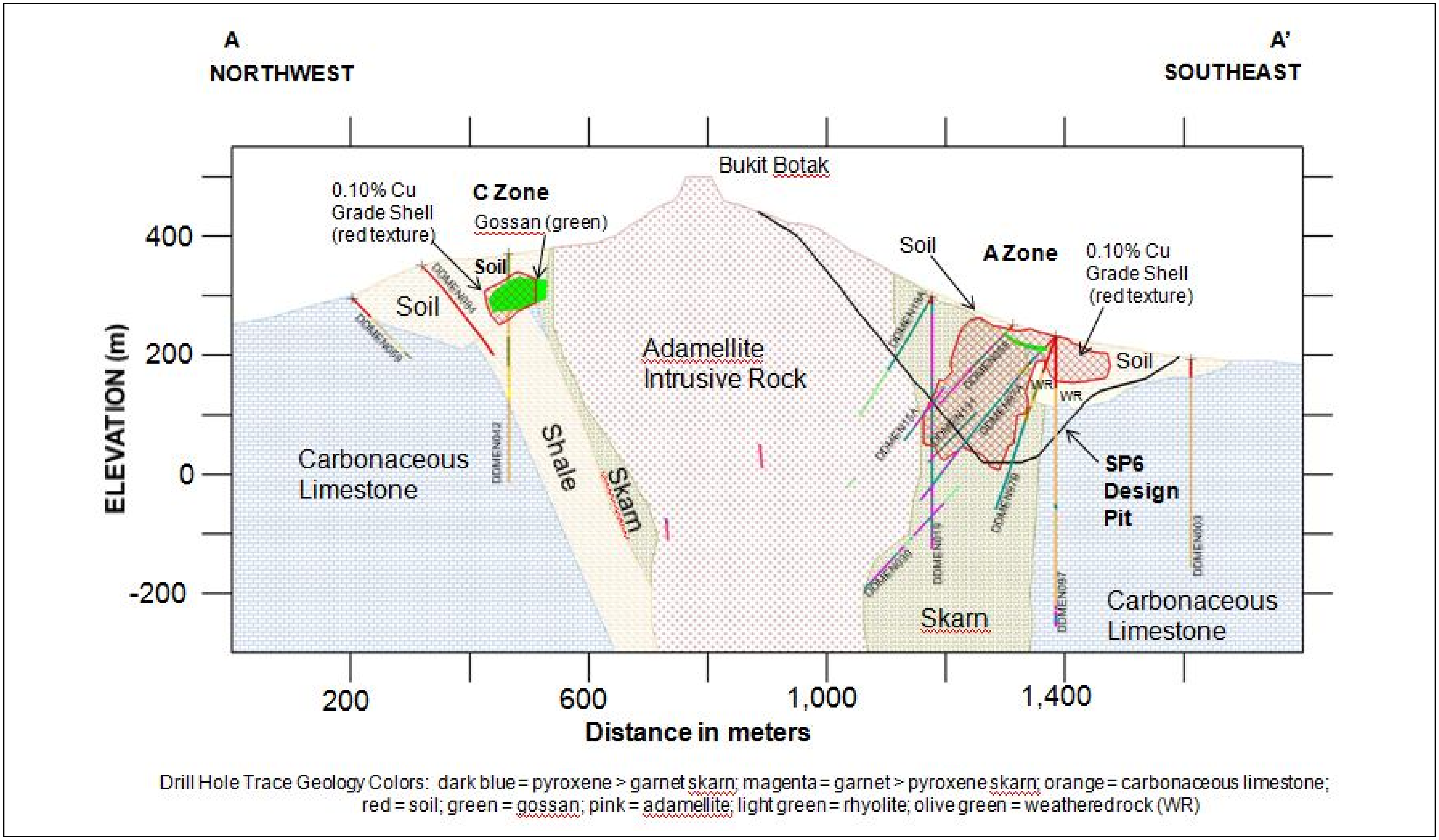


Figure 7.3 Geology Cross-Section A-A' Showing the SP6 Design Pit: Mengapur Project, Malaysia

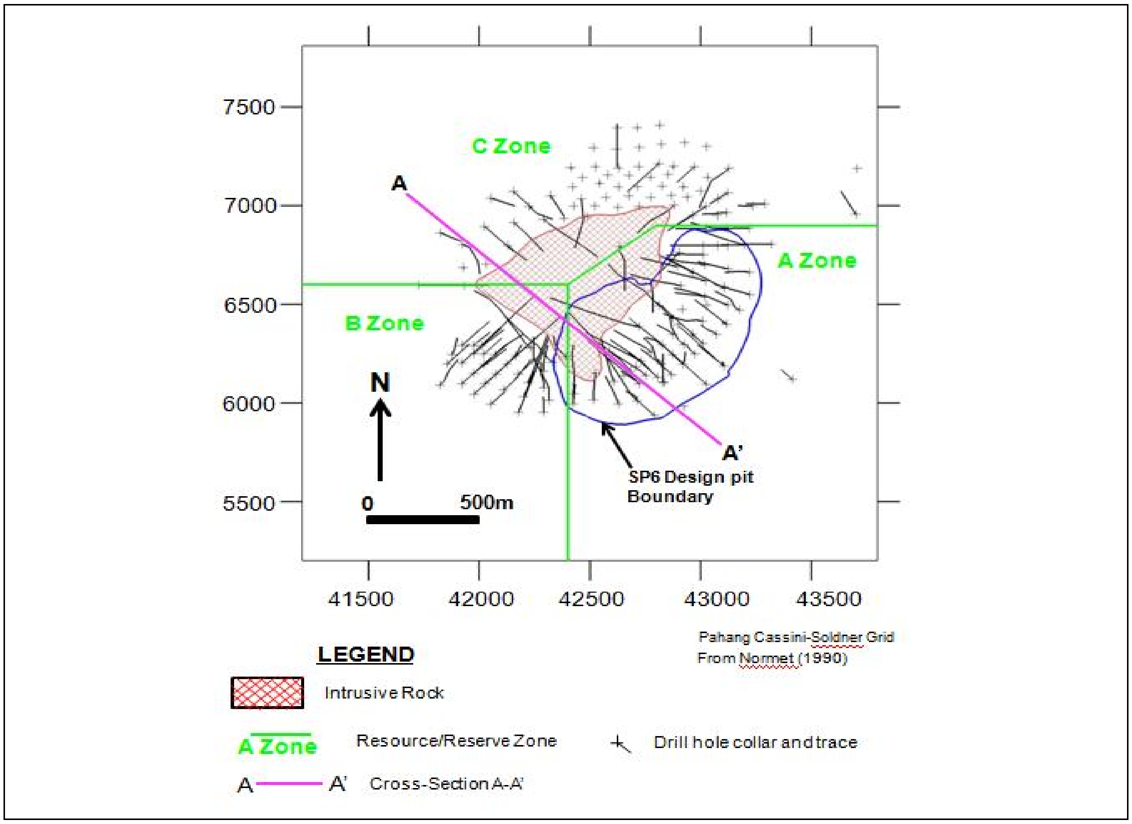


The Mengapur limestones have been intruded by an intrusive complex dominated by adamellite (quartz monzonite) with lesser amounts of rhyolite, rhyolitic tuff, and rhyolite breccia (Figure 7.2). The intrusive complex forms the centre of the Mengapur district and forms a pronounced hill in the area called Bukit Botak. The adamellite consists of a coarse grained core and a finer grained outer chilled margin. The intrusive rocks strike approximately 60° to 65° at the surface and generally dip 60° to 65 degrees to the east-southeast. The intrusive complex is believed to be related to the Lepar Granodiorite which is believed to be Mid-Triassic in age; no published age dates have been recorded on the intrusive rocks at Mengapur.

The structure in the area is dominated by north-south and NW-SE trending high-angled faults and folding. The Bujit Botak Intrusive Complex intruded the Permian Mengapur limestone sequences along the western limb of a synclinal fold. Oriented core drilling by Call & Nicholas determined there to be two dominant fault orientations at Mengapur: a set striking 10°-30˚ and a second set striking 270°-315˚ (Nicholas et al., 1990). Both sets of faults are steeply dipping and consist of broken rock zones with no slickensides, clay, or gauge (Nicholas et al., 1990). MMC geologists envisioned a major east-west wrench fault zone on the northern margin of the intrusive complex which may correspond with the Lerek Fault trend mapped by the GSM.

Soils are locally very thick at the margins of the intrusive complex where they can locally reach up to 300 m in thickness. The soils are thickest on the northern and southwestern flank of the intrusive complex. The soils in the southeastern part of the ore deposit only reach up to 120 m in thickness. The soils are commonly clay bearing and light brown to dark red in colour with the reddish soils typically containing hematite. Hematite-rich soils were logged in the historic drilling and referred to as gossan. Magnetite locally occurs both as gravel to cobble-sized gravel pieces and/or as fine free grains disseminated throughout the soil and/or in gossan zones. The magnetite has locally been exploited in recent open pit mining. Since soils cover a majority of the Mengapur deposit, the historic drilling done by MMC has identified the distribution of geologic units, hydrothermal alteration, and Cu-S-Au-Ag mineralization. A plan map showing the historic drill hole collars and the SP6 design pit, and the A, B, and C resource zones is shown in Figure 7.4.

Figure 7.4 Plan map of collar locations, intrusive rock outcrop, resource zones, and the SP6 design pit boundary



Hydrothermal alteration: skarn and quartz Veins

Hydrothermal alteration at Mengapur is centred on the Bukit Botak intrusive complex with skarn, calc-silicate hornfels, peltic hornfels and quartz hornfels occurring in the adjacent Permian sedimentary rocks at the intrusive rock-sedimentary rock contact zone. The skarn alteration extends outward into the sedimentary rocks up to 400 m to 500 m wide laterally from the intrusive complex in the southwest and southeast areas, respectively (Figure 7.2). The skarn alteration is much less developed on the northern margin of the intrusive complex where it is locally encountered at depth below the deep surface soils in deep drill holes. The skarn alteration dips steeply to the southeast and extends down to 600 m below the surface in the southwestern part of the deposit. The skarn alteration is dominated by pyroxene-rich skarn and lesser garnet-rich skarn. The garnet has been identified as andradite in composition whereas the pyroxene has been identified as diopside. Both skarn varieties can contain small to high amounts of sulphide and iron-oxide minerals. Other silicate minerals noted in the drill hole geology logs or published reports include idocrase, actinolite, tremolite, chlorite, epidote, quartz, carbonates (calcite, siderite), sphene, plagioclase, and scapolite (Lee and Chand, 1981). Andalusite was observed locally in a slate rock. Retrograde alteration of the earlier formed pyroxene and garnet skarn at Mengapur is very minor based on previous descriptions.

Other alteration assemblages in the mapped skarn zone as documented by Lee and Chand (1981) and MMC (1990) include:

- Pelitic or calc-silicate hornfels consisting of equigranular quartz and interstitial chlorite with occasional actinolite, diopside and/or garnet in the matrix; calc-silicate hornfels dominated by diopside and or garnet is also locally present.

- Quartz hornfels developed in impure tuff units and/or quartzite, and/or silicification consisting of equigranular quartz with biotite and minor to moderate muscovite; this assemblage may contain feldspars locally.
- Sericite-quartz hornfels developed in politic rocks rich in fine-grained muscovite.
- Marble (recrystallised limestone), and/or calcification of sedimentary rocks (carbonate veins).
- Sheeted quartz-rich veins with various amounts of carbonate and sulphide minerals.

Hydrothermal alteration in intrusive rocks

The intrusive rocks in the Bukit Botak Intrusive complex are primarily silicified (Lee and Chand, 1981). Silicification is most abundant and occurs as both pervasive flooding and as quartz-rich veins near the contact zone with the skarn alteration in adjacent sedimentary rocks. The quartz-rich veins commonly make up to 10 percent of the intrusive rock and locally up to 20 percent of the rock based on observed surface samples near the eastern margin of the intrusive complex. Chalcopyrite, pyrite, and molybdenite are common in altered intrusive rocks as disseminations and in veins. Fluorite has been observed locally in the granitic rocks where it may occur with quartz, chalcopyrite, and molybdenite as disseminations and/or veins (Lee and Chand, 1981). Additional characterization of the hydrothermal alteration hosted in intrusive rocks at Mengapur is warranted.

Mineralization (Cu-S-Au-Ag)

The Mengapur deposit hosts both a sulphide (hypogene) Cu-S-Au-Ag ore body and an oxide (supergene) Cu-Au-Ag ore body. The bulk of the Cu-S-Au-Ag sulphide mineralization is hosted in sulphide-bearing pyroxene and garnet skarn. Lesser amounts of Cu-Au-Ag mineralization is hosted in oxidized soil, gossan, and locally weathered rock units that overly the sulphide-bearing skarn. The mineralogy of the mineralized sulphide-bearing skarns at Mengapur has been previously described by Sinjeng (1993) and Lee and Chand (1981) in published reports and by Normet (1990) in unpublished reports. The mineralogy of the supergene oxidized ores at Mengapur have been described in Normet (1990) and MMC (1993).

The resource and reserves have been separated into an A Zone, a B Zone, and a C Zone which occur on the southeast quarter, southwest quarter, and the north half of the intrusive complex (Figure 7.2). The SP6 Design pit (Figure 7.4) was designed by JAA and MMC and included in the Feasibility document (Normet, 1990). The bulk of the Cu-S-Au-Ag proven and probable reserves in the Feasibility report are contained in the SP6 Design pit that is located mostly in the A Zone.

Sulphide ores

Both the garnet-rich and pyroxene-rich skarn varieties contain low to locally high amounts of sulphide and/or iron-oxide minerals. The most dominant sulphide minerals in the skarn is pyrrhotite followed by lesser amounts of pyrite, arsenopyrite, molybdenite, and chalcopyrite. Pyrrhotite makes up the majority of the sulphur resource and occurs as massive zones or disseminated within the pyroxene skarn and garnet skarn. The sulphur resource and reserve typically occurs within the Cu resource and reserve; the 10% sulphur grade shell typically lies within the 0.05% Cu shell (Figure 7.5 and Figure 7.6). Pyrrhotite has a composition of 60.4% to 61.8% Fe, and 38.2% to 39.6% sulphur based on limited work by MMC and Normet (Normet, 1990). Chalcopyrite occurs most commonly as fine disseminations throughout the skarn rocks, on the margins of pyrrhotite, and in late quartz veins. Accessory sulphide minerals in sulphide ores include: molybdenite, galena, sphalerite, marcasite, chalcocite, covellite, cuprite, native copper, native bismuth, boulangerite, bouronite, terahedrite, scheelite, freibergite, pyrargyrite, cassiterite, kesterite, anglesite, and native gold. Iron-oxide minerals in pyroxene and garnet skarn are dominated by magnetite. Specular hematite has been noted in some of the geology drill hole logs to occur in skarn but is not common. The magnetite is locally intergrown with pyrrhotite in the skarn.

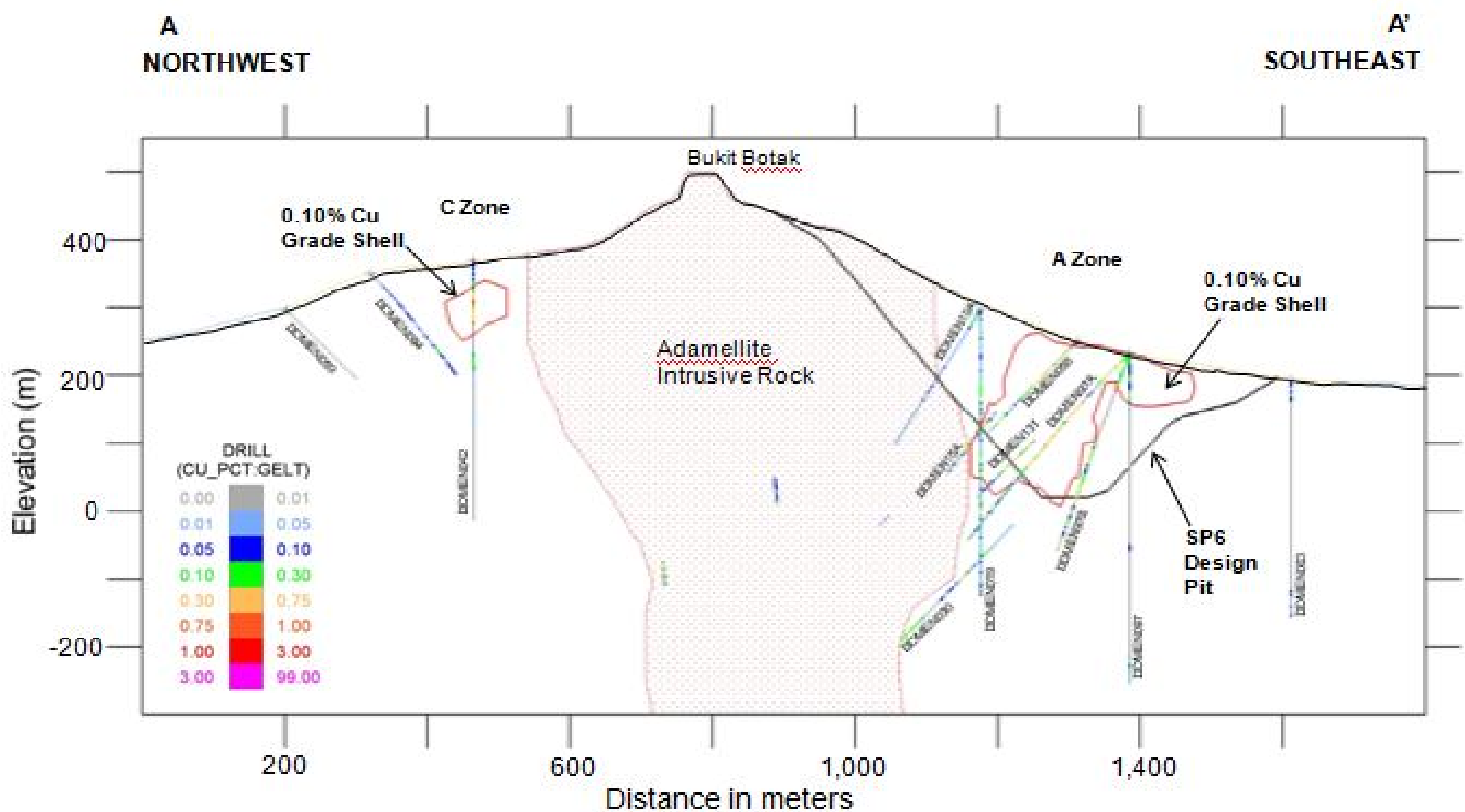
Quartz veins up to 2 meters in width locally cut the skarn assemblages as sheeted veins at similar orientations and contain various amounts of the following sulphide minerals in approximate order of abundance: arsenopyrite, molybdenite, pyrrhotite, pyrite, chalcopyrite, galena, sphalerite, tetrahedrite, native bismuth, and native gold. Lead and zinc veins are common in the marble and may also be associated with boulangerite. Accessory minerals in the quartz veins include calcite, sericite, and siderite.

Oxide supergene ores

Supergene ore zone in the SP6 Design pit is hosted in gossan, soil, and in minor amounts in weathered rock and weathered skarn. The supergene ores occur throughout the oxidized zone, but are typically concentrated in higher abundance directly overlying the sulphide orebody in bedrock skarn where the ores are approximately 3 to 9 m thick (Normet, 1990). The mineralogy of the oxide supergene ore consists dominantly of chalcocite, digenite, covellite, cuprite, and pyrite. Minor green copper oxide minerals have been observed in the soils where they occur in clay, hematite, and other iron oxides (goethite and limonite). The soil and gossan can be elevated in Cu, Au, Ag, As, Bi, As, Pb, and Zn.

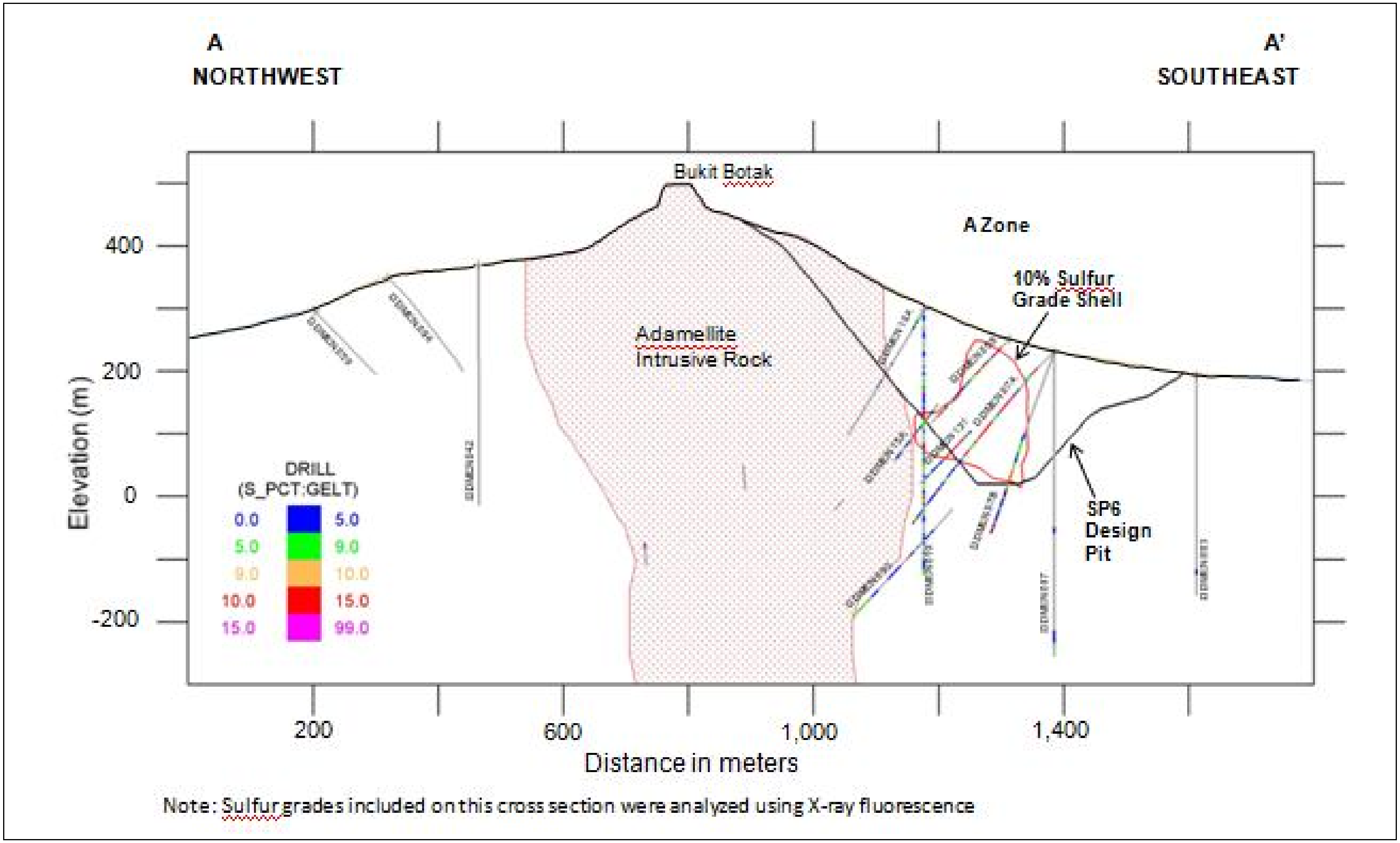
Magnetite is locally abundant in soil and gossan as both fine grained crystals and/or a fine to coarse-grained gravel and cobbles; however, iron was not routinely analysed in the historic drill hole assay samples completed by MMC.

Figure 7.5 Cross-Section A-A' Showing Cu grade (%)



Note: Copper grades included on this cross section were analyzed using atomic adsorption spectrophotometry

Figure 7.6 Cross-Section A-A' Showing S grade (%)



8 Deposit types

The Mengapur mineral deposit is a skarn type deposit. Originally the term skarn was used to describe coarse-grained calc-silicate gangue associated with iron ore deposits of Sweden that included a host of calc-silicate rocks rich in calcium, iron, magnesium, aluminium, and manganese. These were formed from the replacement of carbonate rich rocks. The term skarn is nowadays used to describe deposits like Mengapur which appear to have resulted from the hydrothermal interaction of hot silicate magmas and cooler sedimentary rocks.

There are several different types of skarn deposits that are characterized by the skarn calc-silicate mineralogy, the contained metal(s) of economic interest, and their tectonic setting (Einaudi et al., 1981; Meinert, 1992). Mengapur is best characterized as a copper skarn as it primarily contains economical grades of Cu with much lesser amounts of Au and Ag. The abundance of sulphide minerals is typical of copper skarns mostly in the form of pyrite and/or chalcopyrite. The abundance of pyrrhotite in the skarn, and the targeted extraction of the pyrrhotite to produce a sulphur product, are fairly unique to copper skarns. Pyrrhotite has been documented to be more common in gold skarns with a reduced mineralogy and/or intrusive rock character such as Fortitude, Nevada and Hedley, British Columbia. There are no sulphur skarns defined in the literature.

9 Exploration

9.1 Introduction

All of the resource estimates referred to in this section are historical in nature and have been compiled from the Feasibility Report (Normet, 1990). This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

9.2 Historical exploration

Four main phases of drilling have been carried out at Mengapur to help support the resource and reserve (Normet, 1990). Phase 1 of MMC's drilling was carried out between November 1983 and March 1985 and totalled 49 holes, at a spacing of 140-200 m, for a total of 17,254 m. In 1984, a program of gravity and magnetic surveys was undertaken to assist in the delineation of suitable targets for drilling. 120 line km were traversed at 70 m and 140 m spacing delineating several major conductive zones.

Phase 2 drilling commenced in April 1985 and consisted of 42 holes, at spacing of between 100 m and 200 m for a total of 17,174 m to the end of December 1985. These holes were drilled at 45° to 60° inclination from the horizontal and variable azimuth in order to achieve representative intersections approximating the true width of the mineralised zone. Most of the holes have been drilled to depths of 300 m to 400 m below surface although a few have been drilled to 700 m.

A programme of geological mapping and geochemical soil sampling was carried out to cover a 10 km^2 area at the same time as the diamond drilling was undertaken. The major Cu, Pb, Zn, Bi and Ag anomalies delineated are coincident with the mineralised skarn zones. The major geochemical anomalies were subjected to ground magnetic and time domain EM surveys between April and September 1984. Downhole EM logging was also carried out on 14 selected drill holes in an attempt to determine the geometric configuration of the sulphide body. Minor EM anomalies (weak conductors) were found to be associated with graphitic horizons and black shales.

Phase 3 of the diamond drilling was carried out between April and November 1986 and consisted of 74 holes totalling 17,298 m. The drilling objectives were to close in the drill hole spacing to 70 m from 100 m to200 m spacing in Zones A and B and to 100 m to 200 m from the previous 200 m to 400 m.

The final Phase 4 diamond drilling was carried out between February 1987 and January 1988 and comprised 33 in-fill holes to delineate the higher grade zones in greater detail.

From October 1988 to January 1989, eight oriented core drillholes were completed.

The total number of diamond drill holes completed during the years listed amounted to 221 aggregating 61,052 m.

9.3 Exploration conducted by Monument

Due diligence drilling and on-going data acquisition is underway as part of the Project Due Diligence. Drilling and mineralisation was observed by the authors, but no results have been issued to date.

10 Drilling

10.1 Introdcution

Table 10.1 lists all know historic drilling campaigns.

Table 10.1 Summary of historic drilling and Mengapur

Dates of Drilling	Mining Company	Drill Hole Total	Total Drilling (meters)	Drill Hole Numbers	Drilling Co.	Drilling Method	Reference
After 1962	Jaya Sepakat Mining Company	unknown	unknown	Unknown	unknown	unknown	Lee and Chand (1981)
1979	Geological Survey of Malaysia	4	unknown	CBM7901 to CBM7904	unknown	unknown	Lee and Chand (1980)
August 8, 1980 to March 5, 1981	Geological Survey of Malaysia	11	1,733	CBM8001 to CBM8011	Malaysian Soil Investigation Co. Ltd.	Diamond Drilling	Lee and Chand (1981)
November 1983 to March 1985	Malaysian Mining Corporation	49	17,254	DDMEN002 to DDMEN045, DDMEN19A	Hanover Drilling	Diamond Drilling	James Askew Associates (1990)
April to December 1985	Malaysian Mining Corporation	42	17,174	DDMEN046 to DDMEN063; DDMEN15A	Hanover Drilling	Diamond Drilling	James Askew Associates (1990)
April to November 1986	Malaysian Mining Corporation	74	17,298	DDMEN064 to DDMEN142; DDMEN13A	Hanover Drilling	Diamond Drilling	James Askew Associates (1990)
February 1987 to January 1988	Malaysian Mining Corporation	33	6,342	DDMEN143 to DDMEN167; DDMEN18A	Hanover Drilling	Diamond Drilling	James Askew Associates (1990)
October 1988 to January 1989	Malaysian Mining Corporation	8	1,250	OCH-1 to OCH-9 (OCH-5 not drilled)	unknown	Oriented Core Drilling (clay imprint method)	Call & Nicholas (1991)
TOTAL		221	61,051			Diamond Drilling	

Notes: Only the DDMEN numbered drill holes drilled from November 1983 to January 1988 were used for resource and reserve calculations by James Askew Associates and Normet (1990).
The total number of drill holes used for the 1990 resource and reserve estimate equals 198 (totalling 58,068 m).

11 Sample preparation, analyses and security

11.1 Sampling methods

The historic drillhole assay records indicate that the bulk of the diamond drill whole samples were originally analysed on 3 m sampling widths. The selected sample intervals were separated by geological units so that only one primary rock unit was included in an assay interval where possible.

Historic information from the JAA report states “field repeats” and "duplicate analysis and standards were run at frequent intervals" which are discussed below (JAA, 1990).

11.2 Sample preservation

The historic core storage building burned to the ground in 2005 and as a result no historic core is available for viewing or re-sampling at this time.

11.3 Density determinations

Bulk density for drill core samples was determined by the water displacement method using the 'SG bottle' technique (Normet, 1990).

11.4 Geological and geotechnical logging

Geological logging data was reviewed by the author. Geology logging included the following main rock types: soil, gossan, adamellite (quartz monzonite), rhyolite, rhyolite breccia, dikes, skarn (garnet skarn and diopside skarn), quartz veins, carbonaceous limestone, shale, slate, and weathered rock. Alteration minerals are also logged using an intensity designation system that is not described.

Geotechnical logging was performed on most drill holes completed by MMC and included core recovery and RQD. A separate oriented core program using the clay imprint method was completed in 1988 and 1989 by Call and Nicholas. This oriented core data was unavailable to the author.

11.5 Independent statement on sampling methods

Snowden was unable to verify historical drilling and sampling practices.

11.6 Sample preparation, analyses,

Historic drill hole sample preparation methods were mentioned in the JAA 1990 report and included in the Normet 1990 report. Assays for Cu, Pb, Zn, Ag, As, Mo, and Bi have been carried out using atomic absorption spectrophotometry (AAS). Gold analyses were completed using fire assay/AAS methods. Sulphur analyses of the diamond drillhole samples were originally not analysed as seen on the original assay sheets. It was not until November 1989 that sulphur was analysed using X-ray fluorescence (XRF).

The primary assay laboratory for the drill hole samples was the MMC Laboratory Services located at Batu Caves near Kuala Lumpur. This is based on assay lab sheets and check assay sheets with the MMC and Batu Caves header identification. It is not known if this assay lab still exists or not.

The detailed sample preparation methods for the diamond drill holes (i.e. initial crushing and later pulverizing parameters) have not been described in the Feasibility report.

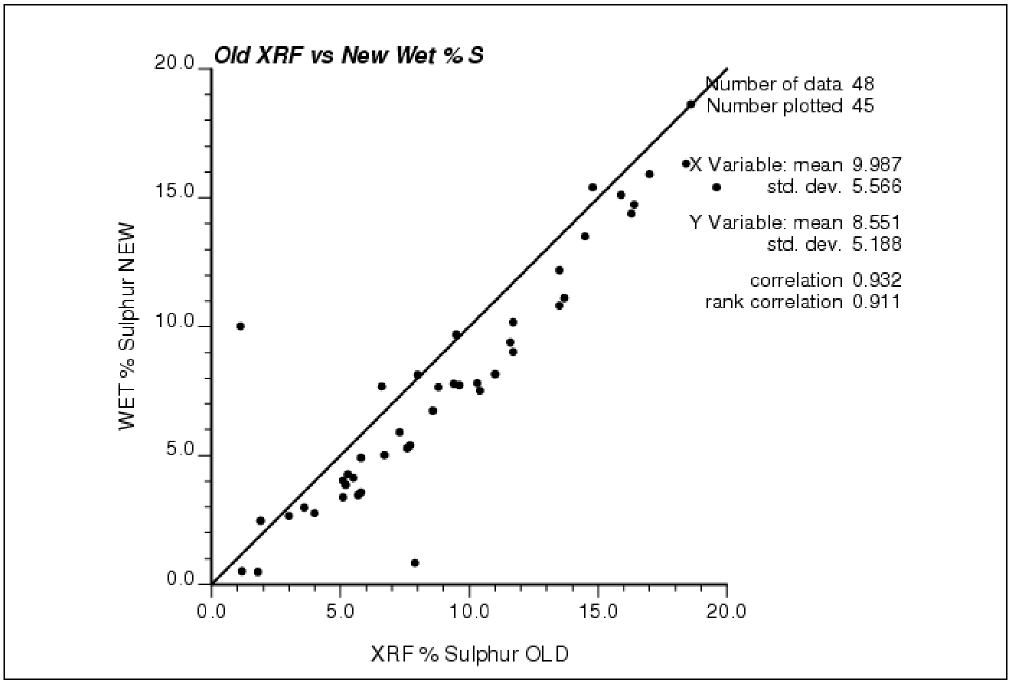
11.7 Quality control measures

The routine insertion of certified standards, blanks, and field duplicates with sample submissions as part of a sample assay QAQC program is current industry best practice, but was not the case historically. Analysis of QAQC data is made to assess the reliability of sample assay data and the confidence in the data used for the resource estimation. Historic quality control measures were briefly reviewed in the Feasibility report (Normet, 1990) and summarized below.

Field repeat (check) samples were routinely conducted for Cu and Ag and other base metals in each of the four main drilling phases from 1983 to 1988. In addition to the resubmission of samples to the MMC laboratory as field checks, both duplicate analyses and standards were run at frequent intervals as a further check on both the accuracy and precision of the assays. No field checks were reportedly run for Au; however, repeat assays reportedly show good assay correlation (JAA, 1990).

JAA note that 50 duplicate drill hole samples were analysed for wet gravimetric sulphur analysis (JAA, 1990), presumably from the MMC Lab. A scatter plot of the data was compiled and the graph is shown below in (Figure 11.1). The graph clearly illustrates the bias of the XRF sulphur results vs. the wet gravimetric sulphur results and this was noted in the JAA report (JAA, 1990). The report indicates that the original sulphur drill hole data were decreased by 15% in grade before they were used in the final resource and reserve calculation. So the sulphur grades reported in the historic resources and reserves in this document should already account for this sulphur analysis bias. Snowden comment that this style of adjustment is not industry best practice.

Figure 11.1 Scatter plot of XRF S% data vs wet lab S% values



Certified standard samples

Certified standard samples are used to measure the accuracy of analytical processes and are composed of material that has been thoroughly analysed to accurately determine its grade within known error limits. Standards are submitted by the geologist into the sample stream, and the expected value is concealed from the laboratory, even though the laboratory will inevitably know that the sample is a standard of some sort. By comparing the results of a laboratory's analysis of a standard to its certified value, the accuracy of the assay results of the laboratory is measured.

Historic data indicates certified reference materials, or standards, whose true values are determined by a laboratory, have been placed into the sample stream at Mengapur to ensure sample accuracy throughout the sampling process. The JAA (1990) confirm that standards were used. However, no complete standard data compilation has been reviewed by Snowden and there has been no independent verification of this process.

Snowden recommends Monument utilize a rate of standard sample submission to achieve the prescribed rate of 1 in 20 samples, with preference given to insertion of standards within mineralised sample intervals.

Blank samples

Field blank samples are composed of material that is known to contain element grades that are less than the detection limit of the analytical method in use, and are inserted by the geologist in the field. Blank sample analysis is a method of determining sample switching and cross-contamination of samples during the sample preparation or analysis processes. Historic reports indicate that blanks were utilized at Mengapur. The author has no independent verification of this practice.

Duplicate drill core samples (field duplicates)

Historic data indicates no field duplicate checks were utilized but field checks were run at frequent intervals for other assays.

Umpire laboratories

Umpire laboratories were utilized for the Mengapur Project. Eight of the diamond drill hole assay samples were sent to other overseas commercial laboratories for check analyses for Cu, Pb, Zn, Mo, Bi, Ag, Au, and As (Normet, 1990). The assay labs that were used include: Charter, Chemex, Amdel, LNETI, and Australian Assay Laboratories (AAL) in Perth Australia (Normet 1990). Some of the samples that were metallurgically tested were also analysed at different laboratories. Snowden believes that more of this work needs to be documented at Mengapur in the future.

11.8 Independent statement on sample preparation, analyses, and security

Snowden comment that historic sample preparation and security of diamond drill core samples for Mengapur cannot be verified at this time.

Drillhole core from previous Mengapur drilling campaigns are unavailable for review as the drillhole core storage facilities reportedly burned down in 2005.

12 Data verification

12.1 Data compilation and verification by Snowden

Re-sampling of drill core

Due to the loss of historic core, re-sampling is not possible at Mengapur.

Twin drillholes

No documents exist of any twin drillholes at Mengapur. Monument are currently drilling holes to confirm grade and interpretation estimates from the Feasibility Study.

12.2 Independent data verification

Independent site inspections

Mr. Roderick Carlson of Snowden conducted site inspections of the Mengapur project in July 2011 The site visit was general in nature and he undertook the following activities:

- review of geologic model
- inspection of on-going drilling and core
- review of on-going drill sampling and logging
- inspection of current core security procedures
- site geology review at site outcrops
- review of mill facilities (grinding and flotation).

Independent sampling of mineralised intersections

Independent samples are taken to verify the presence of mineralised intersections. Due to the absence of any historic core from drillholes this was not possible.

Independent review of drillhole collar coordinates

In the July, 2011 Snowden site visit to Mengapur no historic drillhole collars were inspected, however several drillhole collar markers (cement caps and pvc pipes) were observed by Mr. Todd Johnson, an independent Qualified Person (QP) as defined by NI 43-101, during his site inspection conducted in October 2011.

Independent review of original assay certificates

Historic assay certificates were not viewed by the author.

13 Mineral processing and metallurgical testing

13.1 Introduction

The author has taken this entire section for mineral processing and metallurgical testing from the MMC (1993) Feasibility Study report. This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

Based on the results of studies and test work completed to date by MMC and various consultants, the most viable proposal would be to mine the bulk of sulphide ore of Zone A and the supergene high grade leachable oxide ore which has to be removed before the sulphide ore could be extracted. Approximately 2.75 Mtpa of the sulphide ore will be treated, whilst the high grade leachable oxide ore will be treated during the first 10 years. 80,000 tpa to 520,000 tpa of the high grade leachable oxide ore would be processed during the first three years of operations.

The sulphide ore will be treated by conventional flotation method to produce copper and pyrrhotite concentrates whilst the high grade leachable oxide ore will be heap leached using sulphuric acid and further processed by cementation to produce high grade Cu cement which can be blended and sold with copper concentrates.

Reagent Dosing

The barren solution pond would continually be monitored for free acid content.

Sulphuric acid would be stored in the reagent mixing area adjacent to the barren solution pond. Acid would be dosed continuously to the barren solution pond at the required dose rate to maintain column leaching conditions.

Services

Only a small amount of make-up water would be required due to the high rainfall in the area. Make up water would be available from the tails dam via the process water system. The stormwater pond has been designed to contain a 1 in 50 year storm event. Any excess solution contained in this pond can either be recycled during lower than average rainfall periods or sent to the main tailings dam.

Flowrate metering and totalising of the rain water, pregnant liquor and barren liquor would be monitored by impeller flowmeters and "v" notch weir boxes.

A potable water storage tank and two integrated safety shower/eyewash stations would be supplied at the reagent addition point.

Power for the pumping at the heap leach and pond areas would be provided from the central power generation unit.

14 Mineral Resource Estimates

14.1 Disclosure

The historic Mineral resource cited in this report in this section was prepared by JAA (1990). This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

14.2 Known issues that materially affect the Mineral Resources

Snowden is unaware of any issue that may materially affect the Mineral Resources in a detrimental sense.

14.3 Assumptions, methods, and parameters

All data, assumptions, methods, and parameters utilized for the Mengapur Mineral Resource estimations are from Mineral Resource report (JAA, 1990) and have not been independently verified by Snowden.

Copper Equivalent Calculation

The cut-off grade assumptions utilised by JAA (1990) include the use of a copper equivalent (EQV Cu). The assumptions for the calculation of the EQV Cu are shown in Table 14.1

The starting point in the calculation is the market price of each commodity, which then has various costs and recoveries applied to arrive at an 'equivalence factor' which enables a copper equivalent sample grade or orebody model block grade to be calculated (Table 14.1).

The market price of each commodity has the direct metallurgical operating costs applied to establish a marginal commodity value per tonne or per gram. The direct costs incurred are those associated with the milling and processing of the ore. Overhead costs associated with recovering each commodity are then deducted to give the net commodity value per tonne or per gram. The mill recovery for each commodity can then be applied to give the net recovered commodity value. If the net recovered commodity value is then expressed in terms of the net copper price, then a copper equivalent grade can be calculated for each commodity. It is then possible to factor the sample or block model assays to give an equivalent copper grade (Table 14.1).

The cut-off grade for the Mengapur Project has been calculated by dividing the ore treatment processing and overhead costs by the net copper price. Using the cost estimates calculated on 21 July 1990, the project cut-off grade is 0.336 EQV Cu.

NOTE: These are historic price assumptions that do not reflect current prices or costs.

Table 14.1 Copper conversion and equivalent factors

Commodity	Marginal commodity value (US$)	Net commodity value (US$)	Marginal price per kg (US$)	Net commodity price per kg (US$)	Mill recovery (%)	Equivalent	Factor
Cu (%)	1,373.79	1,338.39/t	1.37379	1.33839	76.6	0.766000	x Cu %
Au (g/t)	4.11	3.417/g	4107.00	3417.00	47.0	0.119994	x Au g/t
Ag (g/t)	0.0658	0.055/g	65.00	55.00	48.0	0.001973	x Ag g/t
S (%)	97.39	94.79/t	0.09739	0.09479	82.0	0.058076	x S %

Mining Cost	US$0.731 /t
Ore treatment cost (crush, grind, float)	US$3.010 /t
Incremental cost of copper recovery	US$1.050 /t
Marginal cut-off grade for EQV Cu = processing cost/net metal price/10kg = $4.060 divided by $13.74	0.296 EQV Cu
Ore processing overhead costs	US$0.436 /t
Total Cost (Processing + overheads)	US$4.496 /t
PROJECT Cut-off grade for EQV Cu =	
processing cost/net metal price per 10kg Cu $4.496 divided by $13.38	0.336 EQV Cu

14.4 Supplied data, data transformations, and data validation

Supplied data

The Resource Report (JAA, 1990) states: The diamond drill holes were logged, sampled and assayed and the data were then entered into computer using Geology System pro-forma. Subsequently, the database was transferred to Datamine software for resource evaluation to be carried out.

The drill hole assays were routinely conducted at the MMC Laboratory at Batu Caves located near Kuala Lumpur in Malaysia (Normet, 1990). Most of the original drill hole paper geology logs, geotechnical logs, and lab assay sheets have been scanned and are in the possession of Monument. The geology and geotechnical drill logs are labelled with the drill hole coordinates (northing, easting, and elevation in Cassini grid) and azimuth and dip at the collar. Much of the drill hole data was collected from Malaco in an excel spreadsheet with the file name called “MenDDMMC1991.xls.” Only 45 diamond drill holes have a complete photographic record. The core shack at Mengapur was reported to have burned down sometime in 2005 so there is no old drill core to review.

Data preparation

It is unknown to the author if any data preparation was performed by JAA on the data or the models.

Data transformation

No transformations or rotations have been performed by JAA on the data or the models.

Data validation

It is unknown to the author if any validation checks were performed by JAA on the data or the models.

14.5 Geological interpretation, modelling, and domaining

Drill hole cross-sections were plotted showing copper grades and rock type. Mineralised zones for each rock type at a lower copper cut-off grade of 0.05% Cu were delineated.

Details on the resource evaluation and the block parameters are described in JAA (1990). In brief, resource modelling is based on primary blocks measuring 50 m x 40 m in plan by 10 m vertical thickness. The software provides automatic sub-division of the primary blocks to effect more accurate modelling of the outlines of the rock types and mineralisation. Figure 14.1 and Figure 14.2 depict the ore block models along section line 18 and on the 180 mRL bench plan at various pit design limits.

Figure 14.1 Ore block models along section line 18

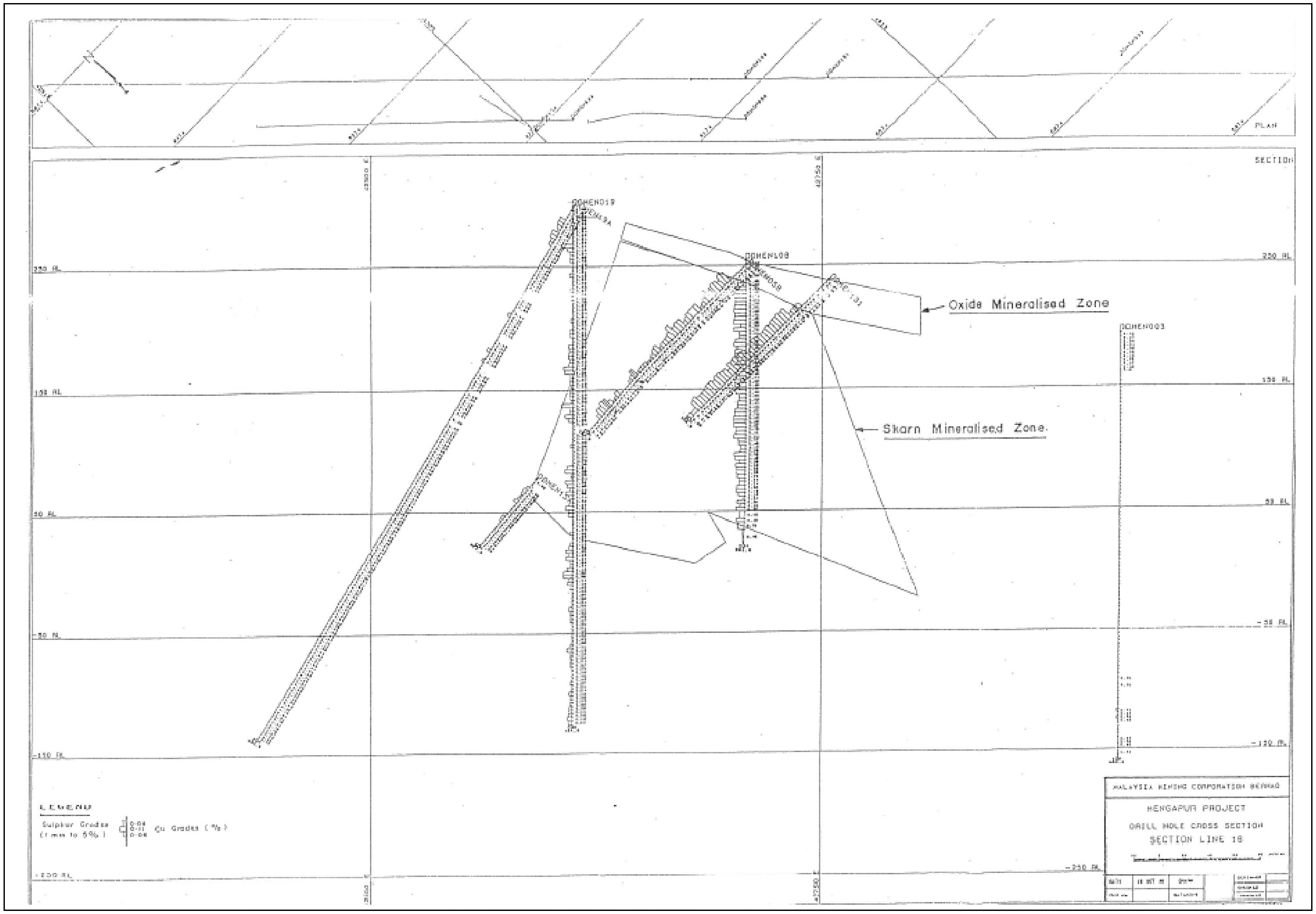
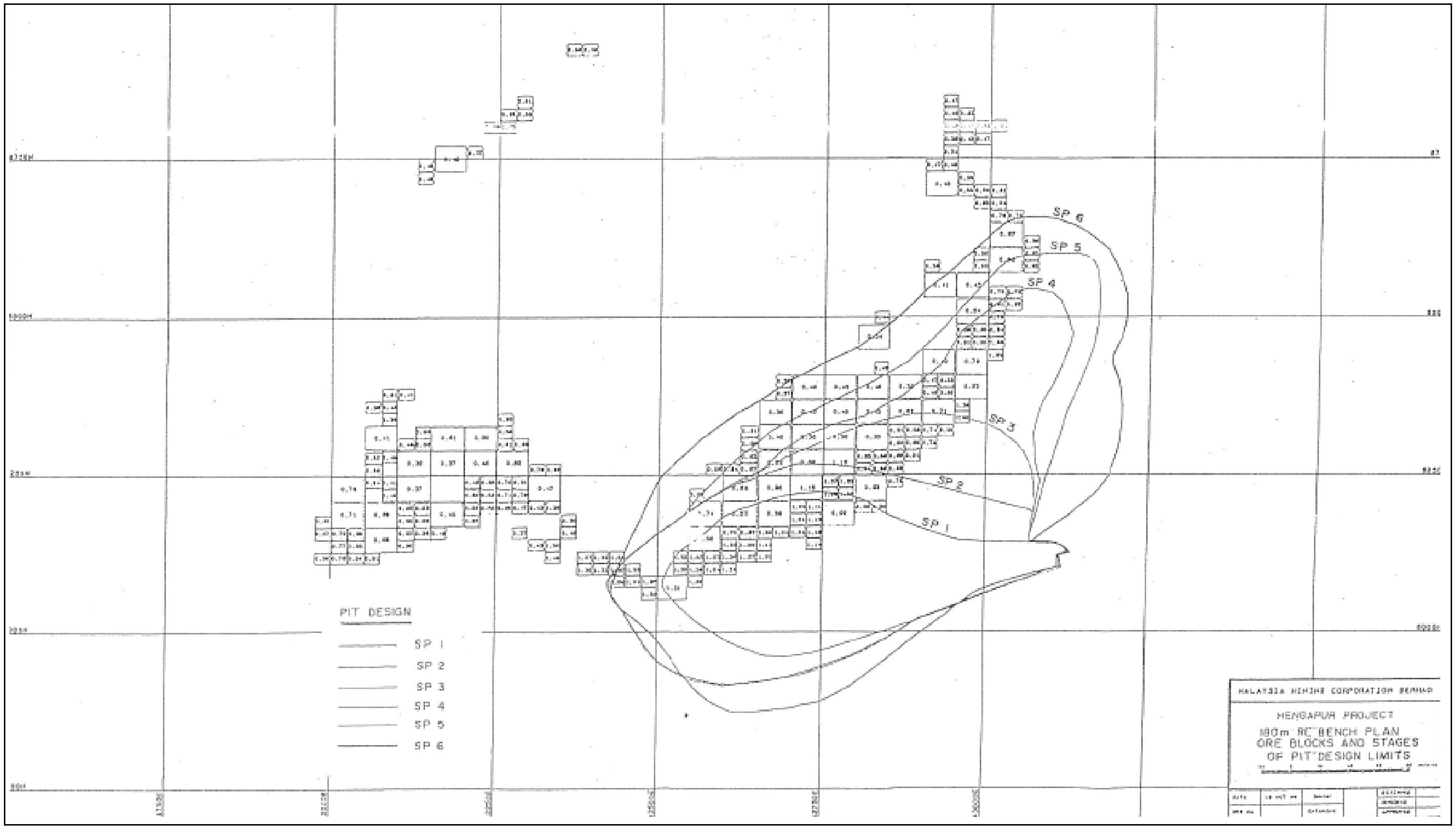


Figure 14.2 Ore block models in RL bench plan at various pit design limits



14.6 Sample statistics

Sample compositing

Compositing parameters used by JAA for the Resource Estimation are unknown to the author.

Core recovery treatment

Drillhole core recovery impacts on estimation are unknown to the author.

14.7 Extreme value treatment

It is unknown to the author what if any top cutting strategy was employed by JAA to complete the Resource Estimation.

Data declustering

It is unknown to the author if any declustering of the data was performed by JAA for the Mineral Resource Estimation.

14.8 Variogram analysis

Variogram Modelling

Estimation of block grades in both sulphide and oxide mineralised zones has been carried out based on inverse distance weighting taking into account the search ellipse orientation and anisotropy factors that are derived from the geostatistical analysis. The review based on 3 m composites is shown in (Table 14.2).

Table 14.2 Variogram Parameters for Zone A and Skarn

Element	Direction	c0	c1	a1	c2	a2
Copper	Downhole	0.018	0.043	200		
	Isotropic	0.018	0.070	180		
	Strike	0.018	0.070	150		
	Dip	0.018	0.048	300		
	Orthogonal	0.018	0.038	100		
Sulphur	Downhole	9	6.5	10	20.0	180
	Isotropic	9	8.0	30	15.0	230
	Strike	9	8.0	25	15.0	280
	Dip	9	10.0	150	11.0	300
	Orthogonal	9	6.0	35	17.5	120

Key:
c0 – Nugget Variance;
c1 - differential sill variance 1st structure;
a1 – Range (m) 1st structure;
c2 – differential sill variance 2nd structure;
a2 – range (m) 2nd structure;
Downhole – west dipping holes only. 6m lag;
Isotropic – all directions averaged. 25m lag;
Strike 45° bearing, 0° dip, 22m lag;
Dip – 135° bearing, 85° dip, 25m lag;
Orthogonal 315° bearing, 5° dip, 25m lag;

14.9 Estimation parameters

Sample search parameters and grade Interpolation

From the JAA report the search parameters used in the Resource Estimation are shown in Table 14.3.

Table 14.3 Block grade interpolation parameters (Gillett et al., 1990)

Zone	Oxide			Skarn		
	A	B	C	A	B	C
Search radius	180	180	180	180	180	180
Dip of Axis 1	30	30	0	60	60	60
Azimuth of Axis 1	135	225	0	135	225	0
Relative length of Axis 1	5	5	5	5	5	5
Relative length of Axis 2	5	5	5	5	5	5
Relative length of Axis 3	1	1	1	1	1	1
Power	2	2	2	2	2	2
Minimum No. Of points	1	1	1	1	1	1

14.10 Bulk density

Specific gravities have been applied to the model blocks within each rock type zone based on the weighted average of the specific gravity determinations on the drill core samples for each rock type. The following values were used: soil waste at 2.0 g/cc; soil ore at 2.2 g/cc; skarn waste at 3.0 g/cc; skarn ore at 3.3 g/cc; and all limestone, adamellite and rhyolite at 2.8 g/cc. For the purposes of density determination all material within the 0.05% Cu outlines is considered as ore.

14.11 Estimation evaluation

The resource estimated for Zones A, B and C has been evaluated at a range of different EQV Cu cut-off grades. The data presented in Table 14.4 and Table 14.5 have been produced at a cut-off grade of 0.336% EQV Cu reflecting the available information in 1990 commodity prices, operating costs and plant recoveries. Other elements present in the ore are Pb, Zn, Mo, As, Bi, Sn and W. However, none of these are considered to contribute to the value of the ore and hence do not influence the calculation of the cut-off grade.

Table 14.4 Total Mengapur historic measured and indicated resources within Zones A, B, and C

		Tonnes (Mt)	EQV Cu (%)	S (%)	Cu (%)	Au (g/t)	Ag (g/t)
OXIDE	Measured	4.866	0.419	0	0.47	0.05	27.82
	Indicated	16.406	0.557	0	0.64	0.12	26.45
	Sub-total	21.272	0.525	0	0.6	0.1	26.70
SULPHIDE	Measured	63.438	0.661	7.622	0.25	0.18	3.3
	Indicated	139.699	0.579	7.04	0.19	0.13	3.85
	Sub-total	203.137	0.605	7.222	0.21	0.15	3.68
TOTAL		224.409	0.597	6.54	0.25	0.16	8.86

The Measured Resource comprises 30% of the total and 9% of the total is oxide material.

There is a vast resource of low grade oxide ore in the Mengapur deposit, particularly in Zone C. At a copper cut-off grade of 0.20%, the measured and indicated oxide resource evaluated amounts to 72.5 Mt averaging 0.32% Cu.

For the computation of mineable ore reserves, Lerchs Grossmann 4-D pit optimisation was carried out on Zone A, which lies to the southeast of the ridge.

Table 14.5 Total Proven and Probable HIstoric Reserve contained within the SP6 optimized pit limit (Zone A)

		Tonnes (Mt)	EQV Cu (%)	S (%)	Cu (%)	Au (g/t)	Ag (g/t)
SULPHIDE	Proven	26.467	0.803	9.2	0.31	0.25	2.46
	Probable	38.324	0.691	8.23	0.24	0.19	2.68
TOTAL		64.8	0.737	8.67	0.27	0.21	2.59

The Proven Reserve comprises 41% of the total ore reserve. This total excludes the oxide material which cannot be deemed to be a Proven and Probable Reserve until such time as the metallurgical recovery can be accurately assessed. Total oxide resource contained within the SP6 pit design is 4,973,000 t grading 0.787% EQV Cu using the same cut-off of 0.336% EQV Cu as applied to the sulphide skarn resource.

A supergene enriched zone has been noted immediately above the sulphide orebody, particularly in Zone A. In the enrichment zone, the re-deposition of copper as simple sulphides (chalcocite, digenite), silicates (covellite), oxides (cuprite) and as sulphosalts is probably caused by pH changes in the percolating leach solution as it moves down the soil profile. A fairly distinct concentration of silver and bismuth values has also been noted together with the copper enrichment. The thickness of the supergene enrichment zone varies from 3 m to 9 m and may grade as high as 17% Cu (e.g. from 36 m to 42 m in Hole MEN135, 48 m to 51 m in MEN013, and 30 m to33 m in MEN015). In most cases, however, it grades only one half to a few percent copper. The major portion of the supergene ore is located at the upper end of the sulphide orebody. However, displacements probably by creeping/slumping have resulted in a portion of supergene ore being displaced to lower levels further -away from the adamellite intrusive and distinctly dislocated from the sulphide orebody.

It is difficult to define the boundaries of the supergene enrichment zone based on mineralogical logging or chemical assays. Diagnostic leaching was therefore carried out in small rolling bottles under standardised conditions. Overall, the leach tests are believed to give an acceptably accurate measure of recoverable copper by heap leaching. Leachable ore reserves of oxidised ore are based directly on the diagnostic bottle roll tests. This ore has therefore been referred to as "High Grade Leachable Oxide", or HGLO ore. As has been noted above, it approximates to the supergene ore zone.

Table 14.6 Total High Grade Oxide reserve within the SP6 pit design

Tonnes (t)	Cu (%)	Au (g/t)	Ag(g/t)
2,344,000	1.294	0.233	32.5

14.12 Mineral Resource classification

The Mineral Resource confidence classification of the Mengapur resource estimate has incorporated several factors, such as the confidence in the accuracy of the drillhole data, the availability of specific gravity measurements, the level of geological interpretation, geological continuity, data density and orientation, spatial grade continuity, and estimation quality.

The portion of the resource model where there was sufficient confidence in the estimate was classified as an Inferred Resource in accordance with the CIM classification standards (2005).

14.13 Mineral Resource Reporting

All mineral Resources and Reserves have been taken from the JAA (1990) and Normet (1990; 1993) report and are considered historical in nature and do not comply with current (2005) CIM guidelines.

15 Other relevant data and information

The author is unaware of any other relevant data for this report. This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

16 Mining methods

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

From the Mengapur Project Feasibility Study Report (MMC, 1993):

The salient features of the pit slope design, the mine design, the mine production schedule and the capital and operating cost estimates for Zone A are presented. Details are available in Volume 10 of DFS by Normet (1990).

MMC commissioned Call & Nicholas Inc (CNI) to undertake the following studies:

- Geotechnical assessment.
- Recommendation of slope design to be incorporated in the pit design.
- Design and costing of the pit dewatering system.

MMC has prepared the pit designs and mining schedule based on the orebody model described in pit optimization section and the slope designs recommended by CNI. On behalf of Normet, JAA has reviewed MMC's work and found it to be soundly based and executed. More detailed mine planning would, however, be required prior to commencement of mining, based on an actual ground survey.

Based on data supplied by MMC, combined with the results of its own investigations, JAA has determined equipment and manpower requirements and prepared capital and operating cost estimates for the mining operation.

The Mengapur sulphide resource is to be mined using the open pit method.

The bulk of the sulphide mineralisation occurs within the skarn, which is the subject of this study. The mineralisation within the adamellite and rhyolite is of limited extent and has therefore not been included in ore calculation.

The upper portion of the deposit has been oxidised from the fresh sulphide rock to form soils, hosting significant oxide mineralisation. These oxidised ores are divided into two types: high grade leachable oxide and low grade oxide ore.

The high grade leachable oxide ore, defined by its recoverable grade in a standardised leaching test, will be heap leached with weak sulphuric acid.

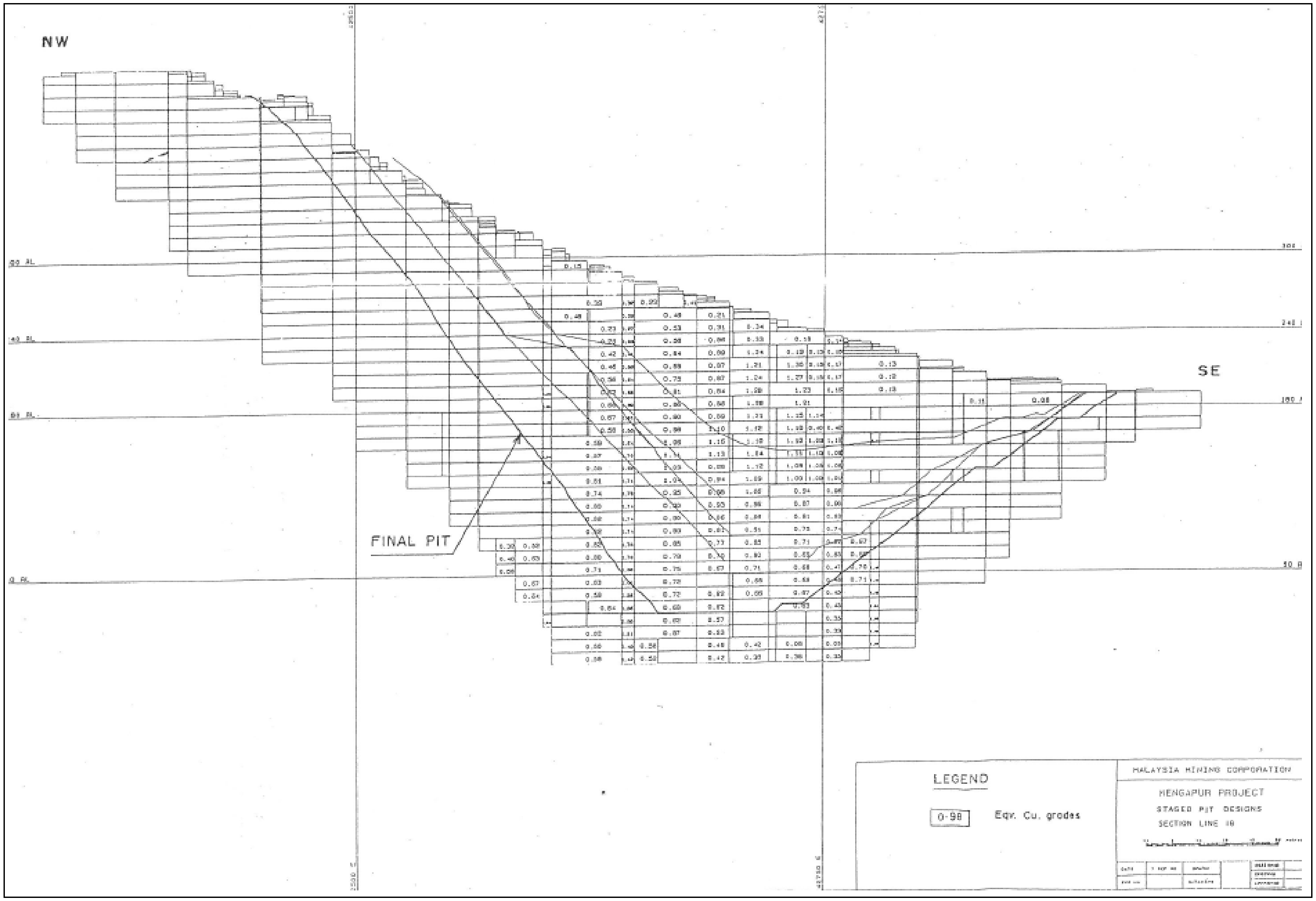
Oxide ores with a low recoverable copper grade (again as defined in the standardised leaching test) will not be treated, but will be dumped separately for possible treatment later. Until the economic treatment of this ore has been confirmed, no significant additional assaying of blast hole samples will be required and the low grade oxide will be excavated based on definition used in the exploration drilling, with check samples being taken as part of the in-pit grade control activity.

The skarn mineralisation occurs as a relatively massive deposit and therefore highly selective techniques are not required when extracting the ore. This allows some economies of scale to be enjoyed. For example a 10 m bench height can be maintained throughout all ore and waste.

The sulphide skarn deposit lies at the base of the southern side of Bukit Botak. In order to mine the resource it is necessary to cut back the southern face of the hill. Mining will extend from a maximum elevation of 430 m RL high up on the hill down to a final pit bottom at 20 m RL.

The development of the Mengapur pit is staged, using interim pits to spread waste stripping more evenly and to provide timely access to ore (Figure 16.1).

Figure 16.1 Staged pit designs



Of the total tonnage mined, 58% comes from above the daylight elevation at 180 RL. This material will be accessed by means of a 15 m wide ramp developed up the hillside. This ramp will provide access for the initial development stage and the first cut-back. The second cut-back will mine this ramp out as it progresses down the slope. In order to re-gain access for the final cut-back, a new ramp will be required. This will be developed outside the limits of the final pit outline.

In Year 3, mining advances below daylight, i.e., a pit is formed as opposed to mining into the side of the hill. Access to the pit bottom is via an in-pit ramp which will be developed on the south wall of the pit. Thereafter, for most of the project life, mining will consist of simultaneously cutting back the hillside and deepening the pit.

Due to its heavily weathered nature, the majority of the soil is expected to be free dig, or at worst require light ripping. However in some areas of soil, large boulders can occur and these will require blasting and accordingly an allowance has been made for blasting 20% of the soil.

The contact between the weathered soil and the fresh rock beneath is usually distinct. Below this contact, the rock is competent and full drilling and blasting will be required throughout this fresh rock. Blasts will be designed to produce broken material with a fragment size less than 1.3 m, the maximum feed size for the primary crusher and a size commensurate with the scale of loading and hauling equipment. In order to minimise the need for secondary breaking, a high powder factor with relatively close hole spacing was selected. Rock blasting will be on a 3.5 m by 3.5 m spacing with a powder factor of 0.3 kg (ANFO)/tonne.

16.1 Grade control

Grade control will be based on the sampling of every second blasthole. The drill chippings for each sampled hole will be split down in the field to a 2-3 kg sample.

An allowance has been made to take samples at this density throughout the deposit. Additional drilling has been included to collect samples in the free-dig material. Once in routine production mode, it is not anticipated that areas of bulk waste will be sampled to this density. The grade control effort allowed for waste sampling will be re-directed to collect additional samples at the ore boundaries, particularly the oxide/sulphide contact zone. In addition, some samples will be taken from the oxide resource as a check on the exploration data.

Whilst no ore is produced during the pre-strip period, it is important to conduct grade control sampling in advance of ore mining. It is during the early years that an enhanced understanding of the orebody will be gained, and grade control procedures refined. The number of grade control samples taken peaks at 14,500 pa in Year 1.

Geologists will conduct in-pit mapping of geological contacts and structural features. The grade control assays will be combined with in-pit geological mapping to determine the ore/waste boundaries. This in-pit mapping will form an important part of the grade control process as the skarn is visually distinguishable from the surrounding rocks.

During the grade interpolation process used to construct the orebody block model on computer, low grade samples were used to incorporate dilution in the orebody model. Since dilution is included in the orebody model, a mining recovery of 100% of the diluted resource is anticipated.

16.2 Loading, hauling and dumping

All mining will be on 10 m benches using hydraulic face shovels with 3.8 m^3 buckets to load rigid frame trucks of 40 t capacity.

Operations will be on a 3 shift, 7 day-week basis. Ore will be mined on all shifts.

Most of the ore will be dumped directly from the haul truck into the crusher. A stockpile will be required adjacent to the crusher to protect the mine from delays caused by crusher stoppages and to protect the mill from interruptions to mining, such as during heavy rain. The stockpile will also be used to smooth out any peaks in the ore tonnage delivered to the crusher.

The waste will initially be dumped to the south of the pit; once this dump reaches its capacity, after about 15 years, a new dump will be formed to the north. The waste will be laid down in 20 m lifts. Dozers will be used to finish the dump faces at a gradient of 1:3. The slopes will be track compacted and then re-vegetated to stabilise them.

The mineralised oxide material will be dumped at the north-eastern end of the southern waste dump. This position is selected as being close to the concentrator, treatment plant in order to minimise the reclamation costs. The stockpile and waste dump will be built simultaneously. Leach pads for the high grade oxide ore will be sited immediately to the north of the concentrator plant.

The streams which currently run through the area designated for the waste dump will be diverted around the dump site. Run-off water from the waste dump will be collected and channelled to sedimentation ponds before being allowed to join the main diversion channels.

16.3 Water management and pit dewatering

The drainage from the area upslope of the pit will be diverted around the pit. Initially it will be possible to gravity drain the mining area but as mining progresses below 'daylight', water will be managed by a system of sumps and pumps. Any water flowing through areas of mining activity will have an increased solids load and so will be passed through sedimentation basins.

It is envisaged that no dewatering of the rock slopes will be required. However, dewatering holes will be used to drain the soil slopes.

16.4 Pit optimization

For the mining studies done as part of the DFS, MMC subjected the resource model to a Lerchs-Grossmann pit optimisation using Whittle Programming Pty Ltd's Four-D package. This process involves assigning a dollar value to each resource model block. This value is calculated by subtracting the operating costs attributable to that block from the revenue generated by that block. Therefore a waste block which generates no revenue has a negative value equal to the costs incurred in mining that waste. Ore blocks may have a negative or positive value depending on the relative sizes of their costs and revenues.

For optimisation, overall slope angles are defined and these angles may vary in different areas and at different orientations. According to the various slope angles, interdependencies are built between model blocks, thereby ensuring that the slope design constraints are not breached in accessing ore blocks located at depth.

The optimisation process identities the sets of interdependent blocks which add together to produce the largest net value. It is these blocks which constitute the optimal pit.

However, there is no one optimal pit for a reserve as the value of blocks is dependent upon many economic and operational parameters, such as operating costs, commodity prices and treatment plant recoveries. The pit optimisation process calculates the optimal pit for one particular set of parameters.

For the Mengapur pit optimisation, the Whittle Programming Pty Ltd's Four-D package uses a "metal cost of mining" concept to combine costs and commodity prices into one variable. In the optimisation process, a range of "metal cost of mining" values can be used to produce a set of nested pits. The smallest being optimal at low commodity prices or high costs, through to the largest which would be optimal at high commodity prices or low costs.

Each nested pit corresponds to a different cut-off grade. The pit where the cut-off grade equals the project cut-off grade is the optimal pit under the conditions set for project study.

16.5 Equivalence factors

In order to evaluate a polymetallic deposit, a measure of value incorporating all contributing elements must be developed. In the case of Mengapur, a "recovered copper equivalent grade" has been selected. The grade of each element is converted to a grade of recovered copper of equivalent value using an equivalence factor. The formula for the equivalence factor for any element is given below:

$$Commodity\ Equivalence\ factor = \frac{Commodity\ Price\ x\ Metallurgical\ Recovery}{Copper\ Price}$$

The grade of a commodity is then multiplied by its equivalence factor. In this way the recovered copper equivalent of each commodity is calculated and then summed to give the net equivalent copper grade (EQV Cu) (Table 14.1).

The Mengapur case is complicated because the costs through the treatment process vary significantly from product to product. In order to solve the problem of different treatment costs, the common processing costs are separated from the product specific costs. To determine the respective commodity, value, the point of sale is considered to be the cal of the common treatment processes. The net commodity price at that point is the market price less the commodity specific costs.

16.6 Project breakeven grade

The project cut-off grade can be considered to be the grade at which the revenue/gained by treating material is equal to the costs incurred in treating that material. For the purposes of cut-off grade determination, all overhead costs have been included with the common processing costs. Since recovered copper equivalent grades, common processing costs and the net copper price are being used the project breakeven grade can be defined as below:

Project Breakeven Grade = Common Processing Costs/Net Copper Price

Using the estimates of costs, recoveries and commodity prices, the project breakeven grade calculates to be 0.336% EQV Cu. For the pit optimisation, a cut-off grade of 0.336% EQV Cu was taken as the project breakeven grade.

16.7 Pit optimization parameters

The mining costs for the operation were estimated with rock carrying a higher cost than soil in order to reflect the drill and blast requirement. Mining costs mill be higher for material coming from above the crusher elevation, which reflects the effort required to truck material down the hillside. Costs will also be higher for material coming from below the "daylight" elevation, reflecting the increased haul times to bring material out of the pit.

The treatment plant cost used, US$3.44/t of ore, is the cost incurred in producing a concentrate that is the common processing cost. This cost includes the common processing overhead costs and a mining overhead cost of US$0.152/t of ore.

The slope designs specified by CNI were used to construct sub-regions suitable for input to the Lerchs-Grossmann pit optimisation. The parameters used for the pit optimisation are summarised in (Table 16.1). Mining costs are shown in Table 16.2.

Table 16.1 Parameters used for Lerch Grossman optimisation

Equivalence Factors	
Copper	0.766000
Sulphur	0.058076
Gold	0.119994
Silver	0.001973

Table 16.2 Mining Costs

Bench	Soil ($/t mined)	Rock ($/t mined)
450	0.84	1.13
400	0.80	1.10
350	0.77	1.06
300	0.73	1.02
250	0.73	1.02
200	0.73	1.02
150	0.80	1.10
100	0.88	1.17
50	0.95	1.24
0	1.02	1.32

Treatment Plant Cost US$3.44/tonne of ore (including overheads).

The slope design recommendations were rationalised to a form suitable for input to the Lerchs-Grossmann optimisation. Six sub-regions were used with overall slope angles varying between 30° and 50°.

A check on the mining cost estimates was performed at the end of the study. The mining costs used for the optimisation were applied to the material within the mineable ore reserves which produced a weighted average mining cost of US $1.14/t of material mined, including overhead costs. This compares closely with an average cost of US$1.13/t estimated during the study.

16.8 Pit optimization results

The Proven and Probable Reserves within the optimal pit, without access roads, at a cut-off grade of 0.336% EQV Cu is summarised in (Table 16.3).

Table 16.3 Reserves within the optimal pit

Reserves within the optimal pit				
Tonnes (000t)	**EQV Cu (%)**	**Waste (kt)**	**Total Material (kt)**	**Strip Ratio**
64,800	0.737	69,485	134,285	1.1

In designing the final pit, the optimization was used as a guide, but the CNI slope design recommendations were strictly adhered to.

16.9 Pit design

Whilst the final pit design is a valid and achievable one, before mining commences the optimisation will be repeated. Once the slopes are correctly represented, optimisation will be used to refine the final pit design. The current pit design is considered by JAA to be conservative with respect to the net value of the material contained within the pit and these future refinements will serve to raise the net value of the final pit.

The development of the Mengapur pit will be staged with five interim pits being designed within the final pit, which have been designated as Staged Pit 1 (SP1) to Staged Pit 6 (SP6) with SP6 being the final pit itself (see Volume 10 of DFS Report).

In designing the series of staged pits, the maintenance of a minimum mining width has been considered, while ensuring that access to the next stage is maintained.

The SP6 design (Figure 16.2 and Figure 16.3) cuts back into Bukit Tembaga at the 430 mRL bench and the design wraps itself around the hill going to the north-east. A pit is formed as mining goes below the 180 mRL where an in-pit ramp will exist half-way along the southern wall of the pit. This ramp progresses into the pit at a width of 20 m and a gradient of 1:10; the ramp makes three traverses of the south wall before reaching the pit base at the 20 mRL.

Figure 16.2 Staged Pit 6 (SP6) Design

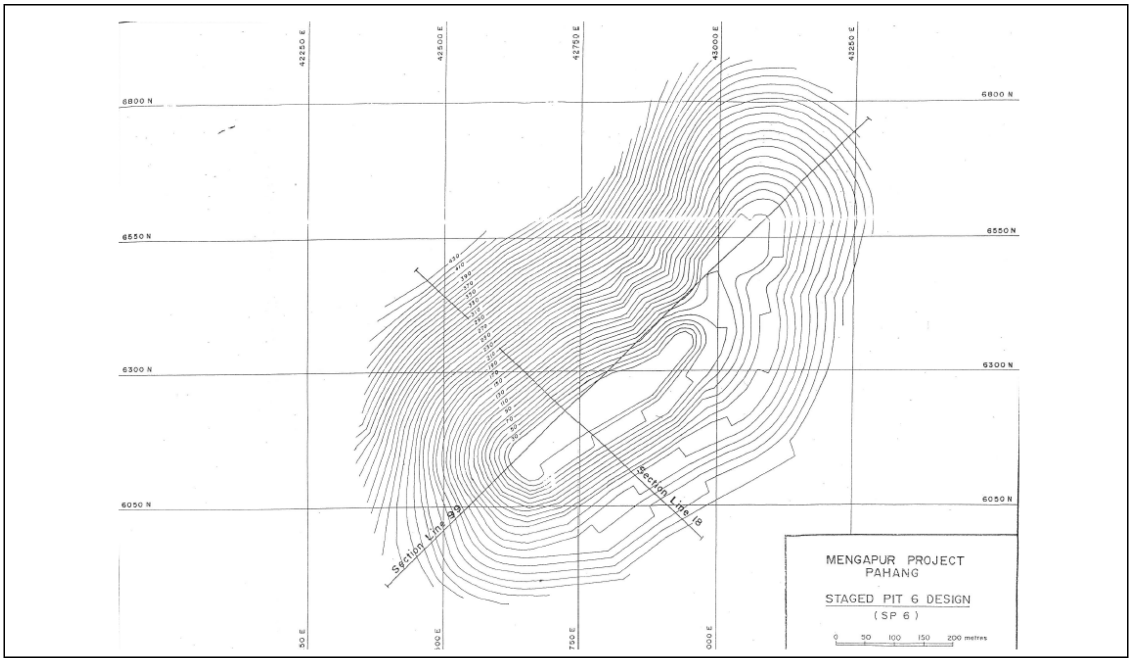
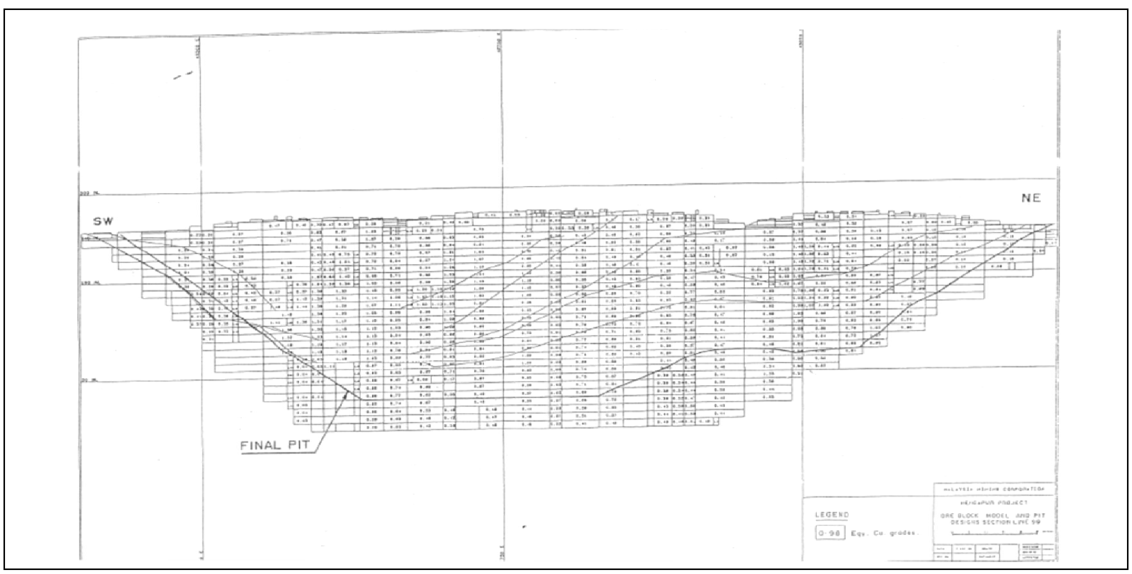


Figure 16.3 Ore Block Model and Pit



16.10 Mineable Ore Reserves

The SP6 pit design was evaluated against the orebody model at a cut-off grade of 0.336% EQV Cu to generate the mineable ore reserve estimate, summarised in (Table 16.4).

The final pit also contains some of the oxide resource. Further metallurgical testwork will be required before a set of equivalence factors and a cut-off grade can be determined. In order to provide a guide to the tonnages and grades involved, the resource was evaluated using the same equivalence factors and cut-off grade as were used for the sulphide resource. The oxide resource within the final pit is summarised in (Table 16.4).

Table 16.4 Oxide Resource within the Designed Pit (SP 6)

Oxide resource within the designed pit					
Tonnes (000t)	EQV Cu (%)	Cu (%)	S (%)	Au (g/t)	Ag (g/t)
4,974	0.787	0.93	-	0.09	6.85

Sulphur grades were not modelled within the oxide resource.

The Mengapur pit will be developed via a series of six staged pits. For mine scheduling purposes, the total mineable ore reserves within each stage were determined according to detailed designs for each pit stage.

Prior to actual mining, the final pit will be reviewed to incorporate the latest optimisation, any refinements to the topography and any changes in slope recommendation. This refined final pit will then be used as a basis to design the staged pits in detail.

The reserves within each staged pit at a cut-off grade of 0.336% EQV Cu were calculated. A summary is presented in Table 16.5. For scheduling purposes, the incremental reserves are presented in Table 16.6.

Table 16.5 Reserves within each staged pit

Staged Pit	Tonnage (kt)	EQV Cu (%)	Cu (%)	S (%)	Au (g/t)	Ag (g/t)	Waste (kt)	Strip Ratio
1	7,381	0.935	0.41	10.38	0.24	2.9	24,559	3.3
2	14,464	0.949	0.41	9.67	0.23	2.8	34,674	2.4
3	22,078	0.867	0.36	9.38	0.22	2.7	44,005	2.0
4	29,366	0.831	0.33	9.03	0.22	2.7	54,594	1.9
5	39,624	0.787	0.3	8.62	0.21	2.7	60,616	1.5
6	64,131	0.737	0.27	10.38	0.24	2.6	94,194	1.5

Table 16.6 Incremental Reserves within the SP Series

Staged Pit	Tonnage (kt)	EQV Cu (%)	Cu (%)	S (%)	Au (g/t)	Ag (g/t)	Waste (kt)	Strip Ratio
1	7,381	0.935	0.41	10.06	0.23	2.9	24,559	3.3
2	7,083	0.963	0.41	10.2	0.24	2.7	10,116	1.4
3	7,614	0.713	0.26	8.33	0.21	2.5	9,331	1.2
4	7,288	0.720	0.22	8.50	0.20	2.8	10,589	1.5
5	10,258	0.664	0.22	8.02	0.21	2.5	6,022	0.6
6	24,507	0.654	0.21	7.97	0.21	2.5	33,578	1.4

Master bench plans for the series of staged pits which form the basis for this division and cross-sections through the pit are presented in Normet's DFS Report dated October, 1991. A longitudinal section of the staged design along Section 99 is shown in (Table 16.7). These figures are included as an illustration of the way in which the staged pits lead to the final pit.

The first production year of the schedule is to be "highgraded", therefore the SP1 pit was evaluated at a cut-off grade of 0.450% recovered copper equivalent. The reserves within SP1 at the higher cut-off grade are tabulated in Table 16.7.

Table 16.7 SP1 Reserves at 0.450% EQV Cu Cut-Off Grade

Tonnage (000t)	EQV Cu (%)	Cu (%)	S (%)	Au (g/t)	Ag (g/t)	Strip Ratio
6,454	1.015	0.46	10.73	0.25	3.2	3.9

16.11 Mining schedule

The mine is scheduled to operate on 3 shifts per day, 7 days per week. Based on a simulation of expected interruptions due to wet weather, 17 days per year have been deducted to allow for rain. A further 11 days have been deducted to allow for public holidays, leaving a total of 337 scheduled production days for the mine per year.

The incremental ore reserves within the SP series of pits (Table 16.7) were used to develop the production schedule, with SP1 being "highgraded".

The rate of ore production has been scheduled to match the sulphuric acid production rate of 594,000 tpa which will be processed to produce 203,000 tpa of phosphoric acid (P_2O_5).

Each tonne of sulphuric acid requires 0.327 t of elemental sulphur. The recovery of sulphur from the sulphide ore is expected to be 82%, and therefore, to produce 1 t of sulphuric acid, 0.398 t of contained sulphur in the feed must be delivered to the primary crusher. In order to produce 594,000 t of sulphuric acid a total of 236,800 t of sulphur must be delivered to the primary crusher annually.

Prior to any ore production there are two years of pre-stripping. During this period, 11,109,000 t of waste is stripped from the hill and 89,000 t of ore are stockpiled for the treatment plant start-up. The pit development thereafter is scheduled to meet the plant requirement for 236,800 t of sulphur per annum.

The cut backs into Bukit Tembaga commence in Years 3 and 8 and 9. The mining operations do not go below "daylight" until Year 3.

The mining rate peaks at 8.8 Mtpa of material moved, and given the scale of the pit this mining rate is easily achievable. Once the initial pre-strip is completed the rate of vertical advance will be very low, and no problems are anticipated in achieving that rate.

During the pre-strip years the mining rate has been reduced to reflect the lower productivity that can be expected whilst pioneering and stripping narrow benches from the hillside.

Ore production peaks during the last 3 years when the average annual rate is 3 Mt of ore. The total material movement peaks in year 1 at 8.764 Mtpa.

The schedule departs from the mineable ore reserves in Year 1, in that a higher cut-off grade of 0.450% recovered copper equivalent was used. The schedule for all years after Year 1 is according to the project cut-off grade of 0.336% EQV Cu.

This "high grading" of the first year raises the head grade of that year at the expense of a higher mining rate and a shorter mine life. Some "high grading" may improve the Discounted Cashflows of the project. Some additional work will be required prior to mining to formulate a definitive "high grading" strategy which optimises the project economics.

The first 10 years have been scheduled on an annual basis; thereafter, production is scheduled in 5 yearly increments to the project end in Year 23.

A set of yearly pit designs has been developed using stage pit designs and schedules. Prior to actual commencement of mining at Mengapur, an updated set of yearly plans will be developed based on the refined staged pit designs, revisions to the topography model and the resulting new schedule.

16.12 Mining equipment

The major units of mining equipment to be used for the project have been selected by using a range of criteria. The more important criteria which have formed the basis of the selection procedure are:

- the match of the equipment to the mine production rates
- the ability of the equipment selected to operate efficiently in the mining environment envisaged
- the match between the shovel and haul truck capabilities and productivities
- local equipment supplier representation and infrastructure to provide both service support and spares stock on consignment, and
- capital and operating cost factors.

In consideration of the above criteria, the primary equipment units (or equivalents) selected for the mining operation are:

- Drilling - 3 x Tamrock DHA 1000S drills
- Loading - 4 x Caterpillar 245B face shovels (3.8 m^3 bucket)
- 1 x Caterpillar 988B front end loader
- Hauling - 14 x Terex 3307 dump trucks (40 to 44 t payload).

Details on the auxiliary equipment, a range of smaller vehicles and trucks including costs have been documented in Volume 10 of Normet's DFS Report (October, 1990).

16.13 Manpower

The organization structure for the mining department is similar to that used for many large open pit mines, using a mining contractor.

In the organisation there are a number of skill bands, ranging from the highest, technical professional level, to the less skilled "labourer" position.

The total manpower requirement for the mining operation, with a contract mining situation is 133.

All manpower associated with the mining operation has been placed into one of five designated working areas, the areas being:

- Management and technical services
- Mine operations
- Administration
- Supervision
- Planning / Monitoring.

The management and technical services department is located within the mine's main administrative complex. The overall direct management of the mining operations is carried out under the direction of the Chief Mining Engineer, who will have both operational and technical staff reporting to him. The mine supervision facet of the department includes the mine captain staff, who are the most senior operational personnel involved directly, on a per shift basis, in the open pit mind. The main job functions carried out in the technical services department are mining, geotechnical as well as the data control for each of these functions.

The mine will have on its staff a Chief Geologist - a position which will have responsibility for all geological aspects of the mine. In-pit grade control will be one of the main functions of the planning and monitoring department.

16.14 Capital and operating costs

The aspects on capital and operating costs have been thoroughly investigated and documented in Volume 10 of the Normet's DFS Report on the assumption of an owner operated mining scheme. For the purpose of reducing the initial capital outlay and in view of other advantages associated with it, contract mining has been considered and proposed for the final project feasibility study. Therefore, the capital and operating costs based on owner operated mining are not detailed in this section.

16.15 Contract mining

It has been a recent trend for large open-pit mines to operate on a contract mining basis. The advantages of contract mining are:

- reduction in the initial capital outlay, particularly on heavy mobile equipment for production
- reduction in manpower under MMC's employment
- benefits of the contractor's operational infrastructure and experience which lead to higher efficiencies and productivities
- enabling MMC to focus on management, operation and grade control as the operating routines like drilling, blasting, loading and hauling and general maintenance are undertaken by the contractor
- no or only very insignificant capital replacement on the mobile equipment.

Several established mining contractors have reviewed the project requirements and have submitted preliminary quotations. Based on these quotations, it is concluded that US$1.25 per tonne material mined would be the average price and this figure has been used in the financial evaluation of the project.

It is believed that when actual tenders are called a competitive bidding environment would he create whereby a competitive contract mining cost advantageous to the mine economics would be possible.

17 Recovery methods

17.1 Sulphide ore

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

Extensive mineralogical and methodological testworks have been undertaken between 1988 through 1993 to determine the amenability of the sulphide ore to various process options (Volume 5 of DFS Report). A pilot plant was also constructed for bulk testing.

Detailed technical and economic evaluation of other process options has indicated that the most viable process route will involve production of a copper and a pyrrhotite concentrate by conventional crushing, grinding and flotation processes. Copper concentrate will be sold for direct smelting and pyrrhotite concentrate will be further treated by roasting to generate sulphur dioxide gas. The acid plant will strip the roaster gas and convert it to high grade sulphuric acid. The sulphuric acid will be processed further to phosphoric acid using imported phosphate rock and this acid will be exported overseas. The by-products, phosphogypsum and calcine cinder will be sold to local and overseas buyers.

Heat generated during roasting and acid manufacture will be recovered as steam and converted to electricity. Power not consumed by processing will be sold to the National Grid suppliers (TNB).

The production of copper and pyrrhotite concentrates will be carried out at Mengapur and the site processing plant is referred to as the "concentrator".

17.2 Concentrator plant descriptions

The general layout of the concentrator plant site is depicted in (Figure 17.1) whilst (Figure 17.2) illustrates the flowsheet of the process option.

Figure 17.1 General layout of the concentrator plant site

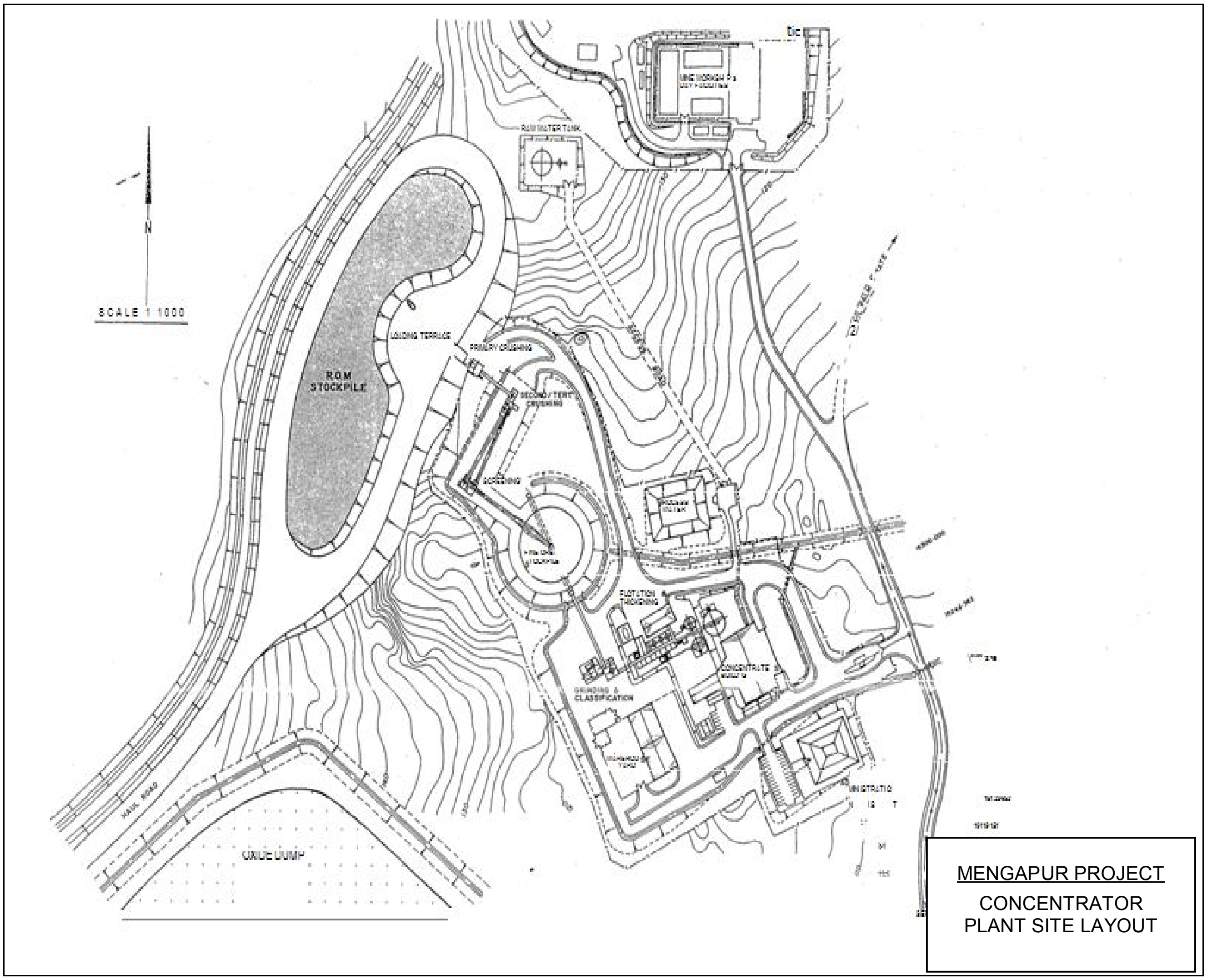
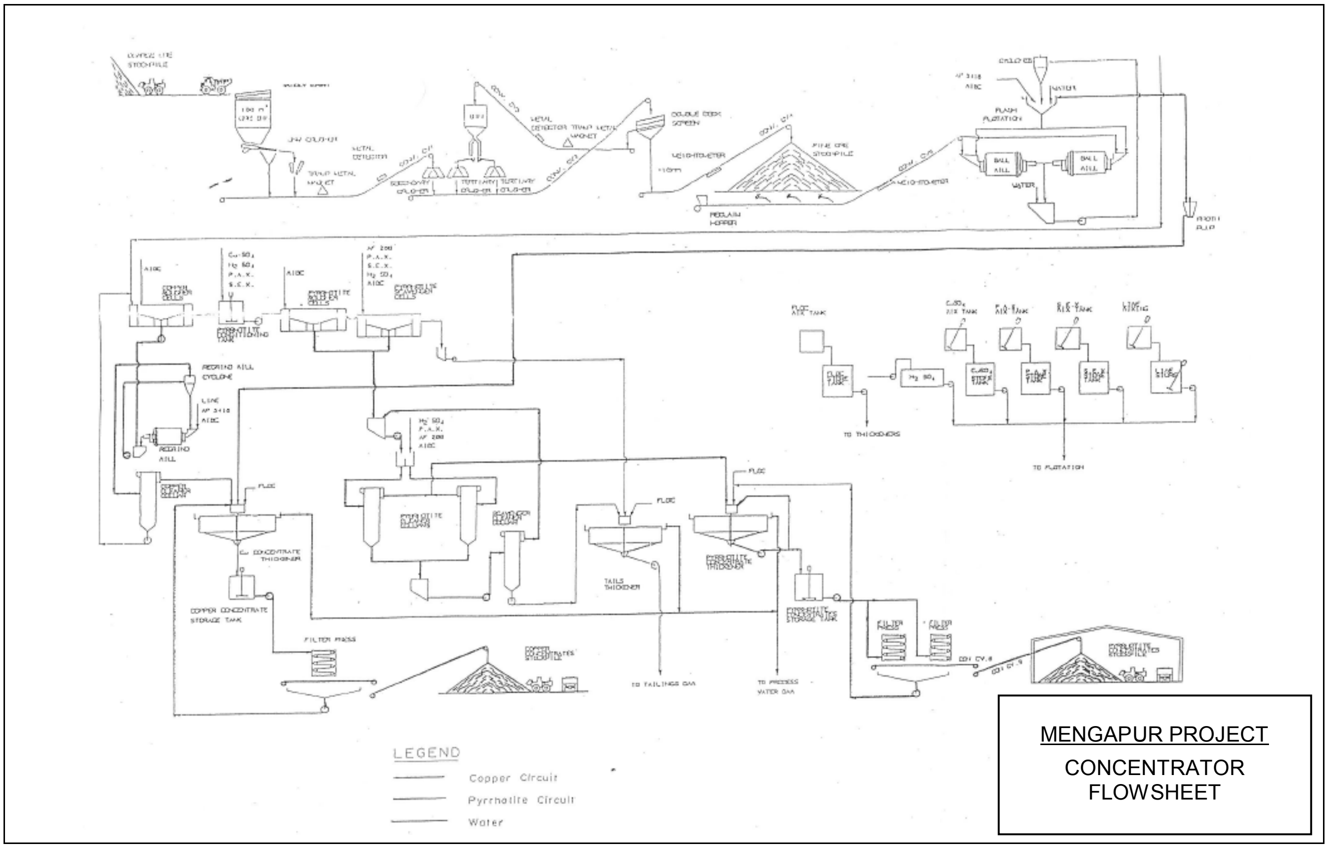


Figure 17.2 Concentrator flowsheet



Details of this option are described as follows:

Crushing

Run of Mine (ROM) ore received from the pit by 40 t to 44 t haul trucks will be dumped onto a 1.3 m grizzly screen located above a 100 m^3 ROM hopper. A 50,000 t ROM ore stockpile area will be available for ore storage. When the mine is not in operation, ore will be reclaimed from the stockpile by front end loader.

ROM ore will be discharged from the ROM hopper using a 2.4 m x 6 m vibrating grizzly feeder. Fines (-150 mm) passing through the grizzly will discharge directly on to conveyor CV1. Oversize from the grizzly will feed a 2000 mm x 1500 mm single toggle jaw crusher with a 200 mm close side setting. Jaw crusher product discharges onto CV1.

Crushed ore will be conveyed to an open circuit 1500 mm secondary cone crusher with a 32 mm close side setting. Secondary crusher product will discharge onto conveyor CV2.

CV2 will transfer material onto a 3 m x 6.1 m double deck screen, with a top screen aperture of 25 mm and a product screen of 10 mm. Screen oversize will discharge onto conveyor CV3 and transfer to the tertiary crusher feed hopper.

Two parallel 1500 mm tertiary cone crushers will be fed from the hopper using vibratory pan feeders. The closed circuit tertiary crushers have a close side setting of 10 mm and discharge onto CV2.

Product screen discharge (-10 mm) is conveyed by CV4 to the 7,000 t live, fine ore stockpile.

The secondary and tertiary cone crushers are protected from damage by tramp metal magnets and metal detectors located on CV1 and CV3.

Production is monitored by a weightometer installed on CV4.

Grinding

Crushed ore will be reclaimed from the fine ore stockpile via two static slots and a central vibrating discharger located in the reclaim tunnel. Ore is fed onto conveyor CV5 using vibratory pan feeders. Ore may be led onto CV5 by front end loader through a reclaim hopper during crusher maintenance periods.

CV5 conveys ore to two ball mill feeder conveyors. Feed rate will be controlled by a weightometer looped with the feeder conveyors.

Primary grinding will be carried out in two 5 m diameter x 7 m overflow ball mills operating in parallel. Both mills discharge into a common cyclone feed hopper where slurry is pumped to a cluster of four 660 mm classifying cyclones. Cyclone underflow will be diluted and fed directly into an Outokumpu flash flotation cell (28 m^3). Copper concentrate recovered from this cell may be cleaned or fed directly into the copper concentrator thickener as final copper concentrate. Flash flotation tails will be returned to the ball mills.

Cyclone overflow at 35% solids by weight and 80% passing 125 microns will be gravity fed to the flotation circuit.

Flotation

Slurry from the cyclone overflow will feed five Outokumpu 16 m^3 copper rougher flotation cells. Copper rougher concentrate overflows via internal launders into the copper rougher concentrate pump hopper. Lime will be added and the pulp conditioned to pH 11.5 prior to discharging into the regrind mill.

The regrind mill operates in closed circuit with hydrocyclones and grinds concentrate from P_{80} 80 microns to a P_{80} of 45 microns. Regrind mill product is fed to the copper cleaner flotation column cell. Copper cleaner tail is returned to the copper rougher cells and cleaner concentrate flows to the dewatering area.

Copper rougher tailings is pumped to the pyrrhotite conditioning tank where pulp is pH modified to 5.7 using sulphuric acid. Conditioned pulp is then fed to nine 38 m^3 pyrrhotite rougher and scavenger flotation cells.

Pyrrhotite rougher concentrate is pumped to two parallel pyrrhotite cleaning columns (3.66 m diameter x 15 m). Pyrrhotite cleaner tails are pumped to the (3.66 m diameter x 15 m) pyrrhotite cleaner scavenger column cell. Pyrrhotite cleaner scavenger concentrate is returned to cleaner columns. Pyrrhotite cleaner concentrate is fed to the dewatering area.

Pyrrhotite rougher tailings and pyrrhotite cleaner scavenger tails are both final tails and are dewatered prior to disposal.

Copper sulphate ($CuSO_4$), potassium amyl xanthate (PAX), sodium ethyl xanthate (SEX) and lime are mixed to a 10% solution and stored in holding tanks. Cyanamid reagents AP3418 and AF208 and methyl isobutyl carbinol (MIBC) are stored in 200 L drums. Weak sulphuric acid (H_2SO_4) from the Kuantan acid plant is stored in an acid resistant holding tank.

Reagents are dosed to the various flotation streams by pumping to a head tank system from which they flow by gravity to addition points. Control for most reagents is manually, by rotameter flowmeters. Lime is continuously circulated through a closed loop system. Lime and H_2SO_4 additions are automatically controlled using pH measurement and variable speed pumps.

Flotation performance and metallurgical control is assisted by the use of an in- stream analysis (ISA) system. The system monitors copper and iron concentrations and pulp density in various streams throughout the flotation circuit.

Thickening and filtration

Flotation tailings are dewatered to 55% solids in a 20 m diameter thickener prior to disposal.

A load cell measures bed mass and controls underflow discharge rate via a pinch valve. Flocculant addition is controlled by a bed level indicator.

Overflow water from thickeners flows under gravity to the process water pond.

Copper and pyrrhotite concentrate streams are dewatered in 3 m and 10 m diameter thickeners respectively to 65% solids and stored in separate agitated holding tanks. Concentrates are pumped to filters for final dewatering.

Two Lasta plate and frame pressure filters remove water by squeezing concentrate and incorporation of a compressed air blow during the filtration cycle ensures moisture content of the pyrrhotite will be less than 8.5%. A total of 770 m^2 filtering area is required to ensure the moisture content will be less, than the transportable moisture limit (9.5%).

Copper concentrate will be dewatered by a single dedicated 16m^2 pressure filter prior to conveying filter cake to a dedicated stockpile.

Both concentrates will be loaded from stockpiles using a front end loader for transport to the roaster or port.

Tailings storage

The tailings storage design is based on a storage capacity requirement equivalent to 23 years of tailings production. The estimated mean annual output of tailings is expected to be of the order of 2.0 Mt annually, equivalent to 46 Mt over the 25 year period. At an assumed settled density of 1.6 t/m^3, this represents a potential storage requirement of 30 m^3, or an annual output of 1.3 m^3.

The proposed design of the main embankment foresees a two-stage construction programme with the first stage of construction having the capacity to accommodate the initial five years production of tailings.

Stage 2 constructions will constitute an ongoing construction phase during which the embankment will be raised to the final height. During the construction a staged temporary spillway system will be maintained to allow the safe discharge of excess water from the storage.

The hydrological studies indicate that during the initial two year pre-production phase, a storage capacity of 200 ML of water would be required to ensure a secure supply of process water prior to start-up. It will therefore be necessary to ensure that construction of the storage is sufficiently advanced to provide the necessary tailings and water storage and that a spillway is provided to manage catchment contributions from major rainfall events.

The storage embankment will comprise a valley impounding structure located approximately 3.4 km distance from the proposed plant site at the eastern end of a west to east trending valley. It is proposed that tailings will be deposited into the storage by open-ended discharge from the head of the valley. The embankment will be a conventional earth and rockfill structure with a clay core.

Tailings description

It is deemed necessary to adopt the values obtained from testwork carried out on material sampled at the pilot plant operation although it is understood that variations in the material characteristics are likely to occur within the orebody.

A number of particle size distribution determinations have been carried out on material collected from the flotation circuit. A comparison is provided is (Table 17.1).

Table 17.1 Tailings particle size distribution

Sieve Size	Dec 1989	Apr/May 1990
-75 µm	67.4	85.2
-20 µm	31.6	43.0

The particle density of the solids fraction was determined on a bulk flotation sample from the pilot plant and a value of 3.4 t/m^3 obtained.

Embankment location

The selected site for the embankment is a steep sided valley controlled by a deeply incised creek and rocky spurs trending across the main drainage feature.

A geotechnical evaluation of the proposed site has been carried out, including drilling works within the proposed embankment footprint. A preliminary assessment of construction material availability has been made. The information- obtained during the course of the investigation has been collated into a separate report (Volume 9 of DFS).

Conceptual storage design

The design proposals foresee that the embankment will be constructed in two major stages to provide a final storage capacity for an estimated 46 Mt of tailings deposited over a 25 year active life of storage.

A preliminary volume vs. height curve has been compiled from a 1:10,000 plan contoured at 50 ft intervals.

Based on the prepared capacity curve, the first stage of embankment construction would be to an overall height of approximately 38 m which would include provision for a 5 m free board and provide storage for the initial five years of tailings production. The second stage of construction would increase the height of the storage progressively to approximately 72 m to provide storage for 25 years tailings output. At completion of each construction stage, a formal spillway structure would be constructed.

17.3 High grade leachable oxide ore

Fundamental design criteria

Studies indicated that the leaching of high grade leachable oxide ore (HGLO) comprising the supergene layer overlying the skarn ore in Zone A, by heap leaching would be viable. Since the amount of HGLO ore totals only 2.3 Mt, a simple process of leaching using a weak solution of sulphuric acid in water is envisaged. In this leaching process, only copper would be dissolved and leached out. The recovery of copper from the oxide ore by this process is expected to be consistently good. The recoverable copper grades for the ore are derived from the recoveries as determined in the bottle roll tests and column tests done by MMC and Normet's laboratory in Perth (Volume 2 of FSOR). The tests also confirm the rapid leaching characteristics of the supergene ore.

Table 17.2 Cut-off grade for HGLO ore

Net Copper Price	US$/t
Gross Price	2173.74
Less Treatment	116.11
Less Not Paid	24.15
Less Refining	205.05
Less Marketing	27.43
Less Transport	7.91
Price ex tonne	1793.09
Less tribute at 5%	89.65
Less iron cost for cementation	280.00
Marginal price for Cu in cement	1423.44

Processing costs

Ore preparation including agglomeration and heap leaching

Metal recovery (excluding iron)	$3.17
Site overheads, manpower, assays	$0.77
TOTAL PROCESSING	$3.94

Project cut-off grade

$$\frac{Processing\ Cost\ x\ 100}{NetPrice/t\ \ Copper} = \frac{3.94\ x\ 100}{1423.22}$$

= 0.276% recoverable copper

The HGLO ore treatment is based upon a sulphuric acid heap leach with metal recovery by cementation. The selected treatment rate is based upon treating the majority of the reserves of HGLO ore over a three year period.

17.4 Design parameters

The overall design parameters are included in Table 17.3.

Table 17.3 Mine design parameters

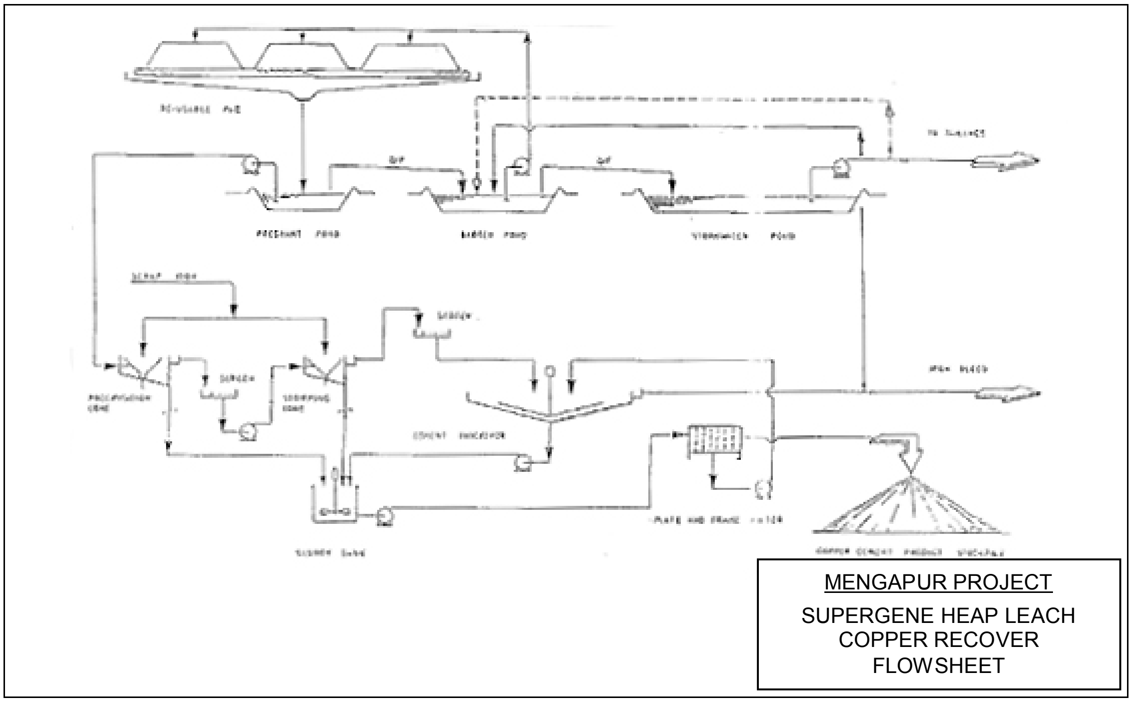
Reserves	
Tonnes	2.3 Mt
In-situ Grade	1.294% Cu
Recoverable Grade	0.845% Cu
Treatment Rate	
Yearly	0.30 - 0.50 Mtpa
Monthly	30,000 t/month
Daily	1,000 t/day
Ore	
In situ Density	2.2 t/m^3
Bulk Density	1.5 t/m^3
Moisture	12%
Leach Pads	
Number	6
Liner	1.55 mm HDPE
Slope	4%
Base	Coarse waste
Length/Pad	58m
Width/Pad	38m
Heaps	
Height of Lift	3m
Capacity (ore)	6,166 dry t
Solution Application Rate	
Irrigation time, including soakage	25 days
Barren solution to active fresh ore heaps	43.5 m^3/h
Pregnant solution returning from heaps	41.8 m^3/h
Rainfall (average) 0.24 L/m^2/h	32 m^3/h
Rainfall (max in 24hr event)13L/m^2/hr	213 m^3/h
Final residual moisture	20%
Irrigation System	
Solution Emitter	Dripper, 8L/hr
Spacing	0.9 m grid
Total area of application	1,073 m^3
Evaporation	1,300
Ponds	
Pregnant	
Live Capacity	2,500 m^3
Length	39m
Width	22m
Barren	
Live Capacity	4,000 m^3
Length	63m
Width	22m
Storm Water	

Live Capacity	4,000 m^3
Length	63m
Width	22m
Liner	1.0 mm
Evaporation 0.14L/m^2/hr (prep/barren only)	0.3 m^3/h
Heap Leach Treatment Cycle	
Treatment Rate	1,000 t/day
Leach Time	25 days
Loading Pad Time	7 days
Unloading pad time	5 days
Total cycle time / pad	37 days
Cementation Feed	
Pregnant solution	
Flowrate	41.8 m^3/h
Pressure	700 kpa
Cu gpl	13.8
pH	2 - 2.5
Cementation Availability	85%
Cementation reactors	
Number of vessels	2 in series
Outer tank diameter	4.3m
Outer tank height	6.7m
Inner cone top diameter	3.0m
Inner cone bottom diameter	0.6m
Inner cone height	4.0m
Recovery per stage	80%
Scrap Iron	
Iron Consumption	1.7 kg/kg of Cu
Iron Consumption pa	8,602 t
Scrap Type	Detinned thin plate
Cone overflow settling	
Thickener feed	41.8m^3/h
Overflow to barren [pond	41.4m^3/h
Thickener underflow to surge tank	0.39m^3/h
Thickener underflow density	25% solids
Filtration	
Total cone underflow to surge tank	1.50 m^3/h
Thickener underflow to surge tank	0.39m^3/h
Total feed to surge tank:	
Flow rate	1.89$m^{3/h}$
Density	25% solids
Filter cycle time	1 hour
Surge tank capacity	6 m^3
Final Product	
Copper Grade	60% - 90%
Moisture	20%

Process description

The flowsheet for the heap leaching of the high grade leachable oxide ore is depicted in (Figure 17.3).

Figure 17.3 Supergene heap leach copper recovery - flowsheet



Pad preparation

An area of approximately 14,000 m^2 would be required for the construction of a re-usable Heap Leach pad.

The ground would have to be cleared, grubbed, filled and excavated where required to prepare a suitable surface for the pad. Mined waste would be used as base material for the leach pad. The pad base would have a slope of 4% in the one direction which would be tilled and compacted in preparation for the membrane placement. A layer of compacted oxide waste fines would be placed beneath the membrane.

Pond preparation

An area or approximately 3,000 m^2 would be required for the establishment of 3 ponds consisting or a pregnant and barren solution pond in addition to a stormwater pond. The ponds would be excavated to a depth of 4 m and compacted in preparation for the membrane placement.

Leach pad, drains and ponds -

A membrane of 1.55 mm HDPE would be laid down onto the graded and compacted pad surface. The membrane when joined by heat welding would form an impermeable base onto which a trafficable surface of screened coarse crushed waste material would be placed and compacted.

The trafficable material would be prepared by utilizing a transportable crushing and screening plant located on the waste dump. Intermediate sized waste (-200 +25 mm) is proposed as suitable material for membrane protection and long term durability for constant loading and unloading of the pad.

The trafficable layer forms 2 purposes:

- protection of the liner during loading and unloading
- improves drainage characteristics.

The total leach pad would be 230 m in length by 58 m wide. This area would be divided into 6 cells.

The cells would be divided by HDPE covered bunded walls. The pads have been designed to enable easy access and maximum efficiency for loading and unloading ore.

The pregnant solution from each cell would drain to a separately located weir box, so the individual leaching performance of each cell can be monitored separately.

The membrane for the drains and ponds would be 1.00 mm HDPE due to the relatively short nature of the operation.

The common pregnant solution drain would feed to the pregnant solution pond. In the event of a pumping failure or excessive storm activity, the pregnant pond would overflow to the barren pond which would subsequently overflow to the stormwater pond.

Heap building

Run of mine supergene ore would be trucked to the pad area and dumped onto the trafficable layer.

A rubber tyred front end loader would then build the heap to a height of 3 m. It would take 7 days to build one leach cell.

During the building of each heap, irrigation piping would be continuously laid and barren solution applied immediately upon completion of the heap building. On completion of the first heap, the second heap construction would commence and so on until all 6 pads were constructed.

The irrigation piping has been designed for easy handling prior to and after leaching of each heap.

Irrigation

The prepared heap of run of mine ore would be leached by continuous application of sulphuric acid bearing solution from the barren solution pond to the top of the heap.

The solution would permeate through the heaped material, contacting and soaking the ore particles and dissolving the acid soluble copper into solution.

The solution would be pumped through trunk and distribution piping into regulator controlled drip emitters. The drip emitters would distribute the solution evenly over the surface of the heap.

The solution would pass out of the heap by permeating through the heap via the trafficable layer to the membrane and flowing into a "v" notch weir which would overflow to a common pregnant solution drain, which would feed the pregnant solution pond.

Cementation

Pregnant solution would be pumped to the first of two scrap cementation cones in the metal recovery area. The two units would be installed in series such that solution from the first cone would be pumped to the second cone.

The outer part of each cementation cone would consist of an epoxy lined tank complete with an overflow launder for barren solution.

The inner section of each cementation unit would consist of a stainless steel cone with an upper screened section to facilitate separation of precipitated copper.

The bottom section of the cone would be fed with solution via a stainless distribution system designed to maintain the necessary velocity of solution to the scrap iron. The cement copper would be dislodged upwards from the scrap iron and would settle through the upper screen to accumulate on the sloping false bottom of each vessel. A dump valve would be installed on the base of each cone for accumulated trash removal.

Screens would be installed on the outlet launders of both vessels to remove trash and scrap iron.

Scrap iron would be fed to the cones from an adjacent scrap storage area by electromagnetic hoist, hopper and conveyors.

The barren solution from the second cone would gravitate into a rubber lined thickener with rubber lined rake arms. Barren solution would be returned to the barren pond of the heap leaching area.

Cement treatment

The settled copper cement on the floor of each cone would be released by a discharge valve which would open intermittently and allow the cement slurry to gravitate through a trash screen to an agitated surge tank.

The settled copper cement from the overflow thickener will also be pumped on an intermittent basis to the same surge tank.

The epoxy lined surge tank would be agitated with a rubber lined impeller and baffles and would ensure that feed for the filter press is of uniform composition.

The cement slurry would be pumped to a filter press with polypropylene filter plates. The cycle time for the filtering process would be one hour.

Barren filtrate from the filter press would be returned to the discharge launder of the first cone. The filtered cement would be manually removed from the press and stored in a covered area for subsequent trucking to the coast for export.

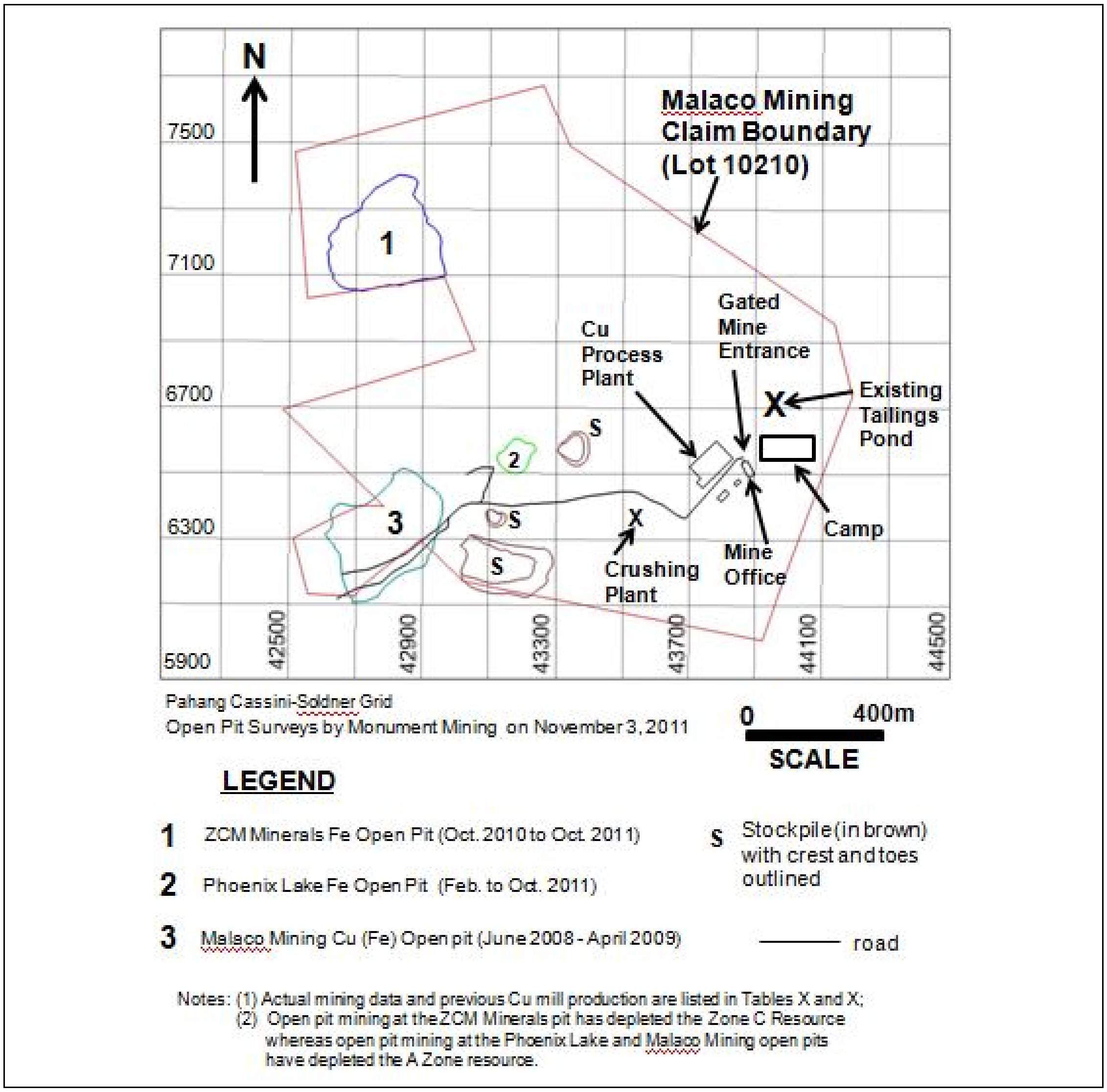
18 Project infrastructure

18.1 Access road

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

Existing access to the mine site consists of 17 km of badly maintained gravel road commencing at the township of the Seri Jaya on the Kuala Lumpur-Kuantan highway (Figure 18.1).

Figure 18.1 Mine Site Layout



The road currently carries intermittent heavy logging and palm plantation vehicles and a consistent but low level of private cars and motor bikes.

The current road alignment and surface is unsuitable for continuous heavy traffic on a 24 hour basis and will require realignment and upgrading to ensure that access to the mine and can be maintained in adverse weather conditions of equal importance is the need to provide safe travel and conditions for the general public including existing users mining staff and trucking companies.

A detailed survey of the existing road was completed. From data received, Normet designed and costed a 7 m wide sealed road from Seri Jaya to the Mengapur site generally within the existing alignment in accordance with the requirements of the Jabatan Kerja Raya - Manual on pavement design and in situ C.B.R. tests. To contain costs, the road will be upgraded and realign only when necessary

It is proposed that upgrading of the road would commence at an early stage of project implementation to ensure that same access is provided for heavy construction traffic.

18.2 Water supply

Potable water for the Mengapur site would be obtained from a small storage dam and located south of the mine (Figure 18.1).

The dam embankment would be of earthfill construction with a compacted clay core constructed from locally won material. The embankment will be 13 m high 10 m wide at the crest with 2 m free board and will contain 170 ML water. The ground surface beneath the embankment would be stripped of topsoil and other unsuitable material. A key trench will be excavated beneath the embankment and filled with compacted selected fill. Spillways would be incorporated to prevent overflows and would pass a 1 in 100 year storm event. Supply to the concentrator plant site would be gravity fed to the raw water storage tank using high density polyethylene pipe with thermal welded joints. An access road would be constructed along the pipe line route to provide all weather access. Full details are provided in the AGC report Water and Waste Management contained in volume eight of DFS report.

18.3 Electrical supply

An 11 kV underground cable will be installed from the low voltage side of TNB 132/11 kV transformers to the incoming terminals of the 11 kV switchboard main circuit breaker. The 11 kV switchboards distribute 11 kV supply throughout the sites from a number of H.V. feeders. At Mengapur an 11 kV feeder supplies power via an underground cable to an overhead 11 kV power line which reticulates power to the tailings return pumps transformer.

The 415 V power supply for the process equipment is obtained from the 415 V the main distribution board located within each substation.

A low voltage 110V power supply is provided to each Motor Control Centre from a 415/110 V transformer within the Motor Control Centre for the control voltage to operate all process plant equipment.

18.4 Transportation and communication

Capital costs for both the Mengapur and Kuantan plants have been based on major processing equipment being supplied either directly from overseas to the port of Kuantan or being shipped via Singapore.

Obviously individual contractors will procure construction materials and supplies for the most cost effective sources however, the steel and timber products will undoubtedly be supplied from the Kuantan area. Normal construction type materials dependent upon the large West Coast and Kuala Lumpur market for sales volumes, will be supplied to site by road from Klang, Kuala Lumpur, Johor Bharu and Singapore. Obviously transportation costs will rise dramatically if the Seri Jaya-Mengapur access road upgrade is not completed at a very early stage of the project development.

18.5 Operations transportation

Because of the large volume of reagent and raw material supplies needed for both the Mengapur and Kuantan sites, direct regular shipping to the Port of Kuantan from China, Singapore, and the West Coast would greatly reduce transportation and plant operation costs.

18.6 Communications

Telecom Malaysia TM was consulted regarding appropriate communication systems suitable for servicing the work force and community requirements of a proposed mine and plant site located at Mengapur.

TM recommended four systems applicable for rural/remote communities.

They were:

- Physical overhead cable from Seri Jaya exchange to Mengapur
- Mobile telephone
- VHF "country set" telephone system
- Microwave radio system.

Information supplied by TM (11 June 1990) provided a brief description of each system listed the advantages and disadvantages and provided initial costs of the four systems.

The results show that a microwave radio telecommunications system would provide the services requested at Mengapur. This system would provide the best quality communications without any congestion problems when both the mine and plant site become operational. The system would require 9 - 18 months to install and require TM survey and approval prior to installation.

The evaluation also highlighted the possibility of utilizing the VHF country sent system during the construction phase of the mine and camp site (Figure 18.2). The system would provide good quality voice requirements however facsimile or telex services would not be available. This system has a relatively quick installation period ranging from two weeks to one month.

Figure 18.2 Telecommunications Systems Assessment

COMPARATIVE ASSESSMENT OF TELECOMMUNICATION SYSTEMS AVAILABLE FOR OPERATION AT MENGAPUR

	PHYSICAL CABLE	MOBILE TELEPHONE	VHF COUNTRY SET	MICROWAVE RADIO
Services Required at Mengapur				
- Multiple lines	Yes	Limited	Yes	Yes
- International Subscribers Dialling	N/A	N/A	Yes	Yes
- Telex	N/A	N/A	N/A	Yes
- Facsimile	-	N/A	N/A	Yes
- Data Quality Line	N/A	N/A	N/A	Yes
Associated Costs (per line)				
- Installation	Paid by Telekom	RM350	RM1,550	Telekom to pay
- Rental	Low	RM770	RM3,000	RM1,980
- Equipment Costs	Low	RM5,000+	Tower, etc	Telekom to pay
Quality of Lines	Very poor	Poor	Good	Very good
Comparative Assessment	Unsuitable	Unsuitable	Proposed during construction period and also as emergency backup	Most appropriate

18.7 Communications conclusions

Of the four telecommunications available for operation at Mengapur the microwave radio system is the preferred option.

- During the construction. A temporary VHF country set is recommended
- Facsimile communications may have to be stationed at Seri Jaya
- Adequate telecommunications facilities are available at Kuantan site
- In plant "Tannoy" or similar intercom systems will be required for each plant site and a VHF 2 way radio control system will need to be installed at the Mengapur site to control operations.

18.8 Accommodation

Permanent accommodation will not be provided at either the mine site or at Kuantan for MMC mine or plant employees. Temporary accommodation will be provided for staff at Mengapur site during the initial years of the project. This will be in the form of cabin type accommodation only. Permanent accommodation would be provided at the Mengapur site after the project payback period. Accommodation at Kuantan site will be mainly rented premises.

Allowances such as housing feel hardship etc will be paid to employees where applicable

18.9 Facilities

Mine and acid plant site

The major infrastructure role facility at Mengapur will be the administration building which will house the senior administration technical and support staff. Other facilities to be provided at Mengapur include cabin type accommodation centralised bathrooms slash portents canteens dispensary and basic recreation. These types of facilities already exist at the urbanized Kuantan area. In addition a modern fully equipped workshop for mechanical and electrical repairs has been included in the concentrator site. At the mine site the special workshop for having mining equipment would be the built by the mining contractor.

While the majority of spares at the concentrator can be shelved, those at the mine site will need to be handled by forklifts. The concentrator site will have a laboratory for in-plant monitoring, metallurgical testwork, and assay work for ore grade control

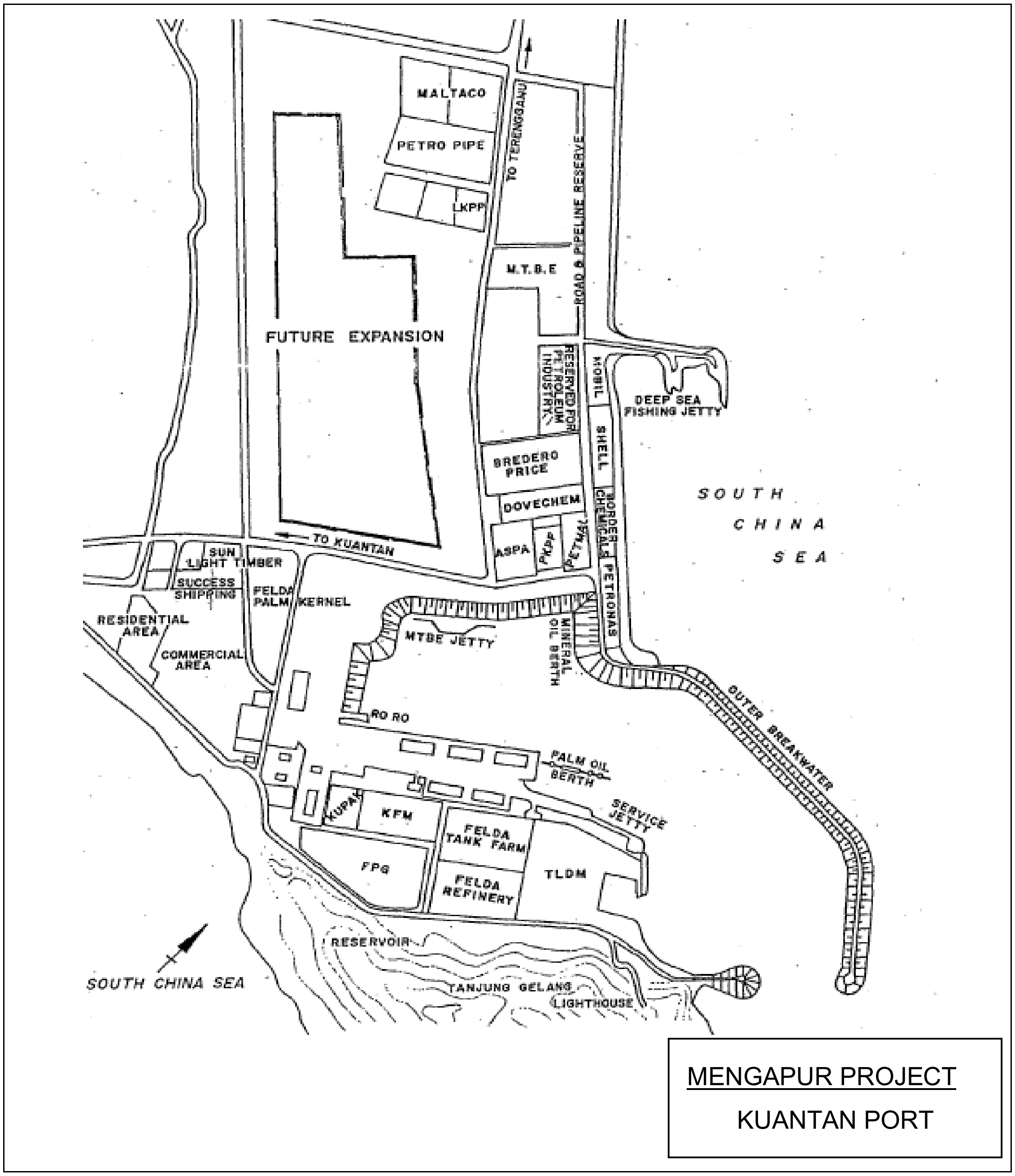
The roaster acid plants will be supplied with a large sophisticated laboratory which will handle all assay work for both plants and provide a section for environmental monitoring.

Both sites will provide basic facilities and amenities for employees including security service.

Kuantan Port

The import of phosphate rock and export of phosphoric acid for the Mengapur project will necessitate the expansion of Kuantan port facilities to meet the large bulk handling requirements (Figure 18.3). In a meeting between MMC and the Kuantan Port Authority (KPA) to discuss the project requirements, KPA has expressed their support on the project and informed that they have included in the Sixth Malaysian Five Year Plan (1990-1995) a provision for the port expansion to handle additional cargo envisaged.

Figure 18.3 Kuantan Port



19 Market study and contracts

19.1 Introduction

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

The Mengapur project will produce a variety of saleable products as listed in (Table 19.1).

Table 19.1 Saleable products

Products	Production per Year (tonnes)
Copper Concentrate @ 22% Cu Containing 5 g/t Au and 60 g/t Ag	30,500
Phosphoric Acid	203,000
Excess Sulphuric Acid	30,000
Phosphogypsum	900,000
Calcine/cinder	550,000
Surplus Electricity	9.5 Megawatts

Detailed product analyses by various consultants are provided in their respective marketing reports and dealt with thoroughly in DFS and FSOR.

For the current financial evaluation, all the price forecasts have been based on real term prices without any escalation. These prices are published regularly for all of the products produced and there are standard industry price forecasts available for most of the products.

19.2 Copper concentrates

The Mengapur project will produce a relatively small quantity of copper concentrates averaging 30,532 tpa and variable amounts of copper cement. This production is very small compared to total world production. This concentrate will assay at 22% Cu, 5.6 g/t Au and 63.9 g/t silver pay metals. There is no market for this product in Malaysia and all of it will be exported for smelting.

The copper concentrate is relatively low grade but contain impurities well within acceptable levels. Bismuth is expected to be within permissible levels allowed without penalty charges. These acceptable and penalty impurity levels vary from smelter to smelter and for some smelters this concentrate might contain unacceptable bismuth levels. However, all other elements are well within the acceptable levels and the copper concentrate will be readily marketable on a long term contractual basis.

The copper concentrate would be shipped 3 or 4 times per year in shipments of approximately 10,000 t and freight costs assume shipment to Japan or other Asian market like China.

The payable metal contents vary with concentrate grade and have been allowed as follows:

- Copper — Contained copper less 1 unit
- Gold — 94% for grades below 5 g/t
  95% for grades above 5 g/t
- Silver — 90% of contained metal

Copper concentrate marketing expenses have been calculated allowing for smelting charges, refining charges, shipping costs and penalty in order to work out the price of payable copper metal. These expenses which also include the smelting and refining charges for gold and silver are estimated at US$750/t of payable copper contained in the concentrate. Hence, the realisable price of payable copper metal contained in the concentrate is US$0.78/lb Cu metal, based on a copper metal price of US$1.12/lb is a fair assumption taking into consideration of the world market copper prices predicted by WEFA (Wharton Econometric Forecasting Associates) as set out in (Table 19.2).

Table 19.2 World market copper price predictions

World Market Copper Prices Predicted by Wefa											
YEAR	**1990**	**1991**	**1992**	**1993**	**1994**	**1995**	**1996**	**1997**	**1998**	**1999**	**2000**
LME HG COPPER CASH PRICE (US¢/lb)	120.7	106	111.5	116	120	125	130	135.5	141.2	147.3	153.8

19.3 Gold and silver

Appreciable amounts of payable gold and silver metal averaging 162 kg/yr and 1,747 kg/yr respectively have been estimated to be present in the copper concentrate. These payable estimates have taken into account the metallurgical recoveries (29% for Au and 27% for Ag) and smelting/refining losses of about 6% and 10% respectively for the metals. Hence, cash revenue generated will be derived by direct multiplication of the metal price and the payable metal content. For the economic evaluation, the gold and silver prices are assumed at US$350/oz and US$5/oz respectively. There should be no difficulty in selling the gold and silver taking into account the relatively small annual production.

19.4 Phosphoric acid

Since the domestic market for sulphuric acid would not likely take up Mengapur’s full production of sulphuric acid, an alternative option was required. Lurgi GmbH agreed with MMC that phosphoric acid could be produced in Malaysia and sold into the Indian market. The market for phosphoric acid and fertilizers has therefore been investigated.

Prices of phosphoric acid are fixed by negotiation between buyer and seller, and not by reference to an open market price. Recent contract prices are however widely published in specialist magazines, such as 'Fertilizer Focus' and 'Fertilizer International'.

An important difference between sulphuric and phosphoric acids is that there is no involuntary production of phosphoric acid. Thus, whilst prices are quite volatile, the lowest costs of production set an absolute floor below which prices will not drop.

To make 1 tonne of phosphoric acid (P_20_5) requires about 2.99 t of phosphate rock, and about 2.5 t of sulphuric acid. The average production of phosphoric acid as P_20_5 at Mengapur will average 203,000 tpa. The amount in acid solution @ 54% P_20_5 would be 370,000 tpa.

Only two areas in the world, the USA and North Africa, produce more phosphoric acid than they consume. (Table 19.3) shows phosphoric acid consumption as a percentage of production for the producing areas.

Table 19.3 Phosphoric acid consumption and production in 1989 ('000tpa P_2O_5)

Region	Cons % of Production	Production '000 tpa	Production Capacity '000 tpa	Cons '000 tpa
Western Europe	102	2826	3590	2896
Central Europe	108	1274	2012	1383
USSR	115	5127	6633	5905
North America	89	10595	11419	9427
Latin America	112	1046	1652	1177
Oceania	141	117	197	165
Africa	61	3196	5413	1966
Near East	110	1255	1903	1380
South Asia	326	391	491	1278
East Asia	110	1244	1735	1368
Socialist Asia	110	50	111	55

Only Morocco and India have announced plans for a significant increase in phosphoric acid manufacture. India is a special case amongst non-rock producing countries, in that it alone is responsible for buying 41% of the total world exports of phosphoric acid. Morocco, as a major rock producer, currently has 45% of the total world export market for acid. The general tendency for non-rock producing countries is to reduce their manufacture of phosphoric acid.

Phosphoric acid however is an intermediate product for the final production of various compounds and high nutrient value fertilizers, such as Di-ammonium Phosphate (DAP). DAP contains both nitrogen and phosphorus. These products can be transported at cheaper costs then those borne by phosphoric acid and can be considered as one of the downstream manufacturing operations.

Taking the above into account, the advantage that Mengapur has for producing phosphoric acid for export is its proximity to Asian markets.

Table 19.4 below shows world imports of phosphoric acid in 1990 and 1991.

Table 19.4 World trade in phosphoric acid by importer ('000 t P_2O_5)

Country	1990	1991 (est.)
Western Europe	478	378
Eastern Europe	78	9
Turkey	270	80
Saudi Arabia	135	218
Australasia	27	32
India	1090	1471
Indonesia	288	192
Brazil	123	155
Venezuela	18	52
Africa	28	19
Other	100	147
TOTAL	2,635	2753
Super Phosacid	725	525

From Table 19.4 a potential market is Indonesia, which currently imports phosphoric acid under contract from the Philippines, as well as from the USA and North Africa.

Selling to Indonesia gives Mengapur the best possible freight advantage: Interacid have advised MMC that they would be able to take all the production of phosphoric acid from Mengapur, and have advised a current price range of US$320 to 350/t P205.

Mengapur has great advantage over the major producers of phosphoric acid in terms of sea freight, and siting the plant near the Kuantan Port would eliminate the land transport costs of phosphate rock.

A recent visit to India by MMC personnel with Lurgi GmbH consultants in October 1992 to carry out a market study has revealed a vast market potential for the acid in India. Contrary to Normet's findings in the DFS, India has recently changed its economic/investment policies and 'opened' its doors to international participation since April 1992. The monopoly on import/export trade which was controlled by the government cartels has now been replaced by individual, private entrepreneurship involving direct international trade, as advised by the Managing Director, Mr. A.K. Raina, of Metallgesellschaft (India) Pvt. Ltd (MG). MG has also promised that they can arrange three different buyers in India for Mengapur's phosphoric acid in order to overcome fluctuations in seasonal demand. MG further advised that they can also assist MMC in the letter of credit (LC) facilities and logistics in connection with the sales of phosphoric acid.

A report has been received from British Sulphur, discussing the factors affecting the future price of phosphate rock and phosphoric acid. They confirmed that growth in world demand for fertilizers is inevitable, and a major growth area will be Asia. Substantial new phosphoric acid capacity will be required in the mid 1990's. Much of the expansion will probably be in Morocco, but it will not have any disadvantage to South East Asian producer due to distance from the growing Asian market.

19.5 Excess sulphuric acid

The Mengapur project will entail roasting of some 600,000 t per year of pyrrhotite concentrate to produce SO_2 which in turn will be converted into 594,000 tpa of sulphuric acid. This production is quite insignificant to total world production but is substantially more than the consumption within Malaysia. As such, processing of the sulphuric acid to produce phosphoric acid using imported phosphate rock is deemed imperative for the project viability.

For the production of 203,000 tpa of phosphoric acid as P_2O_5, the sulphuric acid consumption would amount to 560,000 tpa. The Mengapur project will consume about 4,000 tpa of sulphuric acid for heap leaching and flotation. Hence, the excess sulphuric acid available would be about 30,000 t per year.

The supply and demand of sulphuric acid locally and abroad has been extensively studied in DFS by various consultants like Survey Research Malaysia (SRM). Inferring from SRM's projected strong growth in the consumption of non-titanium dioxide sulphuric acid in the future, there should not be any difficulty in selling the excess 30,000 t in the local market. A conservative price of US$40/t sulphuric acid has been assumed in the current financial evaluation though higher prices of the US$60-US$80/t order have been projected.

19.6 Copper cement

For the Mengapur project, copper would be produced in two forms, copper concentrate from the skarn treatment plant, and copper cement from leaching of high grade oxide ore. The best way to market the copper cement would be to mix with the concentrate. This would increase the average grade of concentrate and may make it easier to sell.

In the DFS report, the smelting charges, refining fees, penalty and transport costs calculated for 1 tonne of payable copper in cement totals to US$396; this is equivalent to US$0.18/lb. Hence, revenue generated from producing 1 lb of payable copper in the copper cement would be US$0.94, based on the LME metal price of US$1.12/lb.

19.7 Other products

The main by-product of phosphoric acid manufacture, calcium sulphate is produced in an amount of about 4.5 t per tonne of acid. Revenue from the sale of this product could improve the economics of phosphoric acid manufacture. The product is commonly known as "phosphogypsum", to distinguish it from the natural gypsum product.

In Malaysia, the major use for gypsum is as a cement additive to retard setting as well as for producing plaster board. Details of imports of gypsum and calcined gypsum (i.e. plaster) into Malaysia are shown in Table 19.5, for the years 1980 to 1991.

Table 19.5 Imports of gypsum and calcined gypsum to Malaysia, 1980-1991

Year	Gypsum	Calcined Gypsum (POP)
	tpa	tpa
1980	122994	2322
1981	94544	2772
1982	121349	4150
1983	130107	7647
1984	158929	11852
1985	174457	11533
1986	145130	7398
1987	129427	7989
1988	154689	23074
1989	202667	19279
1990	236322	20238
1991	401740	25030

Taking an average rate of growth in imports from (Table 19.5), it is estimated that the total Malaysian market for gypsum will be equal to the anticipated production of phosphogypsum from Mengapur, by the year 1996, which is 900,000 t per year.

In order to find acceptance by the cement plants, the phosphogypsum usually has to be dried, partially calcined and agglomerated, to give a product similar to natural gypsum in size and moisture content. The cement plants could be persuaded to accept a dried but fine product. This would in fact reduce their grinding costs, though some adjustment to their materials handling equipment would be required.

In a survey done by In-Depth Research and Management Consultants Sdn Bhd (In-Depth), it has been indicated that there should be no problem for using phosphogypsum for the manufacturing of cement. They concluded that the projected increase in cement production would augur well for the entry of synthetic gypsum.

The domestic market for calcined plaster has grown rapidly, from 2,300 t in 1980 to 25,000 t in 1991. At least three companies are now producing plaster wall and ceiling panels in Malaysia. It is probable that use of these materials will continue to grow. It could be profitable to produce calcined phosphogypsum to supply the plasterboard industry.

In-Depth survey show that the price for the imported natural gypsum varies between RM40-63 per tonne whilst in the current economic appraisal, the price adopted for the base case study is US$10 per tonne (RM26/t), ex-plant.

Calcine/cinder is a high-iron by-product resulting from roasting of pyrrhotite concentrates. The amount of calcine is estimated to be 550,000 t per year. This product is marketable both locally and overseas as feed to iron and steel mills. A local company has expressed their interest in purchasing all the calcine produced by Mengapur for their proposed Integrated Iron and Steel Plant to be set up near to the Mengapur acid plants. They have submitted a draft Memorandum of Understanding in connection with the purchase of the calcine/cinder for MMC's consideration at the price of US$6.00 per tonne dry weight, ex-plant.

There is also a large market in China for the calcine/cinder though at a lower price in view of the freight expenses.

Electricity will be produced from the steam generated by the roasting of the concentrate at Kuantan. This electricity will be used to supply internal power requirements for the acid plants but power generation will be in excess of these needs and 9.5 MW will be available for sale to the TNB.

In view of the current power shortage in Malaysia and the expected increase in demand, it has been the government policy to purchase power from the private producers. Hence, there should be no problem in the marketing of surplus electricity.

20 Environmental studies

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

The proposed construction phase of the Project will extend for approximately 24 months and will result in a number of short term effects such as dust, noise, erosion and sedimentation and the physical disturbance to the site and consequent destruction of habitats caused by earthworks. The construction phase is expected to have a significant positive effect on employment in the region.

The construction phase must be accepted as fundamental to the development of the Project. Planning and design will reduce the deleterious effects of the environmental impacts. Restoration and stabilisation will commence as soon as construction is completed.

Discharge of tailings supernatant through the spillway in the storage embankment is a possible means of discharging heavy metals into the downstream waters of the Sungai Lepar.

Modelling of the tailing system indicated that, even under the most conservative assumptions, concentrations of metals in the Sungai Lepar would rarely, if ever, exceed the water quality standards for Class II.

The tailings retaining system (TRS) has been designed to cause minimal environmental hazards. On the cessation of mining activities the TRS will maintain itself as an open water lake. Hence there will be no environmental impact from solid tailings outside the embankment area.

Acid spillage will not be an occupational hazard as any material split will be retained within the system of bonds and sumps constructed at the plant.

Any rupture or leak within the underground acid pipeline to the loading terminal will be detected instantaneously by pH meters located within underground sumps and will therefore be contained without any impact to the surrounding environment.

Dispersion modelling of SO_2 levels from the stack of the acid plant indicate that ground level concentrations will not impact to any significant extent upon the acid plants area nor to the adjacent residential populations.

There will be an increase in noise levels as a result of Project operations at Mengapur. The sources of noise will include mining and ore haulage, drilling and blasting and the workings of the process plants. The Mengapur area is currently not subject to any significant or continuing sources of noise or vibration. Background night-time noise measurements confirmed the levels at the boundaries of MMC's proposed mining lease were typically in the order of 51 to 56 dBA.

These levels were thus at, and in some instances already exceeding the DOE's guidelines for night-time noise levels.

The increase from the incoming Project related population over a relatively short period of time to the populated centres surrounding the Project site will result in a range of impacts occurring to these surrounding towns. The additional accommodation demand would be a significant increase to the existing demand of the nearby towns. In particular the town site of Seri Jaya and Maran with their current small population are most likely to be the towns for which the majority of the workforce will choose to live in. Arrangements between the proponent and the Government will be aimed at the effective management of residential and other land releases. There will be no significant impact of population increase for the Kuantan area.

Few adverse economic and employment impacts associated with the Mengapur Project can be identified. The economic effects of the Project will be felt throughout the Pahang and National economies. The Project will result in a total of 500 jobs during the operational phase alone, excluding contractors.

It is expected that when the Project proceeds, traffic flows from non-commercial sources on the section of the highway from Maran to Seri Jaya and along the Mengapur access road could increase significantly as a result of the Project related workforce.

Commercial traffic resulting from the transport of the copper and pyrrhotite concentrates will also increase traffic levels, however this impact is not expected to be significant as the trucking campaign will be well within the vehicle capacity design for each transportation route used.

Rural areas are located to the outer periphery of MMC’s proposed mining lease. Neither agricultural land nor any villages are contained within the proposed lease. To the east and the north are the Asia Jaya Sepakat and the Pahang Palm Oil Estates. Both are private estates with a small resident population within the plantations. These are the only agricultural areas and rural settlements within close proximity to the proposed mine.

Information collected during base-line biological studies indicated that the present land use activities within the study area were resulting in some perturbations to the surrounding environment. Human interference with the natural physical environment includes the establishment of palm oil plantations and processing facilities, and the harvesting of timber products.

Environmental parameters recorded suggest that the ecosystem is not in an undisturbed or pristine condition.

In 1990 an Environmental Impact Assessment (EIA) was conducted by D C Blandford & Associates Pty Ltd and later this was revised by Mr Myles Hyams of Normet Engineering Pty Ltd in early 1992. The detailed reports and supporting data including the engineering design and environmental management strategies proposed are set out in Volumes 6 to 8 of the DFS and EIA report (Volumes 1 & 2) by Normet, February 1992.

The Department of Environment (DOE) approved the revised ETA in October 1991. The mining title application has been lodged by MMC over the Mengapur area and is now being processed by the relevant State Authorities.

Further review on the project feasibility to enhance its viability would entail further processing of sulphuric acid to phosphoric acid for export. Phosphate rock will be used in this process which has to be imported. As such the acid plants have to be sited as near as possible to the Kuantan Port. An area of 200 acres located adjacent to the Kuantan Port has been identified for this purpose and the Pahang State has agreed in principle to MMC's application. A separate EIA report for this area will be prepared and submitted to the DOE for final approval.

21 Capital and operating costs

21.1 Project implementation costs

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

For the project implementation, some costs have been estimated to cover the MMC management costs involved in mine development and commissioning the acid plants. The bulk of work during this period will consist of procurement, tendering and contract management, monitor and supervision of the development and construction work prior to production and actual commissioning of plant operations.

The total project implementation costs are estimated at US$4.20 million and are spread over the two years of construction and first year of operation/production(Table 21.1).

Table 21.1 Project implementation costs

Project Implementation (US$'000)	Construction		Production
	Year 1	Year 2	Year 1
1. Mine and Concentrator	400	400	400
2. Roaster/Sulphuric-Phosphoric Acid Plants	1,000	1,000	1,000
TOTAL	1,400	1,400	1,400

21.2 Capital costs

The capital cost estimates are based on the studies undertaken and quotations provided by various technical consultants and contractors. Among these consultants and contractors are Normet Engineering Pty Ltd, Lurgi GmbH and Simon Carves.

The capital costs are estimated in 1993 dollars and wherever possible and appropriate, costs have been prepared on the basis of local manufacturing and construction input.

The total capital cost amounts to US$179.44 million and comprises the following major components (Table 21.2).

Table 21.2 Capital cost estimates

Item	Capital Cost (US$ Million)
Mine facilities on the contract mining basis	4.00
Mine development (pre-stripping)	13.89
High Grade Oxide Ore Treatment (Supergene Ore)	1.55
Concentrator Plant	35.00
Roaster/Sulphuric Acid Plant DFS (Normet/Lurgi) (US$82.29 M) Simon Carves (US$78.90 M)	80.00
Phosphoric Acid Plant	45.00
Total	**179.44**

It is estimated that about 55% of the costs in acid plant construction is based on preliminary local manufacture and construction input. These figures could be fine tuned further when final tanker quotations are given.

21.3 Operating costs

Operating cost estimates have been prepared by Normet Engineering Pty Ltd and its sub-consultants, as well as Lurgi GmbH and MMC.

The operating costs are derived on the basis of unit production costs.

The operating costs including tribute but excluding depreciation average US$59.01 million per annum (1993 constant money) for the project involving mining and on-processing to phosphoric acid. For the project involving mining alone, the operating costs would average US$19.24 million per annum. The operating costs over the first five years for both the operations are listed in (Table 21.3).

Table 21.3 Operating Costs

Operating Costs US$ Million		
Year	**Mine & Acid Plants Combined**	**Mine Only**
1	65.22	24.85
2	61.87	21.86
3	58.21	19.30
4	57.03	18.12
5	57.01	18.10

22 Economic analysis

22.1 Revenue

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

Revenue forecasts have been prepared and wherever possible based on DFS and other marketing reports provided by consultants. Results of recent market investigations and visits by MMC personnel have also been included in the computation of project revenue.

The revenue forecasts for the products of the Mengapur project over the operating period are tabulated in (Table 22.1).

Table 22.1 Revenue Forecasts

Revenue US$ '000 Constant Money Terms							
Product	Year						
	1	2	3	4	5	11	23
Copper concentrate	13,819	14,861	13,984	13,213	11,930	9,793	10,601
Copper cement	9,627	2,353	536	722	978	2,409	0
Gold	1,227	1,778	1,823	1,688	1,486	1,789	2,690
Silver	266	276	267	239	235	305	281
Phosphoric Acid	68,005	68,005	68,005	68,005	68,005	68,005	68,005
Phosphogypsum	9,000	9,000	9,000	9,000	9,000	9,000	9,000
Calcine	3,300	3,300	3,300	3,300	3,300	3,300	3,300
Surplus Electricity	2,600	2,600	2,600	2,600	2,600	2,600	2,600
Excess Sulphuric Acid	1,200	1,200	1,200	1,200	1,200	1,200	1,200
TOTAL	**109,043**	**103,373**	**100,715**	**99,967**	**98,733**	**98,401**	**97,677**

The average revenue generated from one tonne of ore mined over the first 5 years and two of the later years of operations are shown in Table 22.2.

Table 22.2 Revenue per ore mined

Revenue in US$/t ore mined (constant money terms)			
Year	**Mining as a single entity**	**Mining and acid plants combined**	**Increase (%)**
1	11.04	34.90	316
2	10.96	37.94	346
3	11.91	44.06	370
4	12.39	46.80	378
5	10.36	40.51	391
11	7.79	33.23	427
23	8.07	32.59	404

The above comparison demonstrates that on-processing to produce phosphoric acid has increased the average revenue from one tonne of ore mined by more than three folds when compared with the case in which no on processing is considered.

22.2 Economic analysis

This economic evaluation assesses the viability of the proposed Mengapur project on the basis of mining the Mengapur ore to produce copper concentrate/cement for sales and pyrrhotite concentrate for processing to phosphoric acid for export. However, the relative viability of the stand alone project based on mining as a separate entity and the acid plants as another is also assessed to identify the sensitive factors governing the economics of the overall Mengapur project.

The economic evaluation is undertaken on the basis of discounted cash flow (DCF) techniques and 1993 constant money terms to compute the Discounted Cashflow Internal Rate of Return (IRR or DCF yield) and Net Present Value (NPV).

Evaluation parameters

The financial analysis of the Mengapur project has incorporated the major parameters as presented in (Table 22.3) and (Table 22.4).

Table 22.3 Ore and Waste tonnage production

Item	Tonnes (million)
Total sulphide ore mined	63.2
Total High Grade Leachable Ore Mined	2.3
Total Waste Mined (excluding pre-stripping)	81.9
Total Material Mined	147.4
Strip Ratio	1.4

The sulphide ore is processed at an average rate of 2.75 Mtpa, ranging from 2.04 to 3.00 Mtpa.

Table 22.4 Average ore grades and recovery over the Project life

Element	Grade	Recovery (%)
Cu (Sulphide ore)	0.27%	90
Cu (Oxide ore)	1.29%	60-90
S	8.67%	82
Au	0.21 g/t	29
Ag	0.26 g/t	27

- Cu, Au and Ag are recovered in copper concentrate from sulphide ore.
- Cu is also recovered in copper cement from HGLO.
- S is recovered in pyrrhotite concentrate, together with Fe.
- Average copper concentrate production: 30,000 tpa. Average pyrrhotite concentrate production: 600,000 tpa.
- Duration of operation: 23 years +.
- Capital and operating costs have been provided in Section 21.
- All costs and prices are based on constant money terms.
- Project financing is based on all equity. Gearing will enhance the project return.

This is based on straight line over 10 years on depreciable assets and no residual value of assets is assumed.

Taxation

The following represents the current tax structure for the Mengapur project (Table 22.5).

Table 22.5 Monument applicable tax & fee 2012

Corporate Income Tax	Rate	Note & Reference
Resident Co. with paid up capital <2.5M		
1st RM500K chargeable Income	20%	
Subsequent chargeable Income	25%	
Resident Co. with paid up capital >2.5M	25%	
Sale Tax		
Not applicable for export goods		
Salary - Co. Contribution		
Employee's Provident Fund		
if salary =<5000 and age= <55	13%	
if salary =>5000 and age= <55	12%	
if age >55	6%	
Property Fee		
Prospecting Licence <400Ha	RM20/Ha	Annual Fee
Exploration Licence >400Ha	RM10/Ha	Annual Fee
Mining Lease	RM100/Ha	Annual Fee
Rehabilitation Fee - Applicable to Mining Lease	RM10,000/year	Annual Fee
Royalty		
Gold	5%	net smelter return
Silver	5%	
Copper	5%	
Iron Ore	5%	

Summary of cash flow results

The results of the base case economic evaluation on the basis of equity are tabulated in Table 22.6 and Table 22.7.

Table 22.6 Summary of Cashflow Results

Capital Outlay	Mine & Acid Plants Combined US$ 179.44 million	Acid Plants Only US$ 125.00 million	Mine Only US$ 54.44 million
Net Cashflow (US$	586.26	513.84	72.42
NPV (US$ million)			
@ 8.5% discount rate	153.96	145.1	8.86
@ 10% discount rate	119.89	115.93	3.96
@ 12% discount rate	83.71	84.93	(1.22)
@ 15% discount rate	43.75	50.65	(6.89)
DCF yield (IRR)	20.2%	23.0%	11.5%
*Payback Period (years)	4.1	3.8	6.0

** based on production years*

Table 22.7 Mengapur Project cashflow summary

Category	Item	Unit	Combined	Acid Plants	Mine
1.0 Raw Material (quantity)	Pyrrhotite Concentrate	kt	14,289	14,289	
	Phosphate Rock (2.99t/t of P_2O_5)	kt	13,960	13,960	
	Cooling Water (66.00cu m/t of P_2O_5)	'000 cu m	308,154	308,154	
	Process Water (5.00cu m/t of P_2O_5)	'000 cu m	23,345	23,345	
	Chemicals (10.50 kg/t of P_2O_5)	'000 kg	49,025	49,025	
	Power (159KWh/t of P_2O_5)	'000 KWh	742,371	742,371	
2.0 Production	Total Waste mined	'000 t	81,934		819,344
	Total Sulphide to Concentrator	'000 t	63,170		63,170
	Total High Grade Oxide to Heap leach	'000 t	2,344		
	Total Material mined	'000 t	147,448		
	Pyrrhotite concentrate	'000 t	14,289		
	Copper concentrate	'000 t	694		
	Payable Copper in concentrate	t	146,569		146,569
	Payable Copper in cement ex HGLO	t	21,921		21,921
	Payable Gold in concentrate	Kg	3,737		3,737
	Payable Silver in concentrate	Kg	40,188		40,188
	Phosphoric Acid as P205	'000 t	4,669		
	Gypsum	'000 t	20,700		
	Calcine/Cinder	'000 t	12,650		
	Surplus Electricity	'000 MWh	1,495		
	Excess Sulphuric Acid	'000 t	690		
3.0 Gross revenue		US$'000	2,279,777	1,934,415	588,275
4.0 Operating costs	4.1 Mine	"	441,753		441,753
	4.2 Sulphuric Acid Plant	"	155,390	398,303	

Category	Item	Unit	Combined	Acid Plants	Mine
	4.3 Phosphoric Acid Plant	"	759,222	759,222	
	TOTAL OPERATING COSTS	"	1,356,365	157,525	441,753
5.0 Operating profit		"	923,412	776,890	146,522
6.0 Depreciation & Depletion		"	160,000	125,000	35,000
7.0 Taxable profit.		"	763,412	651,890	111,522
8.0 Taxation	34.00%	"	172,504	149,810	22,694
9.0 Profit	After tax & D/D	US$'000	590,908	502,080	88,828
		RM'000	1,477,271	1,255,200	222,071
10.0 Non-cash Items		US$'000	160,000	125,000	35,000
11.0 Total Cash Inflow		"	765,696	638,839	126,857
12.0 Total Capital Outlay		"	179,436	125,000	54,436
13.0 Net Cashflow		"	586,260	513,839	72,420
		RM'000	1,465,649	1,284,598	181,051
14.0 NPV	@ 8.5%	US$'000	153,963	145,103	8,859
	@ 10%	"	119,885	115,929	3,957
	@ 12%	"	83,707	84,928	-1,221
	@ 15%	"	43,751	50,645	-6,894
15.0 DCF	yield (IRR)		20.20%	22.95%	11.48%
16.0 Payback Period	(from production year)	Years	4	4	6
17.0 Payback Period	(from development year)	Years	6.13	5.75	8.04

The above evaluation results demonstrate that the project viability is greatly enhanced by the further processing of the mine products to produce phosphoric acid and by-products.

(Figure 22.1) and (Figure 22.2) depict the net and cumulative cashflows over the entire operations for the mine and acid plants combined project.

Figure 22.1 Net Cash Flow

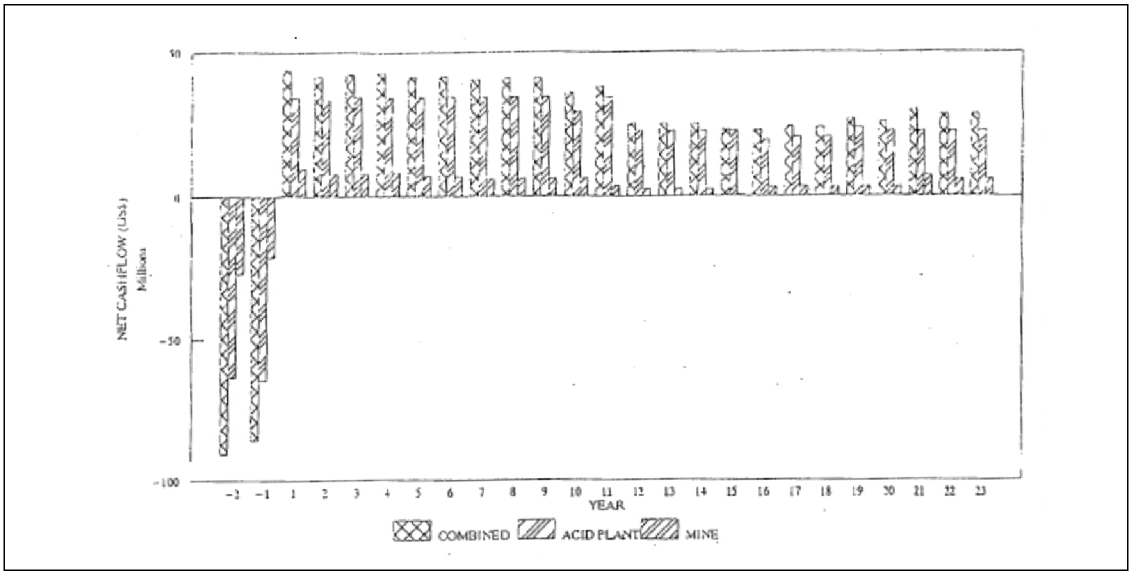
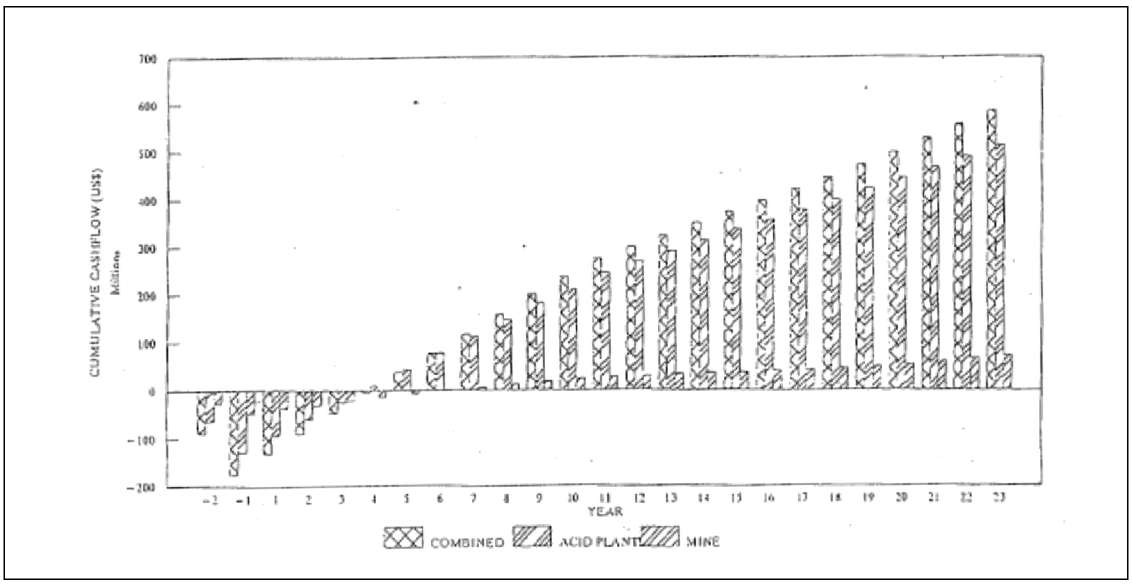


Figure 22.2 Cumulative Cash Flow



The results show that reasonably good returns would be achieved even if the mine and acid plants are considered as two separate stand alone projects. In any case, the mine and acid plants stand alone projects are mutually dependent and complementing each other and therefore they cannot stand alone in the real sense.

The average annual net cashflow generated during the first 5 years of full operation is estimated at US$42.5 million. For the second 5 years, it is US$40.0 million whilst the average over the whole project life is US$33.1 million.

During the initial two years of construction and mine development, revenue would be realised from the leaching of high grade leachable oxide ore; an operating profit of US$3.9 million would be generated in the second year.

The analysis of operating costs for the whole project life reveals the following (Table 22.8).

Table 22.8 Operating cost percentages for life of project

Operation	Cost (%)	Cost (%)
Mine		
Mining	13.6	
Concentrator Plant	14.0	
Others	5.0	32.6
Sulphuric Acid Plant		
Pyrrhotite Transport/Handling	6.3	
Roaster/Acid Plant	4.2	
Others	0.9	11.4
Phosphoric Acid Plant		
Phosphate rock	43.2	
Plant Operations and Others	12.8	56.0
TOTAL	100.0	100.0

The sources of revenue for the first 5 years of full scale production and entire operation life expressed as percentages of the total are listed in (Table 22.9).

Table 22.9 5 year sources of revenue

Sources of revenue	First 5 year Operation		Entire life operation	
	%	%	%	%
Mine				
Copper Concentrate	13.25		11.03	
Copper Cement	2.78		1.99	
Gold	1.58		1.84	
Silver	0.25	17.86	0.29	15.15
Sulphuric Acid Plant				
Calcine	3.22		3.33	
Surplus Electricity	2.54		2.62	
Excess Sulphuric Acid	1.16	6.92	1.21	7.16
Phosphoric Acid Plant				
Phosphoric Acid	66.43		68.61	
Phosphogypsum	8.79	75.22	9.08	77.69
TOTAL	100.00	100.00	100.00	100.00

The average revenue generated from one tonne of ore mined over the first five years and two of the later years of full production operations (Table 22.10) are compared to revenue obtained by processing the pyrrhotite by-product and production of acids.

Table 22.10 Revenue from one tonne of ore

Revenue US$/t ore mined (constant money)			
Year	**Mining as a single entity**	**Mining and acid plants combined**	**Increase (%)**
1	11.04	34.9	316
2	10.96	37.94	346
3	11.91	44.06	370
4	12.39	46.8	378
5	10.36	40.51	391
15	7.79	33.23	427
23	8.07	32.59	404

The above comparison demonstrates that processing to produce phosphoric acid has increased the average revenue from one tonne of ore mined by more than three folds when compared with the case in which no processing is considered.

Sensitivity

A sensitivity analysis has been carried out to establish the various variables that will have significant impact on the feasibility of the Mengapur project. These variables in their order of importance are listed as follows:

- Phosphoric acid selling price.
- Phosphate rock purchase price.
- Contract mining cost.

The overall project returns are sensitive the selling price of phosphoric acid as shown in (Table 22.11).

Table 22.11 Phosphoric Acid vs IRR price sensitivity

Phosphoric acid price		IRR
US$/t P_2O_5	**% Variance**	**%**
368.50	+10	23.6
351.75	+5	21.9
335.00	Base Case	20.2
318.25	-5	18.4
301.50	-10	16.6

Should the phosphoric acid price drop by 10%, the IRR would be reduced by about 18% of the base figure to 16.6%.

The project IRRs are influenced by the purchase prices or phosphate rock (Table 22.12).

Table 22.12 Phosphate rock vs IRR price sensitivity

Phosphate rock price		IRR
US$/t rock	**% Variance**	**%**
46.2	+10	18.9
44.1	+5	19.5
42.0	Base Case	20.2
39.9	-5	20.9
37.8	-10	21.5

It is noted that the IRR would be reduced to 18.9% from 20.2%, i.e. a drop of only 6%. Hence, the analysis shows that the project IRRs are less sensitive to the phosphate rock purchase price as compared to the phosphoric acid selling price.

In the event that the phosphoric acid price is 10% lower than the base case while the phosphate rock purchase price is 10% higher than the base case, then the IRR generated will be 15.2%. However, it is unlikely that the above will happen considering that the phosphoric acid selling price should correspond with the cost of purchase of the raw material, phosphate rock.

The effect of the contract mining cost on the overall project viability is shown in (Table 22.13).

Table 22.13 Contract mining cost vs IRR price sensitivity

Contract mining cost		IRR
US$/t	**% Increase**	**(%)**
1.25 (Base Case)	0	20.2
1.3	4	19.9
1.4	12	19.4
1.5	20	18.9

Judging from the above sensitivity results, it is apparent that the project IRR is quite resilient to the increase in the contract mining cost.

23 Other relevant data and information

This report has been created through the assimilation and compilation of information from other reports, most notably the Mengapur Project Feasibility Study Report published in May of 1993, and as such the analysis and conclusions reported herein from those documents represent historic data and any estimates of ore reserve or resources should be considered as such.

24 Interpretations and conclusions

This technical report represents a compilation of historic information and data that has been provided to Snowden by Monument and all economic assessments and resource statements included in this report are considered historic in nature and there is no certainty that any economic assessments will be realized.

From the 1993 Mengapur Feasibility Study Report:

- The Mengapur project with the processing operations to produce phosphoric acid and by-products like phosphogypsum and calcine will enable MMC to create significantly new industries for the Malaysian economy in addition to generating employment opportunities and export earnings.
- The project life could be extended beyond the 23 years taking cognizance of the measured and indicated resource at Mengapur as evaluated.
- The low grade oxide ore, totalling in excess of 10.0 Mt averaging 0.34% Cu in the current optimised pit, which would have to be dumped as waste in the proposed project, could be treated by heap leaching method during the mining operation, should the testwork to be continued improve the Cu recovery to 60-70%. The current inconclusive testwork done by Bio-Solutions Sdn Bhd indicate a Cu recovery approaching 30%. Based on further enquiry and visit by MMC personnel, it is believed that a viable leaching technique could be found overseas where commercial leaching of several refractory copper deposits had been proven profitable. The additional copper that would be recovered by leaching of low grade oxide ore will add further revenue to the project.
- 65 Mt of oxide in Zone C have potential to be leached on a commercial scale with further studies. This will add tremendously to the revenue from the mine since the infrastructure would already be in place.
- There is a vast potential for further downstream manufacturing to be considered for investment. These include the production of fertilizers such as Di-ammonium Phosphate (DAP) utilising Mengapur's phosphoric acid. This downstream manufacturing option could be viable should the proposed ammonia plant by PETRONAS in Peninsula Malaysia be established.
- Setting up of an Integrated Iron and Steel Plant using calcine/cinder as feed could also be another downstream investment for consideration.

It can be concluded that this project is economically viable and technically proven with ready markets for the main product, phosphoric acid. The project also has the potential for downstream fertilizer production.

Accordingly, upon MMC's Board approval, the following steps are recommended to be taken:

- Formation of a Task Force for project development and implementation.
- Secure Agreements for steady supply of phosphate rock at competitive price. MMC should look into acquiring equity in such mines.
- Carry out Environmental Impact Assessment study for the new site of the acid plants near Kuantan Port.
- Conclude Sales Contracts for:-
  - Phosphoric Acid
  - Copper Concentrates
  - Gypsum and Calcine

  - Excess Sulphuric Acid and Electricity
- Follow-up on Contract Mining.
- Raise project financing, including seeking of Joint Venture partners.

25 Recommendations

The historic data compiled in this report indicates the need for more preliminary test work to be completed before the project is ready to move forward. The resource and reserve areas identified in the Normet 1990 report must be drilled using CIM 2005 standards.

The recommended work plan at Mengapur includes acquiring the land rights to conduct exploration and mine development studies. A first work phase is recommended consisting of due diligence work completed mostly from August 25 to November 25, 2011 at an approximate cost of CAN$788,000. A second work phase includes a 1.6 year drill hole program at an approximate cost of CAN$13.4M using three diamond drill rigs and one RC rig to complete a total of 65,980 m of resource conversion and infill drilling (at a 40 m average drill hole spacing for planning purposes) at an approximate cost of. The total work program is estimated to cost CAN$14.2M and assumes that the diamond drill production is 30 m per 24-hour work shift. The second phase of work should only be performed if the first due diligence phase is successful.

Included in this 1.6 year drilling program is access road and drill pad construction, metallurgy testing on the sulphide and oxide ores consisting of flotation testing, grind testwork for sulphide ores, and leach tests (bottle roll and columns) for oxide ores. Work will also include geological interpretation and mine design modelling, assaying for Au, Cu, Ag, and S along with multi-element ICP, quality assurance and quality control (QAQC) assay program, and contract topographic survey work (air and ground)

The topographic map surveys will be done early to establish good ground control. Conversion of the Cassini grid to Rectified Skew Orthomorphic (RSO) will be pursued. Early drilling will prioritize the A Zone area as this will likely be the location of the starter pit (first 3 to 5 years of mining). A later phase of drilling is envisioned to focus on the B and C Zone resource areas. The metallurgy testwork will proceed in the due diligence period and continue afterwards into 2012 with sulphide variability flotation ore testwork and column leach tests and bottle roll tests for oxide ores.

Table 25.1 Proposed work programmes

Proposed Work	Unit Cost ($CAD)	Approximate Cost ($CAD)
Aerial Topographic survey: Air photograph, Lidar, DTM, 5 m contours, 20,000 Hectares	$10.66 per Ha	$213,280
Resource Conversion and Infill Drilling: on approximate 40m drill hole spacing: A Zone: 116 DDH drill holes totalling 34,130m B Zone: 52 DDH drill holes totalling 17,050m C Zone: 71 RC drill holes totalling 14,800m Due Diligence: 12 diamond drill holes 2,921m Total: 239 drill holes totalling 65,980 m	Diamond Drilling at $150 per meter direct drill cost; $3,594 for each drill move (50) and setup and overburden drilling; mobilization charges and driller's logs $40,000 total for job (5 drills)	Due Diligence Diamond Drilling = $525,000 (all costs) Diamond drilling (post due diligence) =$7,896,700
Drill supplies: muds, concrete, pvc pipe, bits, core boxes, etc. or $1,875/month	RC drilling at $75 per meter direct drill cost; $3,000 for each drill move (60) and setup; mobilization charges and drillers logs $20,000	RC drilling = $1,310,000 Drill Supplies = $350,000
Drillers Expenses: living and food at site	$20/day * 1,706 drill days $20/day * 120 drill days (DD)	$34,120 $2,400
Drill Hole Assaying (Primary Lab) for Cu, Leco sulphur, Au, Ag, and 50-element ICP for sulphide and oxide ores and cyanide soluble Au-Ag for oxide ores; assume 3 m average drill hole assay widths (Includes due diligence drilling at 2 m samples)	$60.71/sample (sulphide and oxide) x 17,060 samples;	$1,035,710
	$18.23/oxide sample (leach) x 10,365 samples (3 m composites);	$188,950 $94,900
	Due diligence assays: $65 x 1,460 samples and $18.23 x 325	$5,925
Drill Hole Assays (Secondary Lab): Duplicate pulp checks for Cu, sulphur, Au, Ag, and multi-element ICP (4 acid digestion); assume 400 samples per year (Includes Due Diligence Drilling); also done for due diligence	$51.54 per sample X 400	$20,616
	$51.54 per sample x 1,100 for due diligence	$56,694
QAQC assays (including standards and blanks one every 20 samples	860 standards X $65	$55,900
Cu-S-Au-Ag standards: insert one every 20 samples	860 x $8.50 each	$7,310
Down Hole Survey tool: Isgyro + software and hardware	$125,000 for tool; other moneys for software and	$145,000

Proposed Work	Unit Cost ($CAD)	Approximate Cost ($CAD)
(Algiz 7 software, centralizer, winch, etc.)	hardware support	
Aerial Geophysics (electromagnetics and EM- DIGHEM) 20,000 hectares (2,235 line km)	$172/line km Mobilization charge	$384,420 $50,000
Metallurgy Test Program: 2 Bulk sulphide samples (grinding and flotation tests) for Due diligence	Grinding and flotation tests; 2 bond ball mill work index values	$61,790
Metallurgy Variability Test Program: 4 Sulphide and 2 oxide (column leach) samples	Flotation and grinding = $32,000 2 column leach tests & bottle roll tests =$50,000	$82,000
Metallurgy Test Program: Oxide Samples: bottle roll leach tests (Due diligence)	2 leach tests	$20,858
Road and Drill Pad Construction: bulldozer, excavator, water truck	$9,375/month x 23 months	$215,625
6 Geologists – includes meals, travel, and housing expenses ;	$50,000 month x 23 months	$1,150,000
Due diligence: 5 geologists	$20,000 month x 4 months	$80,000
Air Flights	$100,000	$100,000
Trucks/Fuel: 4 trucks	Each truck cost $1,030/month rental X 23 months	$23,690
Contract topographic surveying (previous open pit mining areas)	132 acres	$31,680
Camp Upgrade existing 20-person camp to 32 person capacity	$28,125 for additional trailers and office quarters	$28,125
Geologic model (block model estimation) and open pit mine design	Two Contract Mine Engineers	$60,000
TOTAL		$14,230,695

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27 Date and signatures

Name of Report:

Mengapur Project – Technical Report

Effective Date

25th November 2011

Issued by:

Monument Mining Limited

[R.D. Carlson] R Carlson

Mr R Carlson MAIG

November 25, 2011
Date

[W Dzick] Walter A Dzick

Mr W Dzick P.Geol. AIPG AusIMM

November 25, 2011
Date

28 Certificates

CERTIFICATE of QUALIFIED PERSON

(a) I, Roderick David Carlson, Principal Consultant of Snowden Mining Industry Consultants Pty Ltd., Level 15, 300 Adelaide St., Brisbane, Queensland, Australia, do hereby certify that:

(b) I am the co-author of the technical report titled Mengapur Project – Technical Report and dated 25th November 2011 (the 'Technical Report') prepared for Monument Mining Limited.

(c) I graduated with the following degrees BSc. (Geology), Canberra College of Advanced Education (1986), MSc. (Ore Deposit Geology and Evaluation), University of Western Australia (1998)

I am a Member of the Australia Institute of Geoscientists.

I have worked as a geologist continuously for a total of 24 years since my graduation from university. I have particular experience in sampling, QAQC, regolith interpretation, and resource estimation.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in mining and Resource evaluation consulting practice for 14 years.

(d) I have made a current visit to the Mengapur Project from 7th July 2011.

(e) I am responsible for the preparation of the sections of the Technical Report as detailed in (Table 2.1).

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

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

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

Dated at Brisbane, Queensland, Australia this 25 November 2011

R Carlson

Roderick Carlson, BSc, MSc, MAIG

CERTIFICATE of QUALIFIED PERSON

(a) I, Walter A Dzick, Principal Consultant of Snowden Mining Industry Consultants Pty Ltd., 600 - 1090 West Pender St., Vancouver, British Columbia, Canada, do hereby certify that:

(b) I am the co-author of the technical report titled Mengapur Project – Technical Report and dated 25th November 2011 (the 'Technical Report') prepared for Monument Mining Limited.

(c) I graduated with the following degrees BSc. (Geology), New Mexico State University (1978), M.B.A., University of Nevada Reno (2007)

I am a Member of the American Institute of Professional Geologists (AIPG) and the Australian Institute of Mining and Metallurgy (AusIMM).

I have worked as a geologist continuously for a total of 30 years since my graduation from university. I have particular experience in mine geology, QAQC, near mine exploration, and resource estimation.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in mining and Resource evaluation consulting practice for 14 years.

(d) I have not made a current visit to the Mengapur Project.

(e) I am responsible for the preparation of the sections of the Technical Report as detailed in (Table 2.1).

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

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

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

Dated at Vancouver, British Columbia, Canada this 25 November 2011

Walter A Dzick

Walter A Dzick, BSc, MBA, AIPG