Bema Gold Corporation

BEMA GOLD CORPORATION

Technical Report
Summarizing the Kupol Project Feasibility Study, Chukhotka Okrug, Russia

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GA Project No. BEM003

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SUMMARY (Item 3)

On December 18, 2002, Bema Gold announced that it had completed the terms of a definitive agreement with the Government of Chukotka, an autonomous Okrug (region) in northeast Russia, to acquire up to a 75% interest in the Kupol gold and silver project.

The Kupol deposit is located approximately 430 km northwest of Anadyr, 200 km southeast of Bilibino / Keperveyem, 320 km south of Pevek and 1250 km northeast of Magadan. Kupol can be accessed by winter road and by helicopter; an airstrip is under construction for fixed wing aircraft.

Preliminary Economic Assessment (PEA) to Feasibility

Bema Gold completed a Preliminary Assessment of the property in 2004. The results of this assessment indicated that the project should advance to the feasibility level. This document is a full feasibility study of the Kupol Project. The results of the feasibility study summarized herein indicate that the development and production of precious metal ores from the Kupol Deposit is feasible. Total Cash Cost per Ounce of gold is below $100/ounce net of by-product silver credits and the project pays back the initial capital cost of $407,000,000 in less than 2 years based on a $400/ounce gold price and $6.00 silver price. The initial production life of the deposit is approximately 7 years based on Probable Mineral Reserves, with mill production beginning in June 2008. The exploitation of the inferred mineral resources may extend the life of the project.

Feasibility Management

Bema Gold managed the feasibility study efforts leading to this document. Several independent firms were commissioned to provide specific work packages. Bema's Russian entities, the Julietta Mine and the Magadan office in the Magadan Oblast contributed their Russian mining and administrative expertise to major portions of the study. Bema staff, exploration, operations and engineering played a key role in providing input. The majority of the contributors to the Kupol Feasibility have been to the Kupol site.

In addition to the development of the feasibility study, Bema Gold will develop and in some cases has developed several documents required under Russian guidelines.

Feasibility Results (All USD unless otherwise stated)

Preproduction Capital Costs

Preproduction Costs	USD (Millions)
Design Engineering	$4
Owners Site Construction (includes certain inventory items)	$113
Orocon Site Construction	$167
Surface Mining Costs	$19
Underground Mining Costs	$22
Processing Costs	$3
Site Services Costs	$2
General and Administrative	$7
Sub-Total Preproduction Costs	**$337**
Other Costs	
VAT Tax (Supplies and Fuel)	$42
Customs Duties & Clearance	$4
Property Tax & Other	$1
Sub-Total Other Costs	**$46**
Total Preproduction and Other Costs	**$383**
Capital and Inventory (inventory change not actual inventory value) Expenses	**$24**
Total Preproduction Capital Costs	**$ 407**

Items of Note

- Total Preproduction Capital Payback ($407 Million) less than 2 years
- Revenue and Cost Parameters
    - Gold @ $400/oz
    - Silver @ $6.00/oz
    - Fuel @ $40/Barrel
    - Exchange Rates
        - Ruble:USD of 30:1
        - CDN:USD of 1.2 to 1.0
        - Euro:USD of .75 to 1.0
    - Logistics – known costs and benefit of learning curve
    - Labor – Russian Labor February 2005 increased by 8%
- Project Statistics for Probable Reserves – 7.1 Million Tonnes (7 years)
    - $47/oz of gold Cash Operating Cost*
    - $88/oz of gold Total Cash Costs*
    - $730 Million Cumulative Cash Flow
    - $430 Million NPV @ 5%
    - 552,000 Ounces Gold Average Annual Production

    * Net of Silver Credits

The Kupol exploration effort continues. Gustavson considers it likely a substantial portion of the present 4 million tonnes of inferred resources may be converted to probable reserves from further exploration activity and during the exploitation of the existing probable reserves. No major capital, with the exception of continued underground development, will be required to mine beyond the present probable reserves.

Geology, Resources & Reserves

Quartz veins at the Kupol site were discovered in 1966 by a Soviet Government Exploration Expedition and the main vein was discovered in 1995 by the Bilibino based, State funded Anyusk Geological Expedition. Gold and silver mineralization at Kupol is hosted by poly-phase brecciated quartz-adularia veins. The extent of brecciation is variable but generally higher-grade zones are more strongly brecciated. The Kupol deposit can be classified as a low sulphidation epithermal fissure vein type deposit. The silver to gold ratio is on average 12 to 1. Between 1996 and 2002 a limited amount of exploration work was conducted, in 2003, Bema commenced extensive drilling and sampling programs. The following table summarizes the number and meterage for diamond drilling, trenching and channel sampling completed by Bema during 2003 and 2004.

Year	No. of Diamond Drill Holes	Drill Hole Meterage (m)	No. of trenches	Trench Meterage (m)	No. of Channel Samples	Channel Sample Meterage
2003	166	22,200	15	2500	--	--
2004	309	53,800	2	226	87	700

The 2004 exploration activities included diamond drilling, trenching and channel sampling. The diamond drill program was focused on results for exploration, infill (including detailed vein configuration), geotechnical and condemnation work over the entire 3.6 km strike length of the deposit. Overburden removal, mapping and channel sampling was completed in three separate areas. The channel samples and a set (63 drill holes) of closely spaced drill holes provided detailed information on vein continuity and was used for assessing dilution and ore loss factors. Additional testing included 3200 wax-coated density measurements in vein and waste lithologies and an extensive program to define metallurgical domains and confirm recoveries and consumption of reagents.

Mineral Resources are categorized using the classification of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM 2000). In situ Indicated and Inferred Resources for Vein, above a 6g/t cutoff, as published by Bema are as follows:

Resource Class	Tonnes (1000)	Au(g/t)	Contained Au (Ounces X 1000)	Ag(g/t)	Contained Ag (Ounces X 1000)
Indicated	6,403	20.33	4,184	257	52,911
Inferred	4,090	12.45	1,637	171	22,539

The resources reported above do not include dilution or ore-loss.

The total Probable Mineral Reserve outlined by the open pit and underground plans is 7.1 million tonnes at a grade of 16.9 grams / tonne gold and 214 grams / tonne silver. The distribution of the Probable Mineral Reserve is shown below:.

Production (includes dilution and ore loss)		
Open Pit	Tonnages (millions)	1.42
	Au (g/t)	20.4
	Ag (g/t)	193
Underground	Tonnages (millions)	5.66
	Au (g/t)	16.0
	Ag (g/t)	219
Total	Tonnages (millions)	7.09
	Au (g/t)	16.9
	Ag (g/t)	214

Open Pit Mining

Open Pit mining methods during the first 4 years of the project will be used to mine the ore on a two shift per day, 340 days per year schedule (25 days per year are not scheduled to allow for weather delays), at a rate of approximately 1,000 tonnes per day of ore and 11,500 tonnes per day of waste. Low Grade ore will be stockpiled and delivered to the mill in the later years of the project's life. The waste to ore strip ratio is 12 to 1. The average dilution in the final pit model is 22%. The production schedule for the Open Pit is:

OPEN PIT PRODUCTION		2007	2008	2009	2010
	High Grade Ore Pit Production	178,628 t	229,895 t	268,454 t	273,222 t
	Au Grade	21.2 g /t	26.8 g /t	36.9 g /t	22.7 g /t
	Ag Grade	182 g/t	230 g/t	352 g/t	229 g/t
	Low Grade Ore Pit Production	117,030 t	119,304 t	56,194 t	182,968 t
	Au Grade	6.3 g /t	6.3 g /t	6.8 g /t	6.5 g /t
	Ag Grade	57 g/t	64 g/t	69 g/t	79 g/t
	Total Ore Pit Production	295,658 t	349,199 t	324,648 t	456,190 t
	Au Grade	15.3 g /t	19.8 g /t	31.7 g /t	16.2 g /t
	Ag Grade	133 g /t	173 g /t	303 g /t	169 g /t
	Waste Pit Production (2006 waste included with 2007)				
	Total Waste	6,490,000 t	4,140,000 t	3,532,000 t	2,749,000 t
	High Grade Ore per Day ~	525 t /d	675 t /d	790 t /d	800 t /d
	Low Grade Ore (Stockpiled) per Day ~	350 t/d	350 t/d	160 /d	540 t/d
	Waste Tonnes per Day ~	12,000 t/d	12,000 t/d	10,400 t/d	8,000 t/d
	Total Rock Tonnes per Day ~	12,875 t/d	13,025 t/d	11,350 t/d	9,340 t/d

Underground Mining

Underground mechanized sublevel mining methods will be used to mine the ore on a two shift per day, 365 days per year schedule from two portal locations, the South Underground Mine and the North Underground Mine. The South Mine underground development will begin in 2006 and the North Underground Mine in 2008. Each of the mines will reach a production rate of approximately 1,750 tonnes per day of ore during its life. The average dilution in the final underground model is 24%. The production schedule for the Underground Production is:

Underground Production	Tonnes	Gold Grade	Silver Grade	Tonnes per Day
2006				
2007	71,879 t	23.3 g /t	290 g/t	296 t /d
2008	234,821 t	24.6 g /t	312 g/t	643 t /d
2009	648,173 t	19.3 g /t	190 g/t	1,775 t /d
2010	902,172 t	16.7 g /t	235 g/t	2,470 t /d
2011	1,043,364 t	16.7 g /t	223 g/t	2,850 t /d
2012	1,004,288 t	15.7 g /t	219 g/t	2,750 t /d
2013	1,043,003 t	12.6 g /t	213 g/t	2,850 t /d
2014	713,504 t	13.2 g /t	193 g/t	1,950 t /d
2015				
TOTAL	5,661,203 t	16.0 g/t	219 g/t	

Ore Processing

The Mill will process the ore on a two shift per day, 365 days per year schedule at a rate of approximately 3,000 tonnes per day of ore during the operation of the open pit, South Underground Mine and then the North Underground Mine.. Low grade ore from the pit is stockpiled through 2010 then milled in 2011 and beyond to keep the mill at its design capacity as long as possible. The milling rate is 1,095,000 tonnes per year. The gold recovery will average 93.8% and the silver recovery will average 78.8%. The mill availability will be 94%. The milling process will consist of a primary crushing and grinding circuit and will include conventional gravity technology followed by whole ore leaching. Merrill Crowe precipitation will be used to produce doré bars. Doré will be sent to a refinery located near Magadan or shipped by air to a refinery in central Russia. The production schedule for the mill follows:

Mill Production	Tonnes per Day	Tonnes per Year	Gold Grade	Silver Grade	Gold Produced [troy ounces]	Silver Produced [troy ounces]	Gold Recovery	Silver Recovery
2008	To 3,000 t/d	536,850 t	25.0 g/t	255 g/t	376,394 t-oz	3,284,551 t-oz	92.7%	79.2%
2009	3,000 t/d	1,095,000 t	24.2 g/t	238 g/t	767,814 t-oz	6,444,629 t-oz	92.9%	79.1%
2010	3,000 t/d	1,095,000 t	18.4 g/t	241 g/t	619,349 t-oz	6,717,017 t-oz	93.9%	79.2%
2011	3,000 t/d	1,095,000 t	16.5 g/t	219 g/t	551,520 t-oz	6,143,642 t-oz	94.1%	79.3%
2012	3,000 t/d	1,095,000 t	15.1 g/t	208 g/t	504,021 t-oz	5,794,562 t-oz	94.3%	78.9%
2013	3,000 t/d	1,095,000 t	12.3 g/t	206 g/t	417,564 t-oz	5,660,065 t-oz	95.0%	78.0%
2014	2,,943 t/d	1,095,000 t	10.9 g/t	151 g/t	379,371 t-oz	4,369,449 t-oz	94.4%	77.3%
2015								
TOTAL		**7,086,898 t**	**16.9 g/t**	**214 g/t**	**3,616,033 t-oz**	**38,413,915 t-oz**	**93.8%**	**78.8%**

A cyanide recovery system followed by cyanide destruction system will be used to reduce cyanide concentrations to an acceptable level for disposal. These tails will be pumped to a conventional tailings impoundment.

Logistics

The primary access road to the Kupol site is the proven winter road from Pevek to Kupol. A summer road system may be used to transport freight to a staging area 125 km north of Kupol. Airport facilities presently under construction are located approximately 10 km north of the Kupol site will be used for personnel transport, light freight movement, dore´ shipments and fresh food shipments.

Execution Plan

The Kupol Project execution plan encompasses project management by Bema Gold Corporation and utilizes a delivery method comprised of a combination of Engineering, Procurement, and Construction (EPC) combined with multiple prime engineering contracts and some self performed owner construction.

Engineering and Procurement will be managed from the Bema corporate offices in Vancouver, British Columbia, Canada with construction management occurring from the Kupol Site. The majority of equipment and supplies are sourced from North American or European suppliers and is ocean shipped to the North Siberian Seaport of Pevek with subsequent overland delivery utilizing dedicated winter roads constructed and maintained by the project.

The site development will take place year round, utilizing a work force of experienced Russian nationals, trained and supervised by Russian and expatriate supervision, many of whom have worked on Bema’s previous Russian projects. Personnel will be “fly in-fly out” on a standard rotational basis.

Logistics will be supported from the large existing Bema support structure in place and operating in Russia and North America.

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Appendices

Appendix A Certificates of Qualified Persons
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1.0 INTRODUCTION AND TERMS OF REFERENCE (Item 4)

Gustavson Associates, LLC ("Gustavson") was commissioned by Bema Gold Corporation ("Bema") in April, 2005 to assist Bema with the preparation of a Canadian National Instrument 43-101 compliant summary of the Kupol Project Feasibility Study, Chukotka A.O., Russian Federation.

The purpose of this Technical Report is to provide a concise summary of the Kupol Project Feasibility Study.

1.1 Terms of Reference

A significant information base is in the public domain concerning the Kupol Project. In November, 2003 a Technical Report was authored by Tom Garagan, P. Geo.(Bema's Vice President, Exploration) and Hugh MacKinnon, P. Geo. (Kupol Project Geologist), titled "Kupol Project, Chukotka A.O., Russian Federation". A second Technical Report, titled "Kupol Project Preliminary Assessment Summary, Chukotka A.O., Russian Federation" was authored by Tom Garagan, P. Geo. in June, 2004. Most recently, another Technical Report was written by Tom Garagan, P. Geo. titled "Technical Report on the Kupol Project, Chukotka A.O., Russian Federation, Report for NI 43-101" on 31 March, 2005. The most recent report comprises a complete NI 43-101 compliant report on the geology, exploration and mineral resource estimates of the Kupol Project. Each of the above reports is filed on SEDAR.

With respect to Items 6 through 11 of Form 43-101F1, the Garagan report of 31 March 2005 presented the current knowledge of each Item. These Items will be cited herein with a note as to the location in Garagan's report, with any additional, material information added. If warranted, summary tables are presented.

Bema is a "producing issuer" with respect to mineral resource and mineral reserve reporting to Canadian securities authorities. There is no requirement for the independence of the Qualified Person in reporting. This technical report has been prepared in accordance with the guidelines provided in National Instrument 43-101 ("NI43-101"), Standards of Disclosure for Mineral Projects and Form 43-101F1. Where possible, the applicable 43-101F1 paragraph number in this report's section headings, i.e. **(Item 29)**.

William Crowl, Gustavson's Vice President, Mining Sector and Mr. Fred Stahlbush, Bema's Kupol Feasibility Study Manager are ultimately responsible for the preparation of this report. They are Qualified Persons as defined in NI43-101. Certificates by Messrs. Crowl and Stahlbush are included in Appendix A. Mr. Crowl visited the Kupol Project site and critical project infrastructure sites from May 15 through May 18, 2005. Fred Stahlbush visited the site twice in 2004.

The Kupol Preliminary Economic Assessment and the Feasibility Study involved a multitude of professionals in the Bema organization and several individual and consulting firms performing engineering and providing third party reviews. All of the firms and individuals have the requisite expertise and experience to provide Bema with engineering products meeting or exceeding industry standards. Table 1.1 lists the major contributors to the Kupol Project.

Table 1.1: Kupol Feasibility Study Responsible Parties, Contractors and Consultants

Major Feasibility Area	Sub Area	In-House Expertise	Consultants/ Third Party Reviewers	Comments
Business Issues/Systems	Country Taxation	Jim Sullivan and Magadan Staff Mark Corra and Staff		Bema Russian Experience
	Local Taxation	Jim Sullivan and Magadan Staff Mark Corra and Staff		“
	Dore Marketing	Jim Sullivan and Magadan Staff Mark Corra and Staff		“
	Legal Considerations	Roger Richer Bill Lytle Jim Sullivan		“
Organization	HR	Jim Sullivan and Russia Staff		Bema Russian Experience
	Territorial/Region Laws	Jim Sullivan and Staff Bill Lytle		Bema Russian Experience
	Government relations	Magadan Staff Bill Lytle Bema Executives Jim Sullivan		
	Safety and Health Issues	Julietta Operations personnel Jim Sullivan and Staff Bill Lytle		Operations Experience
Surface Mine	Material Handling Characteristics	John Rajala George Johnson Doug Wollant Julietta Operators	Diamondback Consulting	Diamondback formerly Jeneke and Johanson
	Waste/Ore Characteristics	Tom Garagan/Large Experienced Geologic Staff John Rajala		Bema has strong exploration track record

Major Feasibility Area	Sub Area	In-House Expertise	Consultants/ Third Party Reviewers	Comments
	Waste Rock Issues (ARD, acid rock drainage etc.) and Geochemical Characterization	Bill Lytle Fred Stahlbush	Geochemica - Mark Logsdon Steve Atkins AMEC – Peter Lighthall	
	Geotechnical – Slope Analysis and Stability	Fred Stahlbush Hugh McKinnon Vernon Shein Travis Naugle	Wardrop Mining and Minerals – Dave West SRK Consulting	SRK consultant during the PEA
	Surface Mine Production Engineering	Doug Wollant Fred Stahlbush George Johnson	Wardrop – Andy Nichols, Ian Pritchard, and Gordon Zurowski SRK Jim Robertson	
	Equipment, Maintenance Facilities	Julietta Personnel Richard Matson and Kupol Site personnel Doug Wollant Fred Stahlbush Rick Thomas George Johnson Julietta Personnel Orocon Personnel	Wardrop – Andy Nichols and Gordon Zurowski Mike Ross - Orocon	Mike Ross and Richard Matson (with staffs) and Julietta Personnel bring years of experience to this area
Underground Mine	Mine Openings/Mine Preproduction development	Fred Stahlbush Doug Wollant Travis Naugle Don Cameron Julietta Staff – Randy Reichert and Pat Dougherty	Wardrop – Jim Campbell, Dave West and Andy Nichols	Bema has considerable underground excavation expertise
	Underground Mining methods/Stoping Cycles and Backfill Knowledge	Same as above	SRK Wardrop – same as above	Bema has considerable underground excavation expertise
	Geotechnical	George Johnson Jim Sullivan Fred Stahlbush Travis Naugle Hugh McKinnon Vernon Shein	Wardrop Mining and Minerals – Dave West SRK Consulting	SRK consultant during the PEA
	Underground Equipment Capital Cost Experience Operating cost estimation Review of Proposals	Fred Stahlbush Travis Naugle Doug Wollant	Wardrop – Ian Pritchard and Andy Nichols Thyssen	Underground development and mining straightforward – Conventional Rubber-tired development and short level spacing Longhole stoping
	Underground Infrastructure	Fred Stahlbush Travis Naugle George Johnson Doug Wollant	Wardrop - Andy Nichols, Jim Campbell, Martin Drennen Thyssen	

Major Feasibility Area	Sub Area	In-House Expertise	Consultants/ Third Party Reviewers	Comments
Metallurgy/Mill	Metallurgical Characterization	John Rajala Geology Staff at Kupol and Vancouver – Tom Garagan, Hugh Mackinnon, Vern Shein Art Winckers Mike Ross	Orocon Tony Brown & Doug Halbe – reviews Fred Pena of Metso - Grinding Multiple labs used: Lakefield Research McClelland labs McPherson Limited Dorr-Oliver Eimco Canmet (leach optimization) Inco (CN destruction)	Au and Ag recoveries CN use Optimization CN recoveries testwork Thickening testwork Flow Sheet – Whole Ore Leach
	Flow Sheet Development	See Above	See Above Chris Fleming Unifield Engineering - AVR	See Above
	Equipment Selection	John Rajala Mike Ross Art Winckers Julietta Personnel	Orocon – Mike Ross, Art Winkers and several staff	John Rajala and Orocon personnel have construction multiple successful metallurgical installations
	Operations	Mike Ross Art Winckers John Rajala	Orocon Tony Brown Doug Halbe	
	Tailing Disposal	John Rajala Mike Ross Fred Stahlbush Bill Lytle	AMEC – Peter Lighthall	Mike Ross – Looked at conventional tailing impoundment and dry-stacked tailings – conventional chosen
	Water Supply	Bill Lytle	Russian Hydrologists familiar with Arctic Environment AMEC	
Geology	Very Experienced Staff	Tom Garagan Brian Scott Hugh MacKinnon Vern Shein Multiple site geologists Susan Meister Don Cameron	Harry Parker David Rhys Greg Warren Ken Brisebois Bill Crowl	Proven Track record
	Structural Geology	Above	David Rhys	
	Resource Interpretation and Reserve Calculation	Above	AMEC - Harry Parker and team	
Services/Infrastructure /Movement of Personnel and materials	Logistics	Doug Wollant Magadan Staff of Buyers and expediters Steve Olsen	Multiple contracting companies	5 years of Julietta successful construction and operations Kupol – two successful winters getting supplies to the property
	Maintenance facilities	Doug Wollant Richard Matson	Orocon Wardrop	Experience in services available
	Material transportation Costs and Material Supply Costs	Jim Sullivan Magadan Staff\Doug Wollant		Constant ongoing supply stream to Exploration effort and Julietta Major materials to Kupol winters of 2003-04 and 2004-05
	Power Generation	Richard Matson Doug Wollant Julietta	Orocon and consultants	

Major Feasibility Area	Sub Area	In-House Expertise	Consultants/ Third Party Reviewers	Comments
	Water Supply	Bill Lytle	AMEC – Andzej Slawinski	Process water source established Permanent Potable water Source exploration underway
	Site - Geotechnical	Hugh Mackinnon Vernon Shein Doug Wollant	AMEC – Rodney Kostaschuk Peter Lighthall	Considerable Russian Expertise used Also performed aggregate testing
Environmental	Socio-Economic Issues	Bill Lytle Jim Sullivan and Magadan Staff		“Daily” experience
	Flora and Fauna Issues	Bill Lytle	Various Russian Consultants	Operating Experience
	Permitting Regulations and timelines	Jim Sullivan and Magadan Staff Bill Lytle		Multiple project permits in place
General	Construction Experience Equipment Procurement Expediting	Randy Reichert – Polaris Doug Wollant – Julietta George Johnson – Multiple mines John Rajala-Kubaka Fred Stahlbush – Multiple mines with major construction projects	Orocon – Multiple Mill facilities construction Richard Matson – Multiple Road and Civil Construction projects (mining and Petrochemical) George Stone – Catering and Infrastructure support	The exploration man camp at Kupol housed 150+ people for the entire summer during the drilling and construction effort. Roadway, aggregate crushing and mill site excavation currently underway at Kupol Summer 2005 camp of 400+personnel
	Transportation	Doug Wollant Jim Sullivan and Magadan Staff	Orocon Equipment suppliers Komatsu (on site at Kupol now) Tamrock General – Investment activity in Russia has increase in the last 10 years. Infrastructure and logistics are known by the mining industry and the petrochemical industry	Bema Presently: Ships by sea to Pevek and Magadan Ships by rail across Russia Ships by fixed wing propeller aircraft to Keperveem Ships by Chopper wherever required Supplies by full season road to Julietta Supplies by Winter road to Kupol from Pevek and Magadan Charter flights from Nome to Keperveyem
	Cost Estimating	Data from Julietta Kupol Experience Fred Stahlbush Rick Thomas Tony Ketley Doug Wollant George Johnson In general – very experienced staff	Orocon – Experience Independents: James Rawley Anthony Brown Wardrop – Andy Nichols Thyssen Mining Tony Ketley	2004 actual costs 2005 – Experience from actual site activities (civil work) Logistics – Actual cost and recognition of improvements Final contracts signed and fabrication underway (mill services building and permanent camp for example
	Scheduling	Doug Wollant John Rajala Rick Thomas	Wardrop – Andy Nichols Orocon	Significant Site and Logistics events have already taken place

Major Feasibility Area	Sub Area	In-House Expertise	Consultants/ Third Party Reviewers	Comments
		Goran Serdarevic	AMEC (tailing impoundment)	

1.2 Effective Date (Item 24)

The effective date of the mineral resource statements in this report is March 31, 2005, as reported in Tom Garagan's "Technical Report on the Kupol Project, Chukotka A.O., Russian Federation, Report for NI 43-101", dated March 31, 2005. The effective date of this Technical Report is July 4, 2005.

1.3 Metal Prices and Exchange Rates

The currency exchange rates used in the Kupol Feasibility Study are 30 Russian Rubles per US$ and $1.20 Canadian dollars per US$. The gold and silver prices used are US$400 and US$6.00 per troy ounce of gold and silver, respectively.

1.4 Units of Measure

Unless stated otherwise, all quantities are in metric units.

1.5 Qualifications of Consultant

Portions of this report have been prepared based on technical reviews and first-hand examinations/investigations by William Crowl, P.G. from Gustavson Associates, LLC's Boulder, Colorado, USA office.

Neither Gustavson nor any of its employees and associates employed in the preparation of this report has any beneficial interest in Bema Gold Corporation. Gustavson will be paid a fee for this work in accordance with normal professional consulting practice.

William Crowl, P.G. has extensive experience in the mining industry and is a member in good standing of appropriate professional organizations and is a Qualified Person as defined by NI43-101.

2.0 DISCLAIMER (Item 5)

2.1 Limitations & Reliance on Information

Data presented in this report reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time.

This report summarizes the results of the Kupol Project Feasibility Study, which is the result of the compilation of the work of a number of individuals and companies by Bema Gold Corporation.

The achievability of LoM plans, budgets and forecasts are inherently uncertain. Consequently, actual results may be significantly more or less favorable.

This report includes technical information, which requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Gustavson does not consider them to be material.

Gustavson is not an insider, associate or an affiliate of Bema. The results of the study by Gustavson are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

Gustavson reviewed a limited amount of correspondence, pertinent maps and agreements to assess the validity and ownership of the mining concessions. Bema assumes full responsibility for statements on mineral title and ownership.

No information came to Gustavson's attention during their review of the data and information provided by Bema that would cause Gustavson to doubt the integrity of such data and information.

This report was prepared in cooperation with senior Bema personnel, who are persons well experienced in their respective fields. Gustavson and William Crowl, P.G. take responsibility specifically for the reporting of the Mineral Reserves as converted from the Mineral Resources estimated under the supervision of Qualified Persons employed by Bema, namely Tom Garagan, P. Geo. Fred Stahlbush, as Bema's Kupol Project

Feasibility Study Manager, is the Qualified Person responsible for the remaining sections of this report.

2.2 Disclaimers & Cautionary Statements for US Investors

In considering the following statements Gustavson notes that the term "ore reserve" for all practical purposes is synonymous with the term "Mineral Reserve".

The United States Securities and Exchange Commission (the "SEC") permits mining companies, in their filings with the SEC, to disclose only those mineral deposits that a company can economically and legally extract or produce from. Certain items are used in this report, such as "resources," that the SEC guidelines strictly prohibit companies from including in filings with the SEC.

Ore reserve estimates are based on many factors, including, in this case, data with respect to drilling and sampling. Ore reserves are determined from estimates of future production costs, future capital expenditures, and future product prices. The reserve estimates contained in this report should not be interpreted as assurances of the economic life of the Mining Assets or the future profitability of operations. Because ore reserves are only estimates based on the factors described herein, in the future these ore reserve estimates may need to be revised. For example, if production costs decrease or product prices increase, a portion of the resources may become economical to recover, and would result in higher estimated reserves. The converse is also true.

The LoM Plans and the technical economic projections include forward-looking statements that are not historical facts. These forward-looking statements are estimates and involve a number of risks and uncertainties that could cause actual results to differ materially.

Gustavson has been informed by Bema that to the best of its knowledge, there is no current litigation that may be material to the Kupol Project Assets.

3.0 PROPERTY DESCRIPTION AND LOCATION (Item 6)

Refer to Section 4 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR as per provisions of 43-101F1 for details of the Property Description and Location. Figure 3.1 is a general location map for the Kupol Project and Figure

3.2 is a map of the local area with roads, ports, etc. Figure 3.3 is included to show the property boundaries and the coordinates of the property corners.

Figure 3.1: Kupol Property Location Map

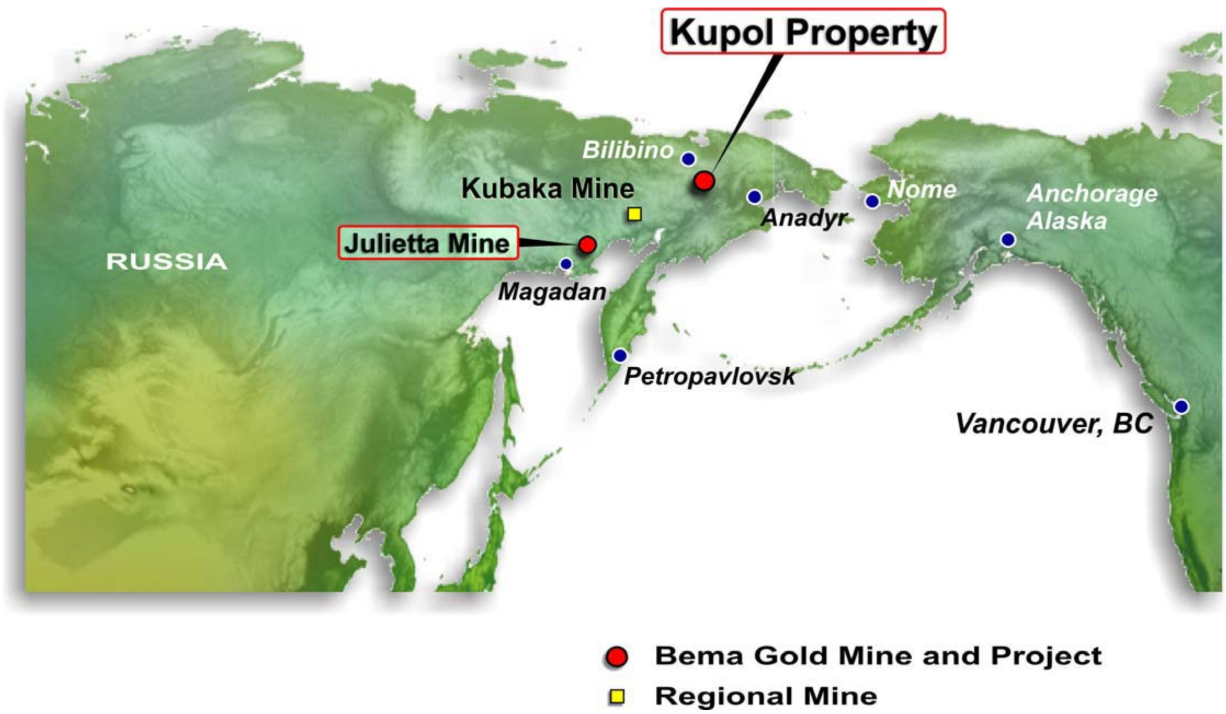


Figure 3.2: Local Area and Infrastructure

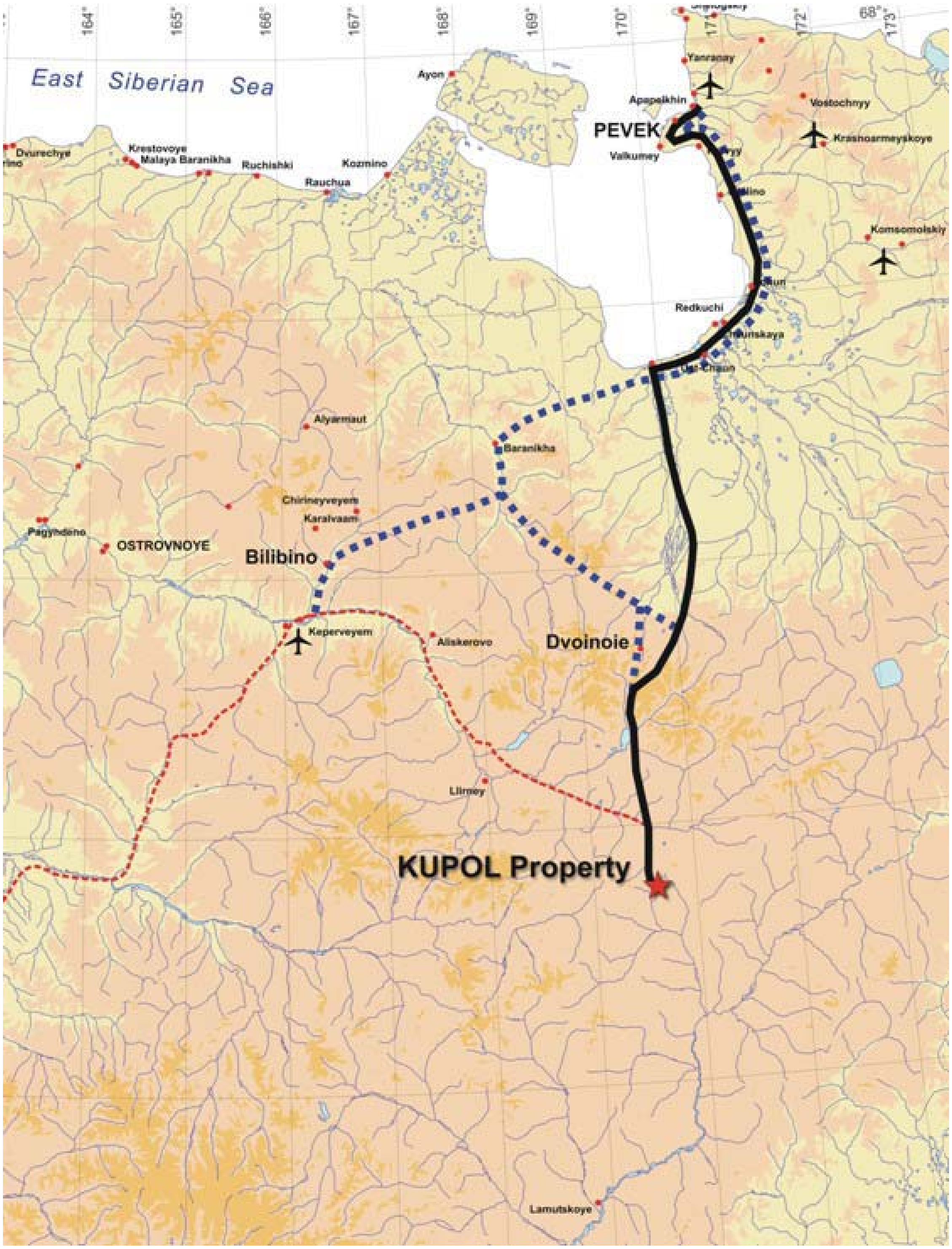
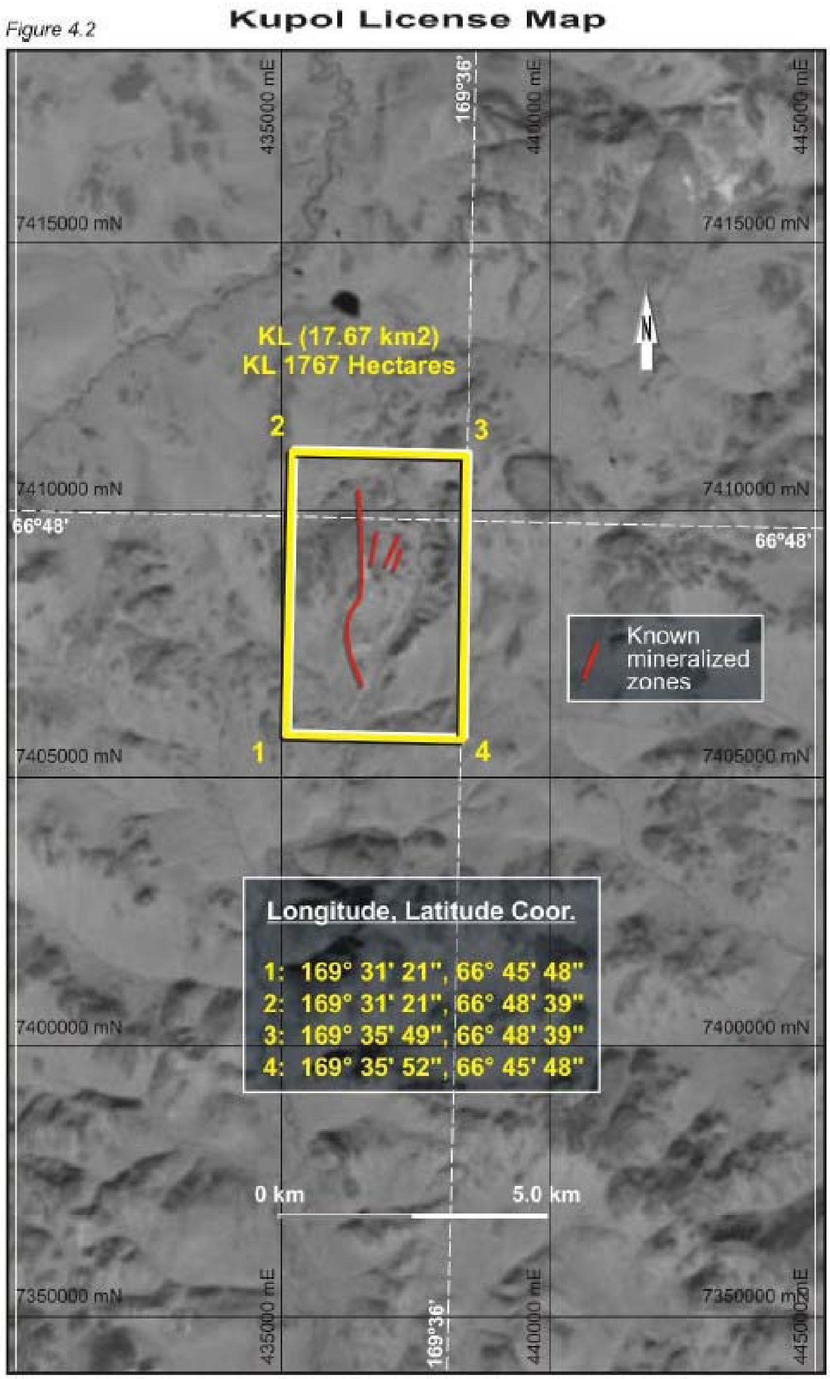


Figure 3.3: Kupol License Area



4.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY (Item 7)

Refer to Section 5 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR as per provisions of 43-101F1 for details of the Property Description and Location. Clarifying additional information on Accessibility and Infrastructure, not provided in the above referenced report, is included below.

4.1 Accessibility & Infrastructure

Refer to Section 20.8 of this Report for information on access and infrastructure.

5.0 HISTORY (Item 8)

Refer to Section 6 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR as per provisions of 43-101F1 for details of the Kupol Project history.

6.0 GEOLOGICAL SETTING (Item 9)

Refer to Section 7 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR as per provisions of 43-101F1 for details of the Kupol Deposit geology and geological setting.

7.0 DEPOSIT TYPES (Item 10)

Refer to Section 8 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR as per provisions of 43-101F1 for details of the Kupol gold deposit types.

8.0 MINERALIZATION (Item 11)

Refer to Section 9 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR as per provisions of 43-101F1 for details of the Kupol deposit mineralization.

9.0 EXPLORATION (Item 12)

9.1 Exploration History

In summary, the vein system was defined by thirty-five trenches over a strike length of three kilometers and by geophysics, geochemistry, and mapping over a strike length of four kilometers. Trench spacing ranged from 200 meters along strike to the south to fifty meters in the Big Bend. The central portion of the vein system was stripped, mapped and channel sampled in detail. A soil geochemical survey that covered 7.8 square kilometers defined the deposit as a gold, silver, arsenic anomaly with localized areas of anomalous mercury, lead, zinc, and antimony. Magnetic and resistivity surveys were completed over a similar area with initial 100 by 20-metre grids followed by detailed 25-metre by 5-metre and 20-metre by 5-metre grids, respectively. This work defined the deposit as an area of magnetic low response and higher apparent resistivity. Twenty-six drillholes totaling 3,004 meters were drilled over a strike length of 450 meters to a maximum depth of 140 meters. In 2003, six more trenches were excavated, and 166 drillholes, for 22,257.69 meters were drilled over a strike length of 3.1 kilometers to a maximum depth of 250 meters. Table 9.1 summarizes the drilling, trenching and channel sampling. Garagan and MacKinnon (2003) documents the 2003 exploration program.

Table 9.1: Summary of Work Prior to 2004

1998		
Type of Work	*Count*	*Meterage*
Drilling	2	160.00
Trenching	4	700.00
1999		
Type of Work	*Count*	*Meterage*
Stripping and Channel Sampling	12	416.50

Drilling	7	741.40
Trenching	1	120.00
2000		
Type of Work	*Count*	*Meterage*
Stripping and Channel Sampling	80	2099.30
Drilling	12	1509.30
Trenching	13	2618.60
2001		
Type of Work	*Count*	*Meterage*
Stripping and Channel Sampling	17	595.00
Drilling	5	593.30
Trenching	16	1595.50
2003		
Type of Work	*Count*	*Meterage*
Drilling	166	22257.69
Trenching	6	805.22

9.1.1 2004 Exploration

In 2004, the field season spanned from May through November. The work consisted of drilling, trenching, and the stripping and channel sampling of the Kupol vein. Table 9.2 summarizes the work completed in 2004.

Table 9.2: Summary of Work in 2004

Type of Work	Number	Meterage
Stripping and Channel Sampling	87	698.89
Drilling	309	52,828.50
Trenching	2	225.53

Additional sampling for metallurgical testing was completed.

9.1.1.1 Trenching

Two trenches, for 225.53 meters were excavated in the South Extension zone. These trenches were cleaned and mapped; seventy-one samples were collected. Coarse-grained stibnite, the first occurrence of larger crystals documented in the deposit area, was observed in trench K-53. The results of significance are presented in Table 93.

Table 9.3: Trench Results

Trench ID	From	To	Width	Au gpt	Ag gpt	Ag:Au
K-53	26.10	28.60	2.50	2.53	59.53	235:1
	35.40	39.00	3.60	1.03	40.12	39.0:1
K-54	58.30	59.00	0.70	2.80	18.00	6.4:1

9.1.1.2 Stripping and Channel Sampling

In 2004, approximately 4,680 square meters of the Kupol vein mineralization was exposed, mapped, and sampled in sections of the North, Big Bend, and Central zones. Prior to 2003, large areas of the Big Bend and Central zones were stripped, mapped, and sampled in the same manner. The purpose of this work was to aid in calculating dilution and assessing grade continuity in conjunction with the very close-spaced drilling in the Big Bend and South zones.

The areas were mechanically cleared of surface debris and were pressure washed using a Wajax pump. A five meter by five meter control grid was established by Russian surveyors over each area. The areas were mapped by Russian geologists at a scale of 1:50. The exposures were channel sampled along east-west lines at five to ten meters spacing. The start and end of each sample was surveyed.

A summary of the extents of this work is presented in Table 9.4.

Table 9.4: Summary of Stripping, Mapping and Channel Sampling

Zone	Area (m^2)	Spacing (m)	Strike Length (m)	No of Lines	No. of Channels	No. of Samples
North	1415	10	60	7	8	195
Big Bend	2080	5 and 10	270	38	49	448
South	1185	5 and 10	100	19	30	328
Total	4680		430	64	87	971

In the North zone, due to freezing temperatures only the southern most twenty percent of the area was washed and sampled. This extended up to fifteen meters east of the main vein system across a zone of sheeted and stockwork quartz veining.

10.0 DRILLING (Item 13)

The primary emphasis of the 2004 drilling program was to upgrade the inferred resource to indicated. The drilling program was successful in confirming and improving upon known mineralization. In the North Zone multiple veins were confirmed, new veins were identified and the deeper mineralization at the north end was extended 350 meters to the north. In the Big Bend Zone, drilling continued to prove the continuity of high-grade mineralization. Mineralization remains open to the south and at depth and along strike in the north. Several parallel structures are untested.

In 2004, 309 drillholes were drilled for a total of 52,828.5 meters. The drillhole locations are shown on Figure 10.1. Figure 10.2 is a composite longitudinal section of the Kupol drilling and mineralization.

The diamond drilling was conducted using two Longyear 38 drill rigs, three Longyear 44 drill rigs and two Russian CKB-4 drill rigs. The Longyear rigs drilled PQ, HQ and NQ diameter core; the Russian rigs drilled NQ diameter core.

Core recovery varies by location. Recoveries in the mineralized zones range from 3% to 100%; the average is 96.3%. Drilling muds and polymers were used extensively to enhance recoveries.

The property grid is a Russian local grid system; it replaces the Gauss Kruger (Pulkovo 42) datum used in 2003. Grid lines are oriented east-west, perpendicular to the average strike of the deposit.

The Kupol deposit is divided into six zones:

1. South Extension Zone: 89525N to 90075N
2. South Zone: 90075N to 90700N
3. Big Bend Zone: 90700N to 91275N
4. Central Zone: 91275N to 92100N
5. North Zone: 92100N to 92425N
6. North Extension Zone: 92425N to 93150N

The zones are contiguous and mineralization has been defined within the zones over 3.6 kilometers of strike. The geology and the results are summarized by zone in the following sections.

Figure 10.1: Drillhole Location Plan

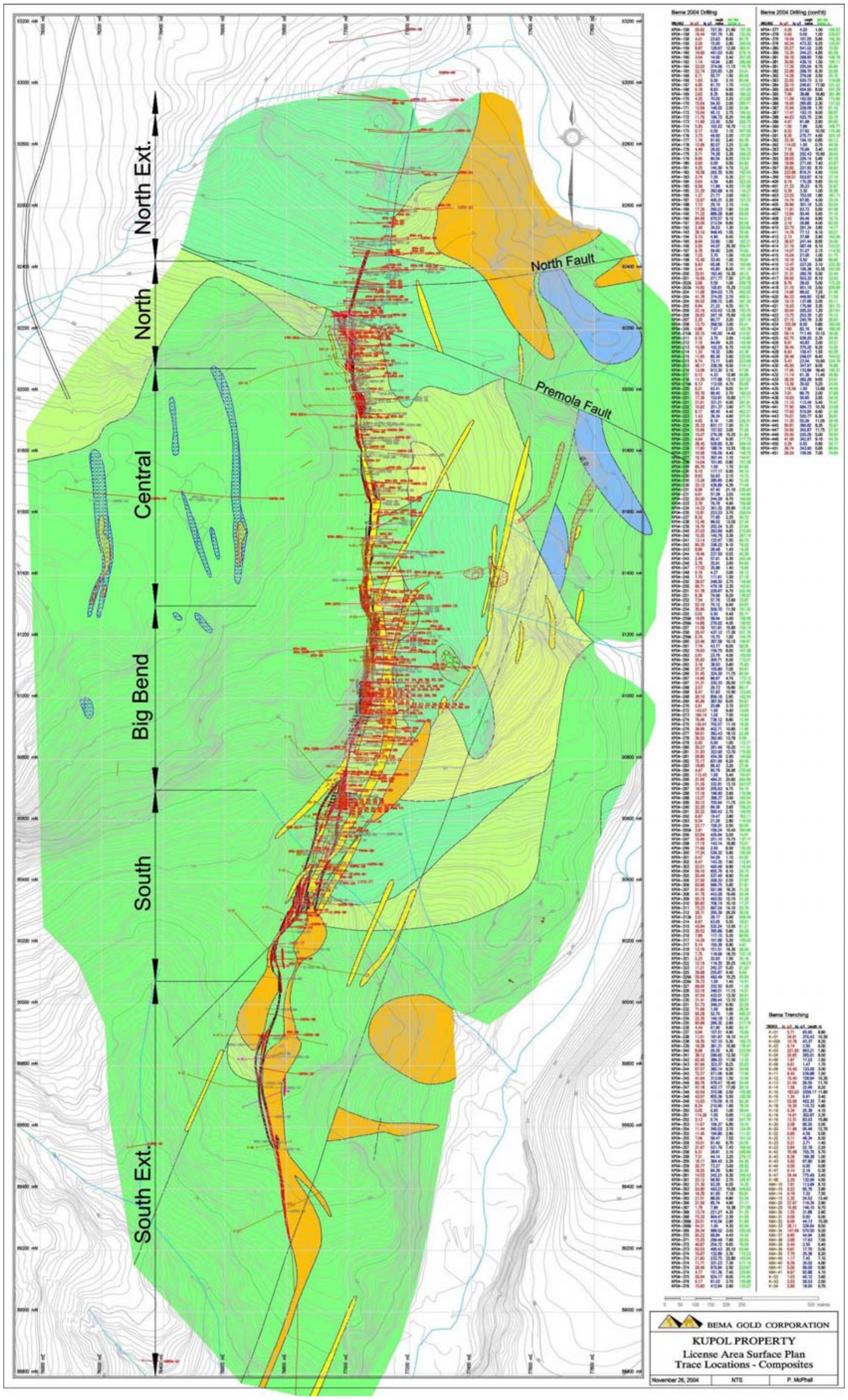
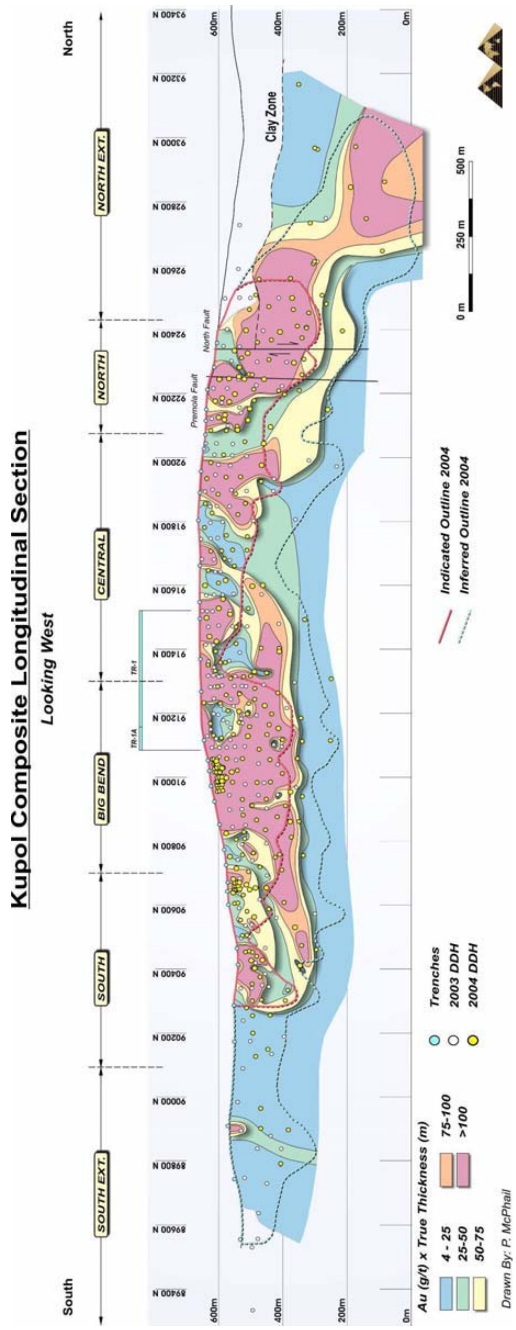


Figure 10.2: Composite Longitudinal Section



10.1 South Extension Zone: 89525N to 90075N

The South Extension zone extends from the Kaiemraveem River at section 89525N to 90075N. In 2004, six drillholes for 1,038.3 meters were drilled. The drill spacing ranged from 75 meters to greater than 200 meters along strike. The zone has been drilled to a maximum vertical depth of 200 meters. The drilling focused on testing down-dip and strike extensions to several higher-grade zones that were intersected in 2003.

Drilling, trenching, and mapping has defined three main veins with variable widths and grades that are spaced over a fifteen to thirty meter wide north trending corridor that is bounded to the east and in part to the west by several rhyolite dykes and flow dome complexes up to 150 meters wide. Significant grades have been intersected in all three veins; however, the ore shoots are not yet well defined. The vein is disrupted by the rhyolite units.

The South Extension zone is characterized by quartz vein material with distinct olive-green quartz and a hematite overprint. Veins are commonly brecciated, not as well banded and the quartz tends to be more sucrosic and/or massive than to the north. The sulphosalts are a bit finer-grained than in the Big Bend zone and, in general, there is more gypsum than in the vein systems observed to the north. There is no obvious zonation or strong base metal signature to suggest that the South Extension zone is distal to the main mineralization pathways. Silver to gold ratios vary from 7:1 to 30:1 with an average of approximately 12:1, similar to the rest of the Kupol deposit. A moderate to strong acid sulphate alteration zone is defined over the zone from the Kaiemraveem River, in the south, to the South zone.

10.2 South Zone: 90075N to 90700N

Fifty-six holes, totaling 7,197.8 meters were drilled in the South zone in 2004. The drill spacing ranged from 50 meters to greater than 100 meters along strike for most of the zone. Sixteen holes were drilled on four sections between 90640 and 90670N, spaced ten meters apart and to a vertical depth of 30 meters vertical depth. This work, in conjunction with the detailed channel sampling of the exposed vein, helped to assess grade continuity, vein contact geometry, and dilution.

The South zone contains up to five significant veins that occur in a forty to seventy meter wide north-northeast trending corridor. The veins are locally disrupted by a series of two to four rhyolite dykes up to 100 meters wide at surface with a similar north-northeast trend. A large rhyolite dyke/flow dome complex cuts off the veins at 90100 N; this marks the southern termination of the South zone. The north end of the zone is defined by a northwest trending fault and the start of the southward bifurcation of the main vein system.

Significant intersections from these veins include:

- KP04-224 - average 35.33 g/t Au and 831.17 g/t Ag over 7.0 meters (4.09m true width)
- KP04-253 - average 52.16 g/t Au with 70.12 g/t Ag over 6.9 meters (3.78m true width)

A well-mineralized vein that occurs sixty meters east of the main vein system is inferred to be the faulted southern extension of the Big Bend vein, and has been referred to as the "Offset Vein." This vein was intersected in only two holes; its extent is unknown.

Significant intersections from this vein include:

- KP04-338 – average 11.01 g/t Au and 181.67 g/t Ag over 16.1 meters (9.23 m true width)
- KP04-451 - average 29.54 g/t Au with 156.56 g/t Ag over 7.0 meters (3.86 m true width)

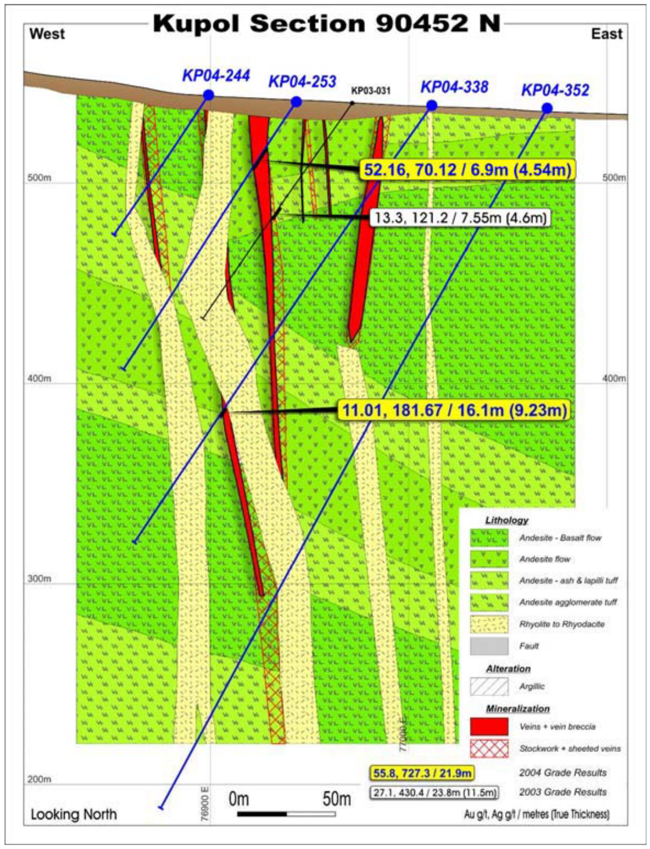
As with the South Extension zone, the South zone is characterized by an abundance of olive green quartz and two phases of hematite. The Offset Vein lacks the hematite rich phases, is less brecciated, and has better developed crustiform banding than the rest of the South Extension zone veins. This supports the inference that the vein is the strike continuation of the main (eastern) South - Big Bend vein.

Preliminary petrographic and infrared spectroscopy (PIMA) studies indicate a mixed alteration assemblage similar to other areas of the deposit. The presence of kaolinite and boiling textures in the vein and at depth in holes KP03-46 and 42 suggest a relatively high stratigraphic position in the epithermal system and good potential for mineralization at depth. Petrographic work to date indicates that acanthite is the dominant silver sulphide mineral.

A northwest trending sinistral fault cuts through the north end of the zone and offsets the main vein. Dyke and vein discontinuities that became evident during the modeling process indicate that additional faults likely disrupt the zone. Due to drilling density and orientation, the fault geometry is uncertain. Additional drilling, including oblique drillholes, is required to better define the vein and fault geometry. Vein fragments in the rhyolitic polymitic breccia, coupled with discontinuities in veins across the dykes suggest that some assimilation of the vein by the dyke has occurred.

South Zone cross section 90452N is presented as Figure 10.3.

Figure 10.3: South Zone Section 90452N



10.3 Big Bend Zone: 90700N to 91275N

The Big Bend zone has a strike length of 575 meters. The north end of the Big Bend zone is defined by the location of a large rhyolite dyke that bisects the zone at Section 91275 N. Southwest trending splays off the main vein structure at a sinistral fault at Section 90700 N defines the south end of the zone.

The Big Bend zone has been tested by 120 drill holes totaling 16,071.9 meters, on sections spaced 25 to 50 meters apart to a depth of 250 meters below surface (400m elev.) and on sections spaced 50 to greater than 100 meter apart to a vertical depth of 400 meters. In the interval between 90930 N and 91050 N, fifty holes were drilled on sections ten meters apart to up to thirty meters below surface. The close-spaced drilling, in conjunction with the detailed channel sampling of the exposed vein, was to assess grade continuity and the nature of the contacts for dilution studies.

The zone is localized along a change in strike of the Kupol structure from 000 to 020 that was previously interpreted as a right-lateral dilatant flexure in the vein system. Various field relationships now suggest low magnitude (tens of meters) sinistral displacement along a pre-existing bend in the Kupol structure during formation of the vein system.

The Big Bend zone is comprised of a single, large banded fissure vein with associated sheeted veining. This vein is divided into footwall and hanging wall segments by a twenty to forty meter wide rhyolite dyke that bisects the zone between 400 to 550 meters elevation. This dyke branches upwards into two to four smaller dykes that are mostly situated in the hanging wall of the vein system. There is no apparent difference in the grade of intersections on either side of the dykes. The width of the Big Bend vein varies from one to twenty-two meters and is associated with a lower-grade stockwork/sheeted vein that is up to 30 meters wide. Clay gouge, sheeted and stockwork veins, and small islands of wall rock occur locally within the main vein envelope.

Gold and silver mineralization exhibits remarkable continuity within individual vein intercepts and between sections. The surface exposure of the vein indicates very strong development of continuous sulphosalt rich banding that helps explain the grade continuity. The new surface exposure of the vein indicates less brecciation of the colloform-crustiform banding than previously thought. Either the polyphase brecciation is a feature of deeper levels of the system or the brecciation seen in core is partially a cockade texture or other irregular banding features of the veins.

Gold and silver grades decrease significantly at 250 to 300 meters below surface over the length of the zone. This reduction in grade at 300 to 350 meters elevation is inferred to represent a precious metal deposition horizon. The controls on the horizon are not yet understood. Textures such as cyclic banding, open space filling, hydrothermal brecciation, cryptocrystalline quartz and partially replaced (by chlorite and pyrite) sulphosalt banding are present below this level, and to the level of the deepest drilling, which indicates that this area is still within the boiling level of the hydrothermal system. It is uncertain if there are stacked precious metal horizons in this area; this will be tested by drilling in 2005.

Overall, the styles of mineralization are similar throughout the Big Bend zone, but within each vein intersection there are varieties of textural and mineralogical types that reflect local variability in hydrothermal brecciation/boiling events. The following five phases have been documented in surface exposures (Rhys, 2004):

- Stage 1: early colloform-crustiform quartz-adularia phase containing sulphosalt bands
- Stage 2: quartz-sulphosalt healed breccia containing quartz fragments in a dark, sulphosalt rich matrix
- Stage 3: quartz-jarosite breccia comprised of cream to yellowy massive quartz +/- jarosite with variable fragments of banded quartz
- Stage 4: massive white quartz that occurs in bands up to four meters wide that anastomose through the core of the vein
- Stage 5: cockscomb textured amethyst as a late open space filling

In general, the highest grades are associated with stages 1 and 2

Gold and silver mineralization occurs as native gold, electrum, acanthite, freibergite/tetrahedrite, stephanite and to a lesser extent pyrargyrite and other sulphosalt minerals. Electrum and native gold is free and occurs adjacent to or within the silver sulphosalts and sulphides. Kaolinite, illite, smectite, and montmorillonite are the dominant clay species with jarosite and minor gypsum present in the upper parts of the zone associated with the clays. Scorodite, after arsenopyrite, is associated with the jarositic fracture filling. A minor amount of chlorargyrite was noted in a single trench sample; it is associated with late jarositic fracture infilling. Adularia, sericite/illite, and clay (smectite + kaolinite) are the dominant alteration minerals associated with the multiple phases of quartz within the veins. Quartz ranges in character from chalcedonic to finely crystalline; coarser comb-textured amethyst is locally present. Banded opaline quartz is virtually absent.

Two Big Bend cross sections, 90972 N and 91030N, are presented as Figure 10.4 and Figure 10.5.

Figure 10.4: Big Bend Zone Section 90972N

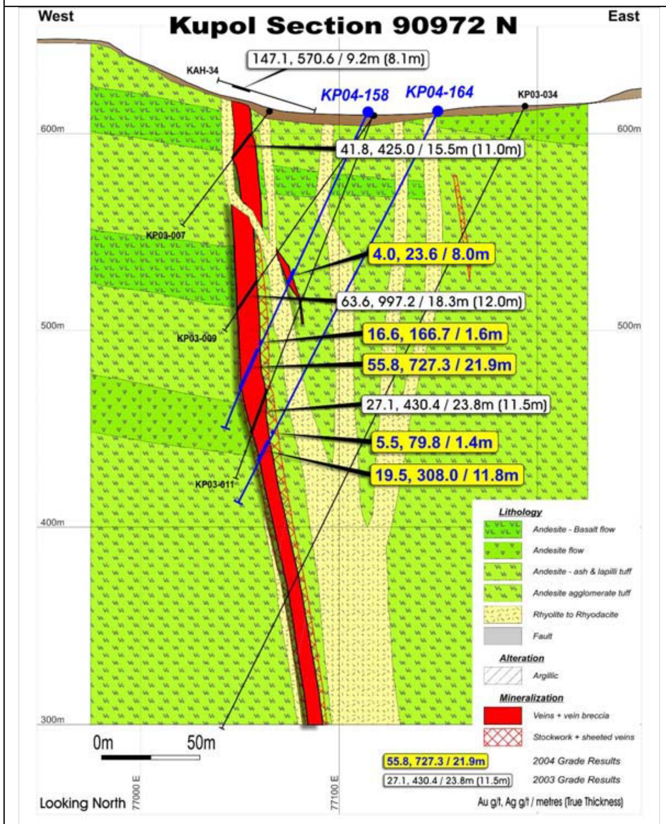
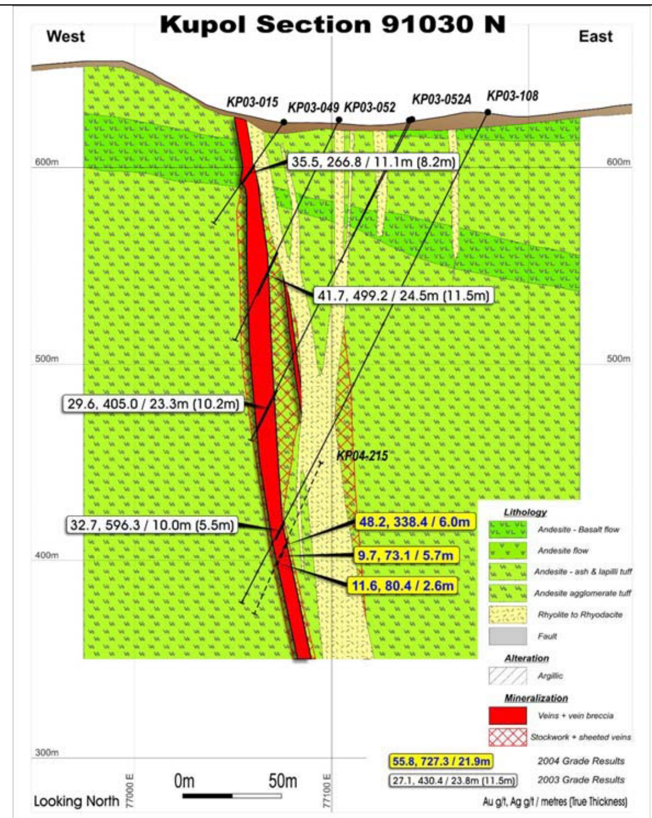


Figure 10.5: Big Bend Zone Section 91030N



10.4 Central Zone: 91275N to 92100N

The Central zone covers an 825 meter strike length of the Kupol vein structure. The zone was tested by fifty-nine drillholes totaling 9746.5 meters in 2004. Drill spacing on the upper levels of the zone ranges from 25 meters to 50 meters, but drillhole spacing in the deeper levels (>75 meters depth) varies from 50 to greater than 100 meters. The zone was drilled to a maximum depth of 430 meters below surface on a single section (91960N). The bulk of the 2004 drilling was biased toward definition of near surface mineralization.

Mineralization in the Central zone is hosted within one to two veins, as opposed to the single vein in the Big Bend zone. These veins occur within a wider, lower grade, sheeted and stockwork zone. Shoot development in this zone appears to be related to dilatant jogs in the vein structure and is possibly related to junctions with northeast and/or northwest trending structures. Individual veins range in width up to fifteen meters but are commonly less than five meters wide. The dip of the veins is shallower at 72 to 78 degrees in the central and southern portions of the zone but steepens to the north. The main rhyolite dyke bisects the zone,into hanging wall and footwall vein segments; this is similar to the Big Bend zone. The dyke diverges to the northwest, away from the zone at approximately 91925N.

The shallower portion of the zone, between 500 to 600 meters elevation, is bisected by a fault zone. Results from holes KP04- 363 (24.90 g/t gold and 493.21 g/t silver over 15.0 meters[10.12 m true width]) and KP04-232 (20.0 g/t gold and 244.29 g/t silver over 9.70 meters[6.96 m true width]) coupled with the results from KP04-403 (23.53 g/t gold and 753.55 g/t silver over 1.80 meters[1.2 m true width]) show that mineralization exists at depth below this barren fault and dyke zone. This indicates that the high-grade mineralization continues north at depth from the Big Bend zone.

The zone is comprised of three high-grade shoots separated by lower grade zones, dykes, and faults. The high grade shoots range in length from 75 to 175 meters. They are defined to the 400 meter elevation level and are open at depth.

Significant results from the northern ore shoot include:

- KP04-298 - average 17.16 g/t Au and 143.14 g/t Ag over 16.8 meters (9.38 meters true width)
- KP04-299 - average 71.6 g/t Au and 2674.49 g/t Ag over 8.5 meters (4.23 meters true width)

Significant results from the southern ore shoot include:

- KP04-363 - average 24.90 g/t Au and 493.21 g/t Ag over 15.0 meters (7.49 meters true width)
- KP04-399 - average 223.90 g/t gold and 819.31 g/t silver over 4.90 meters (3.65 meters true width) and average 159.91 g/t gold and 553.87 g/t silver 6.10 meters (4.58 meters true width)

The Central zone, between 91300N and 91520N had been exposed and channel-sampled at four-meter centers prior to 2004. The vein system ranges in width from five to thirty meters and is limited to the east by a four to seven meter wide rhyolite dyke. There is good continuity of grades within and along strike in this exposed vein area, with the highest grades encountered at the south end of the vein and in the western, hanging wall vein.

Sparse drilling deeper in the main vein system encountered sub-economic but anomalous gold values in cyclic banded and chalcedonic quartz, suggesting the occurrence of boiling to at least 430 meters below surface (240m elevation).

Sulphosalt concentrations are generally lower in this zone and there is a higher percentage of pyrite, especially north of 91770 N. The precious metal-rich fluid pathways in the central and northern portion of the Central zone are more constrained. Lower-grade crustiform and chalcedonic veining occurs adjacent to higher-grade sulphidic colloform-banded brecciated vein material. Banded, crustiform, opaline and chalcedonic quartz is more common in this zone than in the other zones and occurs to depths of up to 350 meters below surface.

The zone has a chlorite-pyrite metasomatic overprint at depth, which is similar to the deeper part of the Big Bend zone. This overprint starts at an elevation of approximately 560 m in the southern part of the zone and continues to below the 400m elevation over the remainder of the zone.

Petrography of samples from near surface in the Central zone indicates similar mineralogy to the Big Bend zone. One sample from 225 meters deep (KP03-99) indicates a complex paragenesis as follows:

- quartz-adularia vein
- breccia infill by a sulphidic (pyritic) iron-carbonate phase
- base metal (chalcopyrite, sphalerite), arsenic (arsenopyrite) and gold-silver (electrum, acanthite, freibergite) bearing phase;
- at least one other phase of quartz + adularia
- partial recrystallization by the thermal aureole

10.5 North Zone: 92100N to 92425N

Forty-four holes totaling 9,995.8 meters were drilled in 2004. Drill spacing was 25 to 50 meters near surface and 50 to 100 meters at depth. The zone has been drilled to a vertical depth of 425 meters (210 m elevation). In addition, the zone was stripped over 300 meters of strike length to 92400N. The southernmost sixty meters was mapped and channel sampled along lines spaced at ten meters apart.

The northern end of the North zone is defined by a point where the top of the vein system starts to gradually plunge northward under a cover of strongly clay altered volcanics. It is uncertain if the plunge is a function of a change in hydrothermal gradient (possibly due to paleo-topography) or fault controls. The southern limit is defined by the start of the North zone high grade ore shoot.

The vein system is laterally offset forty meters by the Premola Fault (92250N), a west-northwest trending dextral-normal fault. Farther north, a second fault is inferred at about 92340N. Here, the Main Marker unit is down-dropped to the north by fifty meters and the vein is displaced dextrally by up to twenty meters. A third shallow dipping fault truncates the vein and a parallel rhyolite dyke at about the 100 m elevation on section 92400N.

Although up to five to six veins are locally present, the North zone is primarily made up of two main veins separated by up to twenty meters of stockwork, with the east, hanging wall vein commonly wider than the west, footwall vein. These two veins coalesce near surface above 550 m elevation. A third vein, locally well mineralized, occurs up to twenty meters west of the two veins between 92350N and 92470N. Narrower (<2m), occasionally gold bearing veins occur up to fifty meters east of the main vein system. Exposure, through stripping, of the southern portion of the vein indicates a complex anastamosing vein zone up to 30 meters wide with individual veins to five meters wide.

Silver to gold ratios range from 5:1 to 15:1 in the east and 6:1 to 35:1 in the west. Grades are generally higher and more continuous in the east vein than in the west vein.

Significant results from 2004 drilling include:

- KP04-267 – average 31.31 g/t Au and 330.33 g/t Ag over 30.5 meters (17.96m true width) (HW + FW veins)

- KP04-280 - average 30.27 g/t Au with 281.46 g/t Ag over 15.2 meters (7.2m true width) (HW vein)
- KP04-286 - average 31.66 g/t Au with 484.31 g/t Ag over 20.6 meters (7.24m true width) (FW vein)

The textures and the character of the quartz and clay mineralogy, even at depth, suggest the intersection of the upper levels of a bonanza epithermal system. Lattice, frothy, vuggy and drusy textures at surface between the Premola and North faults suggest a vigorous boiling environment potentially close to the paleosurface or vadose zone. No consistent changes in silver-gold ratios are present to suggest a progressive change toward the roots of the boiling zone. Examples of high-level style boiling textures present at depth and at surface include quartz pseudomorphing of bladed calcite, opaline quartz infilling of voids, crustiform chalcedony and multiple re-healed breccia phases. In general, there are less sulphosalts within this area than in the Big Bend zone. There is a drop in grade in drillhole KP04-159 (8.87 g/t gold and 128.97 g/t silver over 10.0 meters) suggesting that the lower limit of the precious zone may be 400 to 450 meters below surface. A undercut of this drillhole with KP04-202A failed to intersect any vein; this suggests that a fault that offsets the zone is present at depth or that the vein rapidly pinches to depth.

A representative North Zone cross section, 92352N, is presented as Figure 10.6.

10.6 North Extension Zone: 92425N to 93150N

Twenty-four holes totaling 8,778.2 meters tested the North Extension zone in 2004. Drilling density is low with drilled sections spaced 50 meters between 92450N and 92600N to 100 meters or more north of 92600N.

The two veins comprising the North zone continue northward, locally as one vein, under a cover of clay altered volcanics, to about 92750N where they pinch out. A second set of veins, seventy to 100 meters east of the main vein system, continues northward from 92590N for about 360 meters to 92950N. The eastern vein system comprises up to four veins up to five meters wide within a ten to twenty meter wide, north-northwest trending zone that remains open to the north and at depth.

Significant results from both vein systems include:

- KP04-240 - average 37.03 g/t Au and 293.62 g/t Ag over 4.85 meters (2.15m true width) (main vein system)
- KP04-251 - 51.78 g/t Au and 229.67 g/t Ag over 6.7 meters (2.52m true width) (main vein system)

- KP04-204 - 41.78 g/t Au and 374.05 g/t Ag over 2.7 meters (1.47m true width) (eastern vein system)
- KP04-361 – 14.53 g/t Au and 242.21 g/t Ag over 9.3 meters (5.4m true width) (eastern vein system, northern most intersection)

The two vein systems are covered by a 100 to 150 meter thick cap of intense kaolinite + montmorillonite altered andesite pyroclastics and flows. This alteration zone manifests itself as a broad north trending magnetic low that extends to the limit of the survey. No soil or rock geochemical anomaly is present over the zone. Minor stringer veins, with low gold and silver values occur in vuggy siliceous pyrite-rich veins within the lower levels of the alteration blanket and demarcate the start of the transition into the precious metal zone. Local areas of vuggy silica alteration are present within the clay alteration zone; this is further evidence of an acid leach environment. The clay blanket, together with the presence of amorphous silica colloform banding, kaolinite and open space filling suggests that the veins occur in the higher levels of an epithermal system. The alteration above the veins is inferred to be the steam heated alteration zone at the top of the epithermal system (Hedenquist and White, 2005).

An inferred fault down-drops the two vein systems by 100 to 150 meters between sections 92600N and 92725N. This is based on the rapid northward thickening of the overlying clay altered cap and the steep northward plunge of the top of the main vein system here. The most northerly hole, KP04-195, intersected 8.64 g/t Au and 33.98 g/t Ag over 1.0 meter suggesting that the zone has potential for an on-strike extension of at least 200. Based on the intersection in KP04-361, this hole is inferred to be above the main zone. Well developed colloform ginguro banding is present at depth in KP04-204 but is less well developed in KP04-361.

Both vein systems are weakly chloritic and pyritic at depth (mostly below 250m elevation). This may be due to thermal, and hence mineralogical, zonation within the deposit or a late overprint, similar to that in Central and Big Bend.

A representative North Extension cross section, 92950, is presented as Figure 10.7.

Figure 10.6: North Zone Section 92352N

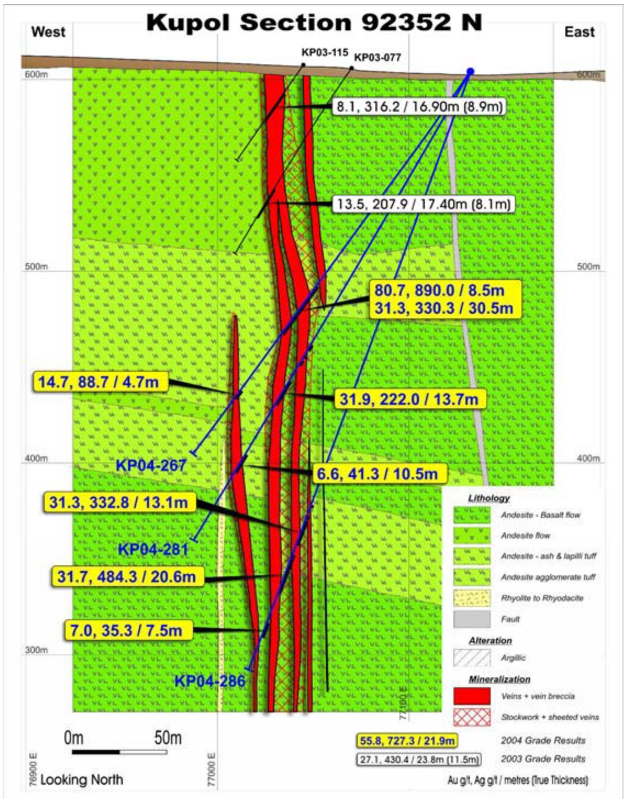
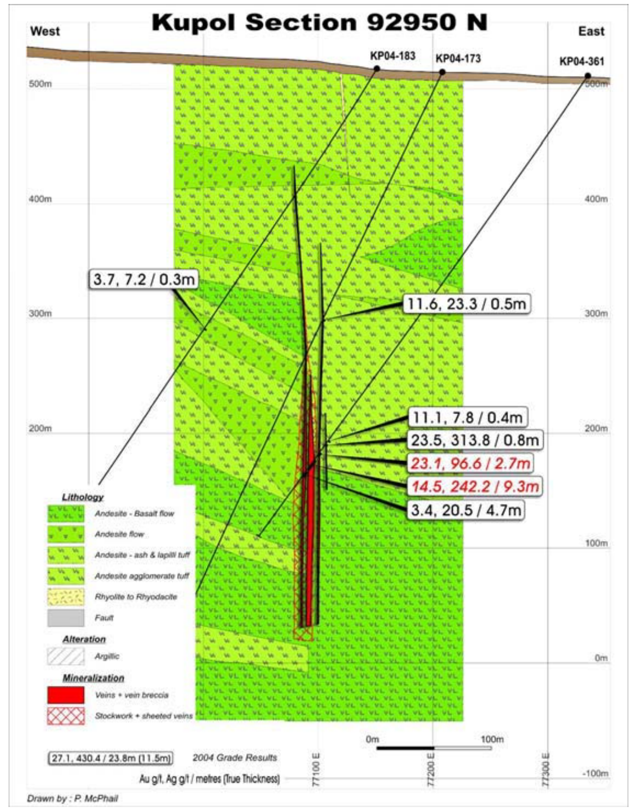


Figure 10.7: North Extension Zone Section 92950N



10.7 Replacement Drilling

The Russian holes drilled prior to 2003 were excluded from the resource and geological interpretation for the following reasons:

- Questions about accuracy of collar and downhole surveys; down-hole surveying was not performed on all holes due to difficult ground conditions.
- There was no geological QA/QC program to confirm or check the results.
- There was an internal laboratory control program but the control results could not be verified.
- The whole core was sampled, so representative core does not remain to verify results.
- The drill program was not supervised by a qualified person.
- Whenever the drillholes were utilized for geological interpretation there was a resultant complication that was more likely a function of inaccurate surveys rather than incorrect geology.

All pre-2003 Russian drillholes, except those noted below, were 'replaced' with new ones drilled in 2004.

The geology of the holes CKB-4, 10 and 19 were included in the resource model for the following reasons:

- There was not enough time to drill new holes to replace these ones.
- The geology of the holes is such that the exclusion of these drillholes would have a material effect on the interpretation of the vein.

Due to poor core recoveries in the drillhole KP03-144 it was re-drilled as hole KP03-240; KP03-144 was excluded from use in the resource estimations.

10.8 Logging Protocols

This section includes detailed information on the logging protocols used at the Kupol project in 2004, including:

- geological logging
- geotechnical logging

10.8.1 Geological Logging

A quick log for each hole was completed by the drill rig geologist responsible for that hole. Detailed logging was conducted by university trained, professional Russian geologists. Logging is onto paper forms.

In addition to the lithology, the colour, grain size, structures, core axes angles and the intensity of occurrence or non-occurrence of the following geological characteristics were noted on the detailed logs:

- Oxidation type, mineralogy, and intensity
- Mineralization
    - Pyrite
    - Chalcopyrite
    - Sulphosalts
    - Acanthite
    - Arsenopyrite
    - Visible gold
- Alteration
    - Silicification
    - Carbonate
    - Propylitic
    - Argillic
    - Sericite-adularia
- Vein texture and intensity
- Magnetism
- Structure and bedding

Most of the parameters were logged categorically, using integers of 0 (absent) to 3 (strong). Additionally, all the information that was codified or categorically logged was fully described in text. The original logs also contain a graphical log.

Vein intervals for the first forty drillholes of 2003 were re-logged to ensure conformity of early logging with later logging.

10.8.2 Geotechnical Logging

The protocols for geotechnical core logging program were established by Bruce Murphy, Senior Rock Mechanics Engineer, SRK Consulting. Bruce Murphy also provided training. Audits of the data collection were done at site by SRK and AMEC personnel later in the 2004 field season.

Total core recovery, rock quality designation (RQD), rock strength, length of broken zone, percentage of weak rock and fracture counts for all were routinely recorded by geotechnicians. Additional detail such as the fracture fill information was recorded in accordance with the protocols defined by SRK. Predominant fracture orientations, fault attitudes, and fault gouge zones were recorded by the geologists in the detailed logs. In 2004, all core was photographed while dry; only the sampled intervals were photographed while wet. The digital core photographs are named and stored in a logical and consistent manner.

Point load testing was also conducted throughout the 2004 drilling season.

10.9 Survey Data

The following section describes topographical surveys, drillhole collar surveys, trench and channel sample surveys, and downhole surveys.

10.9.1 Topographical Surveys

The Kupol area is covered by Russian State non-classified topographic maps at 1:200,000 and 1:100,000 scale and by classified maps at 1:25,000 scale. An area of eight square kilometers around the Kupol deposit was surveyed in detail to create a 1:2000 scale map with two meter contour spacing. A survey control net, lain out in local grid coordinates with a classified origin, is tied to the regional survey control points. Most control points were shot in 2000; additional survey control points were added in 2003 and as required in 2004. These points are used by exploration and engineering/construction for survey control.

In 2003 and 2004, sections of the Kupol area have been surveyed in detail and have been incorporated into the official topographic map.

10.9.2 Drillhole Collar Surveys

A local grid (LG) system is the official datum for the Kupol project. All surveying is conducted using this datum. The control points used were those established in 2000 by the Russian surveyors and in 2003 by Design Alaska (Fairbanks, Alaska). In 2003, the Gauss-Kruger (GK) geodetic system was used as the official datum. Prior to the 2004 drilling campaign, all coordinate data was converted from GK to LG.

All surveying was conducted by qualified Russian surveyors. Drillhole collar locations were preserved with four-inch PVC pipe branded with the drillhole name that was placed immediately after the drill rig pulled off the setup.

In 2004, surveys were performed using a Trimble total station device connected to an HP data collector. Representatives from Design Alaska trained the Russian surveyors in methodology and in the use of the instrument. Survey point coordinates, expressed in LG, were calculated automatically by the instrument. The drillhole collars were surveyed while the drilling was in progress. Points on the rig set up were also surveyed in order to determine the drillhole orientation at the collar. The final collar coordinate and the azimuth of the drillhole were calculated automatically by the device; the inclination was calculated in a spreadsheet using trigonometric functions.

To confirm the survey, a few drillhole markers were surveyed after the drill rig left the site; the differences between the two sets of coordinates were insignificant.

The survey coordinates and orientation information were presented in a single spreadsheet. The results were finalized in certificates that were signed by the surveyor; copies of the field notes were attached.

In 2003, the collar locations were determined by surveying the drillhole marker using conventional theodolite and survey rod instrumentation. The coordinates were manually calculated, entered into a spreadsheet, and presented in both the GK and LG.

In late 2003, all the preserved collar markers were re-surveyed by Design Alaska using total station instrumentation and reported in LG coordinates. These results were compared against the surveys produced by the Russian surveyors. In general, the coordinates correlated well, with variably oriented discrepancies from less than one meter to seven meters. In cases where there were significant differences, both sets of coordinates were checked and the hole was resurveyed as necessary.

If a drillhole collar was not surveyed then the proposed coordinate for that hole was used. The holes affected were KP04-219, -295, -304, -358A, and -398A.

In 2004, no external audit was performed on the collar survey data. However, a few 2003 and 2004 collar locations were resurveyed by the Russian surveyors using the total station; the differences were insignificant. A loop between known control points was completed on a regular basis to check on the accuracy of the survey instrument. The Trimble total station instrument was calibrated and certified by the manufacturer before use in the 2004 program.

The collar locations for pre-2003 drilling are accepted as provided in the original Russian data files. No supporting documentation was located.

10.9.3 Channel Sample Surveys

In 2004, sections of the vein in the Big Bend, North and South zones were exposed, pressure washed, mapped and sampled. As control for the mapping and sampling, the surveyors established a five-meter-by-five-meter grid over the exposure. After the sampling was completed, the locations of the start and end of each sample were surveyed. These coordinates were provided in a series of data files.

The survey points for pre-2003 mapping and sampling were provided in a single data file. These coordinates cannot be verified; however, they are closely represented in the original hand-drawn maps.

10.9.4 Trench Surveys

In 2003 and 2004 trench excavations were surveyed only at the locations need to accurately represent the excavation – this includes the start, the end, and any inflection points within.

The survey points for pre-2003 mapping and sampling were provided in a single data file. These coordinates cannot be verified. In many cases, the coordinates are suspect and the affected trenches have been excluded from the model.

10.9.5 Downhole Surveys

Drillholes

Downhole surveys were measured using a Reflex EZ-Shot electronic solid-state single shot instrument. The measurements were read directly from the display on the instrument and transferred, by hand, by the driller or geologist, onto paper slips that were submitted to the data department at the end of each shift. There is no permanent record of the readings; however, the instrument stored up to fifty readings so that the data entry for questionable measurements could be confirmed soon after the survey.

The information from each paper slip was hand-keyed into a spreadsheet. The azimuth readings recorded by the instrument were relative to magnetic north. To convert to local grid, the azimuth reading was adjusted by -1.9347 degrees, which was the declination (1°53' W) in 2003.

The drill was aligned with bearing pickets that were set by the surveyor. The head angle was set using a Brunton compass and was checked after the casing was set. In 2003, the setup orientation was used as the initial (zero depth) record. In 2004, and occasionally in 2003, the initial azimuth and inclination were derived from a survey of the drill ram and rods while the drill rig was on site.

Downhole measurements were taken at depths of twenty-five meters, fifty meters and at fifty-meter increments thereafter. In shorter drillholes, measurements were taken at appropriately spaced increments. On the Boart-Longyear drill rigs the test were taken by the driller as the hole was being drilled. On the Anyusk CKB-4 drill rigs, the tests were performed by the rig geologist on retreat from the hole.

Downhole survey readings that were clearly erroneous were excluded from the database; the physical record is preserved. Several holes lack downhole surveys due to caving, abandonment or a lack of instrumentation. The following holes were affected: KP03-012, -037, -039, -042, -043, -05, -073, -085, -098, -107, -109, -130, -145, -146, -148, -149, -152, -153, KP04-259, -271A, -295, -304, -313, -326, -330, -358A, -368, -398A, -440, -447, and -451.

Trenches and Channels

Channels from the exposed veins and trenches were treated as drillholes. The downhole survey and distance, for each were calculated from the individual trench survey points.

11.0 SAMPLING METHOD AND APPROACH (Item 14)

11.1.1 Core Sampling

Drill core was delivered from the drills in covered wooden boxes. The core was either laid out or dead stacked prior to the logging geologist taking possession. In 2004, the quick logs and geotechnical logs were completed at the drill site. Detailed geological logging was completed in core tents by Russian geologists. The core was photographed by the logging geologist immediately after it was logged.

Sampling intervals were determined, marked up, and tagged by the Russian geologists. The intervals were based on geology (lithology, mineralogy, texture and structure). Sampling across contacts was only permitted if the vein width was less than the minimum sample width. The core was manually oriented to ensure that the core was consistently split and that there was no sample bias.

The minimum sample length was 0.25 meters for HQ diameter core and 0.30 meters for NQ diameter core. Generally, the maximum sample length was one meter. Mineralized zones were bracketed by a minimum of one to three meters of sampling into the footwall and hanging wall. All vein zones and alteration types of interest were sampled and each major zone was continuously sampled.

Samples containing visible gold or abundant sulphosalt mineralization were indicated by a white sample bag at the start of the sample interval, so sampling technicians would employ contamination minimization protocols during cutting and laboratory preparation. Field duplicate samples were marked with flagging tape.

Core to be sampled was delivered to the splitting shack and either taken inside or dead-stacked on pallets outside. Core was 2/3 split using a diamond saw; the remaining third was returned to the core box as a permanent record. The rock saw core jig was calibrated to ensure that an even 2/3 split was taken of the core for both HQ- and NQ-sized samples. For samples of strongly broken core, care was taken to ensure a 2/3 split of the sample. This commonly involved the use of a metal divider and a spoon. The core was split in consecutive sampling order, from top of hole to the bottom. Field duplicate samples were created by cutting the 2/3 split into two 1/3 sections; both samples were sent for analysis.

The saw blade was cleaned on a regular basis using a dressing stone, and was cleaned after every sample that was well mineralized or contained visible gold. Fresh water was used at all times to protect against re-circulation contamination.

Samples were bagged and field blanks/reference standards were inserted into the sample stream by the geologists. The samples were assembled into batches of twenty, in the order they were sampled, and submitted to the laboratory two to three times per day. Well-mineralized or visible gold-bearing samples were indicated on the submission form to ensure that contamination reduction protocols were followed by the laboratory.

The remaining core for all 2003 and 2004 mineralized intersections is stored in racks in a locked core storage tent. The remaining un-mineralized core is stored in racks in locked containers (2003) and the in open racks (2004). No mineralized intersections remain from the pre-2003 Russian drilling because whole core was consumed for sampling. The remaining un-mineralized core from the pre-2003 drilling has been organized and stacked.

PQ core for metallurgical testing was shipped offsite as whole core. Standards and blanks were submitted as quality control samples; there were no duplicates.

11.1.2 Trench Sampling

Trench sampling followed the same sampling and quality control protocols as the cores. In excavated trenches, samples were collected using a chisel and hammer to cut an even channel across each zone. Care was taken to collect equal volumes of rock across the sample channel to ensure that there was no sampling bias based on rock softness or fracture density.

11.1.3 Channel Sampling

In 2004, portions of the Big Bend, South, and North zones were stripped of cover and pressure-washed. The channel edges were cut using a diamond rock saw, and the samples were chiseled from the cut and collected into plastic sample bags. Sample intervals were marked with metal tags. The same quality control protocols as for core were employed.

12.0 SAMPLE PREPARATION, ANALYSES AND SECURITY (Item 15)

Due to the remote location of the Kupol Project and the difficulties with shipments of samples within and from Russia, a containerized field laboratory was set up at the Kupol site. The laboratory was set up and run as an independent 'arms length' laboratory that operated as a Russian-certificated Anyusk State Mining and Geological Enterprise field laboratory (Kupol Laboratory). The laboratory was overseen by qualified North American laboratory managers that supervised Russian-certified laboratory managers and assayers. No non-laboratory personnel were allowed in the laboratory areas unless accompanied by a laboratory manager. The laboratory procedures and internal laboratory protocols were audited in 2003 and 2004 by B. Smee of Smee and Associates Consulting Ltd. (Sooke, BC).

Samples were received at the laboratory as follows:

- Samples were delivered to the laboratory by the sampling technician accompanied by a submission form signed by the geologist and the sampling technician
- The submission form and samples were checked for accuracy and completeness
- The samples were logged into the laboratory system
- A laboratory technician signed the submission form, made a copy of the submission form and returned the original to the sampling technician
- The samples were placed in a secure container prior to processing

The sample preparation and assay procedure was as follows:

- All samples were dried in a locked, heated container, either within the sample bag or on a steel tray. Dried samples were transferred to the sample preparation area.
- Each sample was crushed in a jaw crusher to 95% of minus 10 mesh (<2 mm) and then divided by a Jones riffle splitter into two one-kilogram samples. The first sample was preserved as a geological coarse reject that was kept in sealed plastic containers; the second sample was passed on for further processing.

- The sample was pulverized to 90% minus 150 mesh (.005mm) in a LM2 bowl and puck pulverizer. The pulverized sample (pulp) was split into four 250 gram samples that were placed in kraft paper sample envelopes. One pulp sample went for fire assay, one kept as a lab reject, and two were retained as geology duplicates. All pulps are stored in locked containers.
- A fifty gram split of the pulverized sample was analyzed for gold and silver using standard fire assay techniques with a gravimetric finish.

For each twenty samples, one additional sample was split from both the crusher and pulverizer splits to ensure compliance with laboratory quality control specifications.

All equipment was air-washed between samples. A blank silica sample was run as a cleaning medium every twenty samples, and after samples with visible gold or strong mineralization.

12.1 Quality Control

12.1.1 Field Data

The field quality control program for 2004 included the insertion of standard reference material (standards) to monitor accuracy, coarse blank material (blanks) to monitor contamination and sample mix-ups, and field duplicates (duplicates) to monitor precision. This program was used for drill core, trench, channel, and rock samples. These protocols were also used in 2003; there is no field quality control data for work conducted prior to 2003 (Table 12.1).

Table 12.1: Count of Field Quality Control Samples – 2003 and 2004

YEAR	No. of Samples	No.of Standards	No. of Blanks	No. of Duplicates	All Samples
2004	15049	900	1162	1025	18136
2003	8386	500	633	646	10165
All	23435	1400	1795	1671	28301

The performance of quality control samples was monitored on a daily basis as the results were received. The results were accepted or rejected based on criteria established at the beginning of the program (Table 12.2).

Table 12.2: Criteria for Rejection

	Sample Type	Rule
1	Standard	If the result is greater than three standard deviations from the mean, then it is a failure; tests accuracy
2	Standard	If the results for two adjacent standards are greater than two standard deviations from the mean, on the same side of the mean, then they are failures; shows bias
3	Blank	If the result is greater than the warning limit, then the sample is a failure; the warning limit is 0.5 gpt; shows contamination

If data was rejected, it was deemed a failure and withheld from the project database until the cause for the failure was determined or the samples had been re-analyzed and the results accepted. Requests for re-analyses were made immediately and the new results were returned within two days. The results were charted bi-weekly or monthly.

There are no outstanding issues regarding the quality control data.

The quality control program has been audited by B. Smee (Smee and Associates Consulting Ltd). He concluded that the field quality control program is producing data that meets or exceeds the requirements of NI 43-101 and is of a quality suitable for inclusion in resource estimations.

12.1.1.1 Standard Reference Material

In order to monitor the accuracy of the laboratories, several gold reference standards, covering a range of grades, were purchased from CDN Resource Laboratories (CDNRes, Canada) and ROCKLABS (RL, New Zealand). The gold concentrations for these standards range from 1.75 gpt to 33.50 gpt (Table 12.3). The standard samples are not blind to the laboratory; however, the large number of different samples, some with very similar gold grade, help prevent the laboratory from guessing at values.

Table 12.3: Reference Samples Used in 2003 and 2004 – Kupol Laboratory

			Au (gpt)		Ag (gpt)			
STDName	**Source**	**Count**	**Mean**	**StdDev**	**Mean**	**StdDev**	**Used2003**	**Used2004**
GS-4	CDN Resource Lab	166	3.45	0.105			YES	NO
GS-5	CDN Resource Lab	119	20.77	0.455			NO	YES
GS-6	CDN Resource Lab	165	9.99	0.250			YES	NO
GS-7	CDN Resource Lab	107	5.15	0.230			NO	YES
GS-8	CDN Resource Lab	260	33.50	0.850			YES	YES
GS-9	CDN Resource Lab	102	1.75	0.070			NO	YES
GS-12	CDN Resource Lab	98	9.98	0.185			NO	YES
SI-15	ROCKLABS	87	1.80	0.067	19.68	1.02	NO	YES
SN-16	ROCKLABS	97	8.37	0.217	17.64	0.96	NO	YES
SP-17	ROCKLABS	101	18.13	0.434	59.16	2.95	NO	YES
SQ-18	ROCKLABS	84	30.49	0.880			NO	YES

The CDNRes standards of sixty or seventy-five grams came in individual Kraft paper envelopes. The RL standards of fifty grams came as individual plastic wrapped sachets. The standards were composed of pulverized material. Each standard is certified with an accepted mean as obtained through a round robin assay program.

Standard samples were inserted into the regular sample stream at a ratio of 1:20, according to a predetermined schedule based on the sample number.

In 2004, of the 900 standards analyzed, eighty-one standards, about 9.0 %, failed. These were due to sample mix-ups, data entry errors, or laboratory failures. Laboratory failures counted for 4.7%, a rate comparable to that of western laboratories. The failure rate was slightly higher nearer the beginning of the season but decreased and remained constant throughout the remainder of the year. No biases were revealed.

In 2003, of the 500 standards analyzed, forty-seven, about 9.4 % failed. Laboratory failures counted for 4.4%. The failure rate was very high at the beginning of the season and virtually non-existent by mid-season. Two of the three standards showed a bias for gold values that was not observed in the silver values; however, all the results produced by the Kupol lab were deemed acceptable.

12.1.1.2 Field Blanks

In order to monitor contamination and sample mix-ups, field blanks composed of non-auriferous rhyolite derived from the north end of the Kupol property were inserted into the regular sample stream. The blank is composed of coarse material. Blanks were inserted at a ratio of 1:20 and after samples that displayed good mineralization or visible gold.

During 2003 and 2004, 1795 blanks were analyzed. There were nine blank failures representing a rate of less than 0.5%. The failed samples followed very high-grade samples so the affected batches were not re-analyzed.

The blanks were monitored for gold only. There is a broader range in silver concentration. A regression line through this data indicates that the laboratory was improving in all aspects of cleaning.

The data are free of contamination that originated during the sampling, preparation, or analytical processes. There are no outstanding issues regarding blanks.

12.1.1.3 Field Duplicates and Overall Precision

In order to monitor the precision of the laboratory, duplicate samples were inserted into the sample stream at a ratio of 1:20. Additional duplicates of well-mineralized samples were also inserted. The field duplicate was created by slicing a two-third core split lengthwise into two one-third splits. The remaining third of core remains in the box as a permanent record of the core.

Duplicates are not failed unless they are significantly different from each other or there was any other reason within the same analytical batch to suspect either a sample mix-up or analytical error. There were no failures in 2004.

In 2004, 1025 field duplicate samples were analyzed. A plot comparing the results from the two sets of analyses shows that there is even scatter around the 1:1 line and therefore no bias. However, the large scatter, especially at higher concentrations, is expected for Kupol-style mineralization which includes several occurrences of visible gold. There are no problems with the field sampling techniques.

Sampling precision was estimated by plotting the mean gold value against the absolute percent relative difference. A Thompson-Howarth chart could not be constructed for this data set due to the negative y-intercept created by the wide scatter at the higher gold concentrations. This relationship shows that the overall sampling precision at Kupol is about 35 to 40% at the 10 gpt gold concentration.

Two additional duplicate samples per twenty samples were collected by the laboratory, during the sample preparation process. A preparation (prep) duplicate was split off the main sample after crushing; a pulp duplicate was an additional fifty gram split from the same 250 gram split used for the original assay. These samples are part of the laboratory quality control to show the degree of sampling error that is present in the preparation and analytical process.

The results from the precision plots indicate that splitting the sample introduces about 10% error; the act of cutting the core adds an additional 20 to 25% error. These results are similar to other deposits with nugget gold.

The scatter in the data indicates that there is no sampling bias in the prep duplicate analyses. The wider scatter at higher concentrations indicates that there is some coarse gold; however, the regression line is in close agreement with the 1:1 line. This relationship shows that the overall preparation precision at Kupol is about 15% at the 10 gpt gold concentration.

The scatter in the data indicates that there is no sampling bias in the pulp duplicate analyses and there is excellent agreement between analyses. This relationship shows that the overall preparation precision at Kupol is about 5% at the 10 gpt gold concentration.

12.1.2 Laboratory Data

In 2004, there were 900 standard and 1,162 blank samples used in the laboratory.

12.1.2.1 Standard Reference Material

The Kupol laboratory used four gold reference standards to internally monitor accuracy. The gold concentrations for these standards range from 1.298 gpt to 15.15 gpt. The accepted values for each are summarized in Table 12.4.

Table 12.4: Laboratory Standard Reference Samples Used in 2004

STDName	Mean Au (gpt)	StdDev Au (gpt)
STD-A	1.298	0.066
STD-B	2.643	0.12
STD-C (before 25 August 2004)	1.844	0.098
STD-C (after 25 August 2004)	15.15	0.58

The Kupol laboratory inserted 362 samples of STD-A and 318 samples of STD-B. STD-C was divided between two standards: 153 samples of a higher grade sample prior to 25 August 2004 and 146 samples of lower grade after.

The assay results for these samples were reported with each assay file and certificate. These data were charted bi-weekly or monthly.

There were a few failures of STD-A at the time of the laboratory start-up, with only three accuracy failures after. A small negative bias drifts to no bias throughout the program.

There were four failures for STD-B. There is a slight negative bias but there is no drift throughout the program.

There were no failures of the higher grade STD-C, but there is a negative bias and a small negative drift throughout the program.

There were no failures of the lower grade STD-C, but there is a slightly negative bias and a small positive drift throughout the program.

12.1.2.2 Preparation Blanks

The Kupol laboratory inserted 980 preparation blanks during the 2004 programs. No failures were reported.

12.1.2.3 Laboratory Duplicates

The results from the 2004 field and laboratory quality control programs indicates that the Kupol laboratory is functioning at a quality that is equivalent or better than that of western laboratories. The analytical data are suitable for inclusion in resource estimations and meet the standards of NI 43-101.

12.1.3 External Check Samples

During 2004, selections of check samples were routinely sent to Assayers Canada Laboratory (Assayers) to be analyzed by the same method used by the Kupol laboratory. The samples were selected systematically: the pulp for each sample ending with a '9' plus one sample per mineralized intersection were assembled and shipped. Fresh standards matching the originally submitted standard were inserted into the sequences. No blanks or duplicates were used.

Additionally, a selection of vein samples, with assays greater than 3.0 g/t, from those submitted to Assayers was forwarded to ALS Chemex for assay. New standards that did not match the originals were supplied (Table 12.5).

Table 12.5: Count of External Check Samples

Laboratory	Number of Samples
Assayers Canada	2496
ALS Chemex	349

Each set of samples, as submitted to each laboratory, was free of any quality control issues. The method of analysis was a fifty gram fire assay with a gravimetric finish for both gold and silver.

12.1.3.1 Assayers Canada

In 2004, 2496 samples (approximately fifteen percent of the total sample set) were submitted to Assayers Canada Laboratory for analysis. A comparison of the standards analyzed by the Kupol laboratory and Assayers indicates that there is no bias toward one laboratory. The mean for each set of results is less than 3.0% from the accepted mean, both datasets are accurate.

The results from each laboratory were plotted against each other to show bias. X-Y plots show bias between laboratories; Q-Q plots show the grade ranges at which bias occurs.

The results from Assayers Canada confirm the results from the Kupol laboratory.

B. Smee of Smee and Associates Ltd has reviewed the 2004 check data from Assayers Canada. He determined that the results from Assayers Canada were an acceptable confirmation of the accuracy of the Kupol results.

12.1.3.2 ALS Chemex

In 2004, 349 samples were submitted to ALS Chemex for analysis.

A comparison of the ALS Chemex results to the Kupol laboratory results indicates that there is no bias for either gold or silver. A comparison of the ALS Chemex results to the Assayers results indicates that there is a weak bias toward ALS Chemex for gold and a slightly stronger bias toward Assayers for silver, especially in the 30 to 45 gpt range. A comparison of the Assayers results to the Kupol laboratory results indicates that there no bias for gold and a slightly bias toward Kupol for silver, especially in the 20 to 50 gpt range.

The sample results from ALS Chemex, Assayers Canada and Kupol laboratory were compared in a series of X-Y and Q-Q plots. These charts are presented as Figures 12.1 to 12.3.

Figure 12.1: ALS Chemex versus Kupol – X-Y and Q-Q Plots for Au and Ag

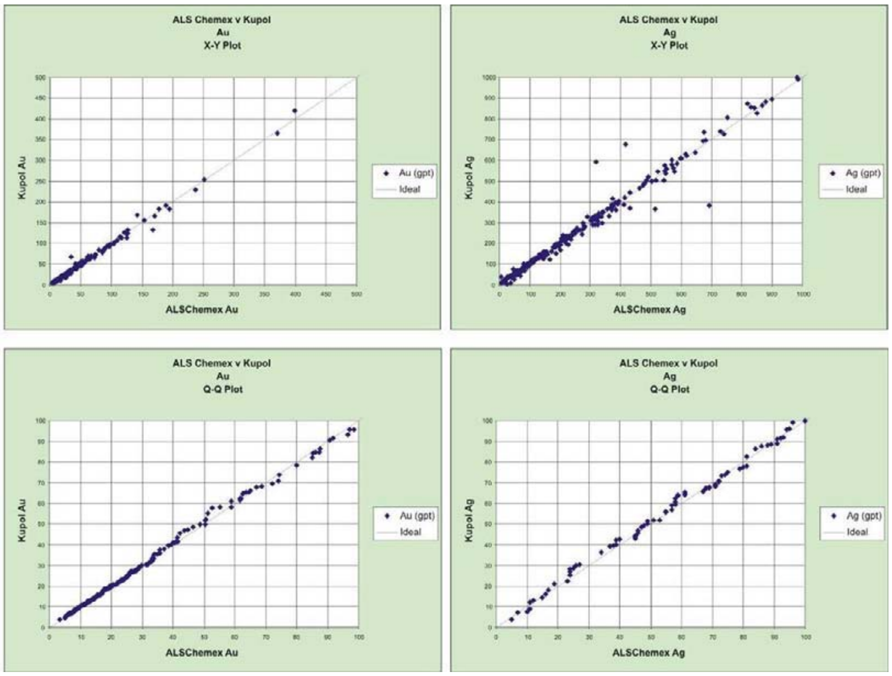


Figure 12.2: ALS Chemex versus Assayers – X-Y and Q-Q Plots for Au and Ag

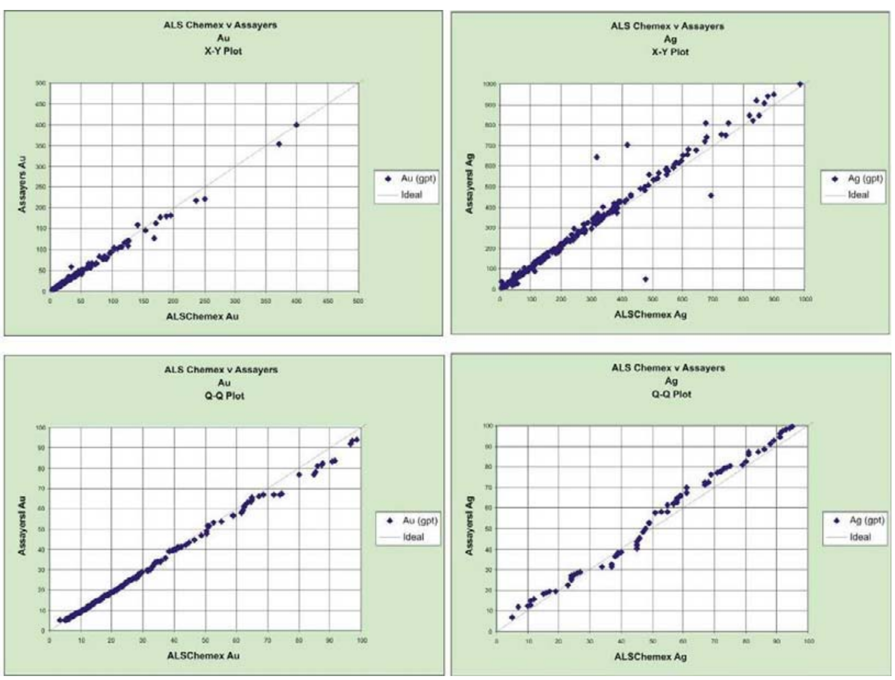
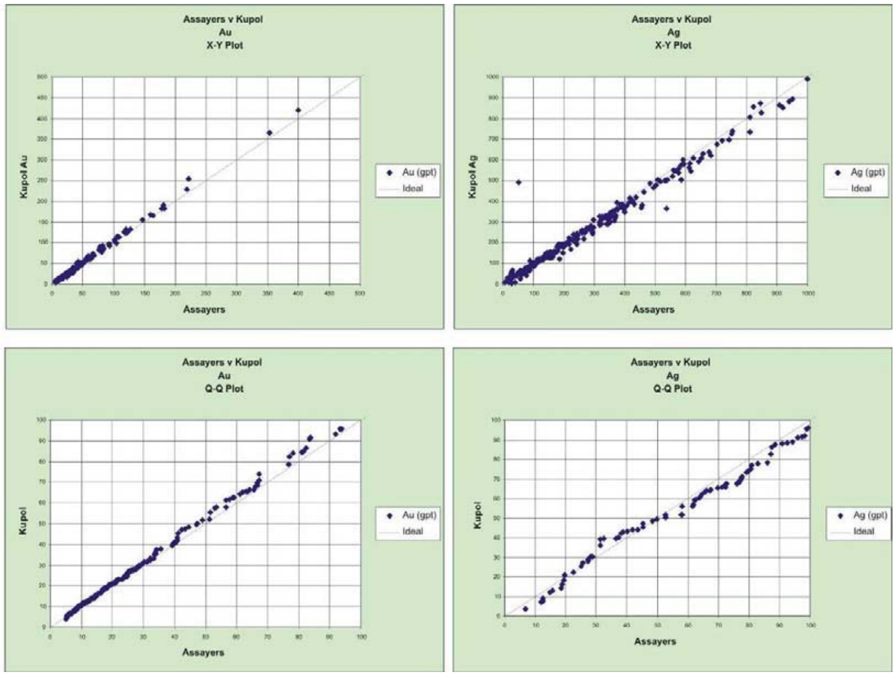


Figure 12.3: Assayers versus Kupol – X-Y and Q-Q Plots for Gold and Silver



12.1.4 Security

No unauthorized personnel were allowed in the core storage, logging, or cutting facilities during the core logging and sampling process. Core for sampling was delivered directly to the core-cutting tent or to a secure storage container before cutting. Lids were kept on boxes during transfer.

Once cut, the samples were assembled into batch shipments within the core-cutting tent. These batches were stored in sealed rice bags pending submittal to the laboratory. The batches were delivered, along with a sample submission form, to the laboratory several times a day. At the laboratory, each sample submission was checked for accuracy. The laboratory signed off on the receipt of the shipment and took custody of the samples. Non-laboratory staff was prohibited access to the samples after this point. Prior to processing, the samples were stored in a locked container.

External check sample shipments were assembled by the laboratory staff in accordance with a submission list prepared by the QC manager. Samples for each submission sealed within plastic *Secur-Pak* bags along with a submission form signed by the laboratory manager. The *Secur-Pak* bags were sealed in fiber bags that were shipped to Assayers Canada Laboratory. The laboratory received these secure bags, took inventory of samples, and transmitted a list of samples received back to the QC manager. The laboratory has never reported that the bags have showed evidence of tampering.

13.0 DATA VERIFICATION (ITEM 16)

During Mr. Crowl's site visit to Kupol in May 2005, core from several drill holes was examined and compared to drill logs and assay certificates. No discrepancies were noted. Further, examinations were made of the surface geological mapping of the South, Big Bend and North Zones. The mapping corresponded well to the outcrop geology.

13.1 Database Management

13.1.1 General

The database (KupolFeas25Jan05.mdb) used for the 2004 Kupol feasibility study is an MS-Access format file (Jet 4.0) that contains information for all holes drilled and trenches excavated on the property. This database is a subset of the data, time-stamped at 4 November 2004, extracted from the 'live' project database (GD_KupolProject.mdb) that was built and is maintained using GEMS 5.* geological software and MS-Access.

The six drillholes completed after 4 November 2004 are included in the dataset. Assay, lithology, specific gravity, and geotechnical information for all drillholes was updated and completed. Collar and downhole survey data was not changed.

The Kupol feasibility database (KupolFeas25Jan05.mdb) consists of the following tables (Table 13.1):

Table 13.1: Tables in Feasibility Database

Table Name	Description
Header	Collar information
Survey	Downhole survey information; includes zero distance (collar) record
Assay	Assay results; Au and Ag for the lead lab (usually Kupol) only
DetailLog	Lithology codes and categorical codes for alteration and mineralization from the detailed geological logs
TransLITH	Lithological description from the geological log;

	text in original Russian and translated to English
Geotech	Geotechnical logs
SG	Specific Gravity measurements
ICP	ICP data, from Assayers Canada Laboratory
Catalogue	Listing of Tables and field with descriptions

13.1.1.1 Database Creation

All modifications to the database were performed in GD_KupolProject.mdb, using either GEMS 5.* and/or MS-Access. KupolFeas25Jan05.mdb was created by running a macro in GD_KupolProject.mdb that created a snapshot of the data. Only the table and fields relevant to the modeling and grade interpolation were extracted; GD_KupolProject.mdb stores much more information than KupolFeas25Jan05.mdb.

The following section describes the creation of GD_KupolProject.mdb, which serves as the source of the feasibility database, KupolFeas25Jan05.mdb.

The database was originally built in 2003 from a collection of spreadsheets that originated in Russia and Canada and from data contained in the GEMS 4.11 project GCDBKU (GCDBKU.mdb). All 2003 data were compiled from hundreds of MS-Excel spreadsheets that captured drilling information. In 2004 data was compiled externally in databases and spreadsheets, checked, and then transferred into the project database.

Data validation was incorporated into the database to exclude invalid data from being loaded; however, this validation cannot prevent valid yet erroneous data from being loaded.

13.1.1.2 Data Entry and Data Management

Data Entry and Data Management

In 2004, all data entry and checking procedures were established and supervised by V. Park, P. Geo, the Kupol database and QC manager. Data entry clerks entered drill and trench data and translators converted geological descriptions from Russian to English.

For all cases except for collar surveys, assay results and translations, once the data was loaded into the project database it was removed from the working spreadsheet.

The database was kept current with drilling.

Header Table: The collar survey data was loaded directly from an MS-Excel spreadsheet maintained by the site surveyors. The file required minor manipulation to standardize formatting but the survey data was not adjusted. Final drillhole lengths were obtained from the geological log. All other information was added as it became available.

Survey Table: The downhole survey data was hand entered from paper slips into a spreadsheet and then imported or merged into the database.

Assay Table: The assay data was loaded directly from digital files provided by the laboratory into a QAQC database external to the project database. The assay intervals were loaded from a spreadsheet. The quality control samples were vetted and if valid then the assay table information was exported from the QAQC database and loaded into the project database. In all cases, values that fall below the detection limit are reported as half the detection limit – eg. <2.0 gpt Ag = 1.0 gpt Ag.

DetailLog Table: The data for this table, hand entered from the geological logs, was provided in a single spreadsheet file.

LithDESC/TransLITH: The text portion of the geological log was entered, in Russian and in English, into local databases external to the project database. Each translator or data entry clerk retrieved intervals or Russian text for translation from the MasterTranslation database. Once the translation was completed, it was sent back into the master database, from which it was exported to the project database.

Geotech Table: The data for this table was provided in a single spreadsheet file. The RQD and recovery values were calculated from the data within the database.

SG Table: The data for this table was provided in a single spreadsheet file. The specific gravity values were calculated from the data within the database.

ICP Table: The assay data was loaded directly from digital files provided by the laboratory into a QAQC database external to the project database, compiled and then exported to the project database. In all cases, values that fall below the detection limit are reported as half the detection limit – eg. <2.0 gpt Ag = 1.0 gpt Ag.

13.1.2 Hardcopy Storage

Physical data, such as handwritten logs and assay certificates, are the ultimate resource for information. At Kupol, most data is collected to paper and is then made digital. Each drillhole (or trench/channel) has its own file folder and all documents pertaining to that drillhole are stored within that folder. The types of records stored include collar survey certificates, downhole survey slips, geological and geotechnical logs, point load and density test forms, assay certificates, shift reports, timesheets and database reports.

All original documents are located at the Kupol site during the field season and in the Magadan office during the remainder of the year. An exact replica of these folders and their contents are maintained in the Vancouver head office.

Reports, created from the database, have been printed, bound and shelved as reference material.

Two sets of signed and stamped 'original' assay certificates were issued – one set for Canada, the other for Russia. These original certificates are filed, in bulk, in a different set of folders. Copies of the assay certificates were filed in the drillhole folders.

In 2004, duplicate signed 'original' collar survey certificates were issued – one for Canada, the other for Russia. These are filed in the appropriate drillhole folder.

The notes from the field books used by the geologists were copied and filed by drillhole. The original field books from 2003 are currently in Canada; the 2004 notebooks are in Russia.

Pre-2003:

All documents that originated in Russia are stored there. Copies of those documents have been filed and bound as described above. These files do not contain assay certificates. Some Russian assay certificates exist in a bound volume.

13.2 Data Verification

Most drillhole data was made digital manually. Surface survey information and assay results were provided in digital format. Hand-written documents were presented to a data entry clerk who entered the data into a spreadsheet. The quality of the data was checked prior to being loaded into the database and then checked later as an export from the database. Digital data was always checked against the original hand-written documents.

Once the data was entered, the clerk printed off her work and checked the digital data against the original document. Errors were identified and corrected immediately and the data was re-printed and submitted for a crosscheck by another clerk or the database manager. Any errors discovered at this stage were presented to the clerk who entered the data for correction. Once the data was deemed correct it was loaded into the project database and removed from the spreadsheet.

Data validation was built into the database to ensure that the data falls within acceptable limits. However, this validation does not protect against valid but erroneous data. The data entry clerks were diligent about checking for invalid and/or incorrect data. If a clerk encountered suspect data then she discussed it with the geologist or technician responsible for the document and the errors were corrected both physically and digitally.

Most errors were caught and corrected before the data was loaded into the database; therefore, true data entry errors are rare. There are, however, errors that are data collection errors – the data entry is correct but the original document is wrong; this type of error was common in geotechnical logs. These errors, when not identified during data entry, were identified in database export checks and then corrected on the original document and in the database. Other apparent data entry errors emerged when the original documents were changed but the digital records were not – a procedural problem that was corrected early in the 2004 program.

The database is routinely subjected to a validation provided by GEMS that checks for obvious errors such as inconsistent drillhole lengths, zero length intervals, out of sequence intervals and missing intervals. These types of errors are rare but when detected and found to be truly erroneous, are corrected and re-validated. Additionally, the data is subjected to a series of query-driven checks in MS-Access that check for invalid-but-untrapped or sloppy-but-valid situations.

All data from 2003 and 2004 has been subjected to rigorous post-load checking. The data from all tables was exported from the database and checked against the original documents. Some data, such as the downhole surveys, were checked multiple times. Any errors were immediately corrected.

All data included in the feasibility database has been validated and is of a quality adequate for use in resource estimations. It is recommended that on-going checks be performed to ensure and improve the integrity of the database contents.

For a complete description of the Data Verification programs for the Kupol Project, refer to Section 14 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR.

13.3 Bulk Density Measurements

A program to determine the in-situ bulk density (specific gravity) of major vein and non-vein rock types was conducted, at the Kupol site, during the 2003 and 2004 field seasons. Samples were systematically selected from drillholes KP03-118 through KP03-156 and KP04-162 through KP04-449. Random samples were selected from drillholes prior to KP03-118.

In 2003, bulk density testing was conducted on 488 samples using a plastic-wrapped/immersion method. In 2004, testing was conducted on 3223 samples using a wax-coated/immersion method; two glass standards were used for quality control. Sixty-nine samples were sent to a commercial laboratory (ALS Chemex, Canada) for independent testing using the wax-coat / immersion method. Additionally, studies to determine if there is a bias between different methodologies were conducted.

The results from the external checks indicate that when compared to the results from the ALS Chemex tests, the results from the 2003 plastic wrap / immersion are biased 3.2% lower and the results from the 2004 wax coat / immersion method are biased 1.68% lower. Based on the method comparison tests, techniques involving plastic wrap produce less accurate and precise results.

All bulk density data is stored in the SG (Specific Gravity) table in the project and feasibility databases.

A bulk density of 2.48 was used for the 2004 feasibility resource estimation. A bulk density of 2.55 was used for the 2003 preliminary economic assessment.

14.0 ADJACENT PROPERTIES (Item 17)

There are no adjacent properties as defined by NI 43-101.

15.0 MINERAL PROCESSING AND METALLURGICAL TESTING (Item 18)

Please see Section 20.2 of this Report.

16.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES (Item 19)

For a complete description of the Mineral Reserve and Mineral Resource estimates, refer to Section 17 of Garagan, T., 2005, Technical Report on the Kupol Project, Chukotka A.O., Russian Federation – Report for NI 43-101, dated 31 March 2005 filed on SEDAR.

16.1 Mineral Resource Estimate

16.1.1 Grade Distribution Studies for Gold and Silver

Statistical and geostatistical analysis of the data was completed to provide guidance for separating the deposit into divisions suitable for grade estimation (domains). The studies were undertaken on assays and composites.

The dataset includes most of the Bema diamond drilling and the available trench data. There are a few documented exceptions in each case.

16.1.2 Geological Interpretation

Most of the precious metal bearing mineralization at Kupol occurs within the vein lithologies. There is scattered high grade mineralization in the stockwork zone. Vein, stockwork, dykes, and major faults within the vein/stockwork area were interpreted on east-west trending cross sections. Logged lithology was the main control on the vein interpretation, with assay grade used locally depending upon the geometry of the mineralized material. The sectional interpretation was digitized and reconciled on levels; wireframe models were built from the reconciled sectional interpretations.

16.1.3 Block Model

The block model used for reporting resources and reserves was built using Datamine software. Blocks were coded with vein, stockwork, fault, dyke and basalt wireframes, which served as the primary control grade estimation. Additional items modeled include:

- Topography,
- Overburden,
- Argillic alteration,
- Intensity of Carbonate and Pyrite (for Acid Rock Drainage characterization)
- Metallurgical domains

16.1.4 Block Grade Estimation

Assay intervals, composites and blocks were coded from the vein, stockwork, dyke, fault and basalt wireframe models. A detailed review of the gold and silver distributions within the interpreted vein and stockwork zones guided the approach used for block grade estimation.

Gold and silver grade estimation within vein and stockwork (separately) were controlled by use of an indicator. The indicator was set to 1 if sulphosalts were logged as present or gold grade was greater than 11g/t. The gold indicator was also used for silver grade estimation. Variograms were modeled for the indicator variable, high-grade gold, high-grade silver, low-grade gold and low-grade silver populations using two variogram domains based on mineralization orientation.

Ordinary Kriging was used to estimate block values for each of these variables. Fifteen estimation domains were used to control block estimation search orientations. The domains were not used as hard boundaries. This technique was required to better mimic the orientation of mineralization within the vein. The whole block grade was calculated by weighting the high and low grade estimates on the indicator estimate.

16.1.5 Capping by Risk Adjustment Method-Analysis and Implementation

The uncertainty related to the amount of high-grade metal was evaluated using a Monte Carlo simulation technique developed by Dr. Harry Parker (AMEC E&C). The variation in the amount of high-grade metal present in annual (for Indicated Resources) and global (for Inferred Resources) production increments was calculated. Capping levels were determined for different data spacings because the risk associated with high grade assays increases with decreased data density.

Capping was implemented by reducing the indicator of high grade blocks and re-combining the whole vein grades using a different combination formula than was used for the uncapped grades. The factors applied to the indicator and the recombination equation for whole vein grades was developed iteratively until the targeted metal reduction was approached. The actual metal removed is slightly different than the targeted metal removal. The metal removed is noted below in Table 16.1:

Table 16.1: Metal Removed

Domain	Gold (% metal-at-risk)	Silver (% metal-at-risk)
Indicated (25 X 50 m) or better	5.8	5.7
Remaining Indicated (50 X 50 m)	11.3	12.2
Inferred	3.7	3.3

16.1.6 Model Checks

The resource model was checked as follows:

- Visual inspection of estimation results (sections/plans on screen and sections on paper)
- Comparison of kriged estimates and nearest neighbour (declustered) composite distributions
- Analysis of block model statistics,
- Analysis of grade profiles by northing and elevation, and
- Analysis of change-of-support statistics.

Comparison of the kriged estimates to the nearest neighbour distribution and the grade profiles by northing and elevation were useful in identifying areas of the estimation plan requiring modification.

16.1.7 Resource Classification

Mineral Resources are categorized using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM 2000) definitions. Indicated Mineral Resources are estimated where drill holes or trenches intersect the vein(s) at approximately 50-meter spacing (83 percent of Indicated Resource is supported by 25x50-meter drill hole spacing). Grade appears continuous from hole to hole at this spacing and is further confirmed by closer spaced drilling and channel sampling. Inferred Mineral Resources are estimated down-dip and along strike from Indicated Resources in areas drilled at approximate 100-meter spacing. Projection distances are limited to within 100 m of a drill hole.

16.1.8 Resource Statement

Indicated and Inferred Resources are summarized in Tables 16.2 and 16.3, respectively, by gold grade cutoff. Mineable reserves are included in these tabulations.

Table 16.2: Indicated Resource, All Veins, All Zones

Au (g/t) Cut off	Tonnes (1000)	Au(g/t)	Au (Ounces X 1000)	Ag(g/t)	Ag (Ounces X 1000)
0	7,661	17.52	4,317	222.08	54,702
2	7,351	18.22	4,305	230.52	54,481
4	6,865	19.29	4,258	244.06	53,866
6	6,403	20.33	4,184	257.02	52,911
8	5,673	22.02	4,017	277.86	50,677
10	4,942	23.95	3,806	300.45	47,741
12	4,224	26.16	3,553	326.25	44,311
14	3,616	28.38	3,299	351.50	40,859
16	3,069	30.76	3,035	377.96	37,288

Table 16.3: Inferred Resource, Vein, All Zones

Au (g/t) Cut off	Tonnes (1000)	Au(g/t)	Au (Ounces X 1000)	Ag(g/t)	Ag (Ounces X 1000)
0	7,033	8.52	1,928	120.46	27,240
2	6,049	9.75	1,895	136.47	26,540
4	5,105	10.97	1,801	152.91	25,096
6	4,090	12.45	1,637	171.39	22,539
8	3,093	14.18	1,410	191.18	19,010
10	2,293	15.99	1,178	208.35	15,357
12	1,659	17.90	955	223.05	11,895
14	1,141	20.21	742	238.91	8,768
16	840	22.10	597	250.82	6,773

16.2 Mineral Reserve Estimate

16.2.1 Resource Conversion and Classification

Using metal prices of $400/oz of gold and $6.00/oz of silver, and the detailed mine planning for the open pit and underground mines at Kupol, Bema engineers converted the Indicated Mineral Resources stated in Table 16.2 to Probable Mineral Reserves. There are no reserves classified as Measured in keeping with Bema Gold's policies established at the Julietta Mine. See Section 20.1 of this report for a description of the open pit and underground mine details.

16.3 Mineral Reserve Statement

Table 17.4 summarizes the Kupol Probable Mineral Reserve. The Indicated Mineral Resources in Table 16.2 are inclusive of the Probable Mineral Reserves stated here.

Mr. William J Crowl, P.G. is the Qualified Person as defined by NI 43-101 taking responsibility for the estimation of the Mineral Reserves at Kupol.

Table 16.3: Kupol Probable Mineral Reserves

Production (includes dilution and ore loss)		
Open Pit	Tonnages (millions)	1.42
	Au (g/t)	20.4
	Ag (g/t)	193
Underground	Tonnages (millions)	5.66
	Au (g/t)	16.0
	Ag (g/t)	219
Total	Tonnages (millions)	7.09
	Au (g/t)	16.9
	Ag (g/t)	214

17.0 OTHER RELEVANT DATA AND INFORMATION (Item 20)

None.

18.0 INTERPRETATION AND CONCLUSIONS (Item 21)

The preparation and completion of the Kupol Feasibility Study has been a cooperative effort by a number of firms and individuals over the last 1-2 years. As the project progressed, the data were evaluated to ascertain if they supported project development. As drilling results were interpreted, Bema planned necessary in-fill drilling and further geological investigations. It is Gustavson's opinion that the Mineral Resources and Mineral Reserves as stated by Bema are valid and ready for exploitation under the plans put in place by Bema.

19.0 RECOMMENDATIONS (Item 22)

Based on the results of the Kupol Feasibility Study, the authors recommend the continued development of the Kupol Project.

20.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES (Item 25)

20.1 Mining

The mine planning was completed by Bema with assistance from Wardrop Mining and Minerals. The Probable Mineral Reserve estimate was completed by Bema Gold staff. William Crowl, P.G. is the Qualified Person responsible for the Mineral Reserve.

The open pit will deliver 1.42 million tonnes over a mine life of 4 years at an average grade of 20.4 g/t gold and 193 g/t silver. The underground will deliver 5.7 million tonnes over a mine life of approximately 7 years at a grade of 16.0 g/t gold and 219 g/t silver.

The production schedule has the open pit and underground mines operating at the same time.

20.1.1 Open Pit

The ultimate pit depth in the Big Bend area will reach approximately 100 meters and the pit depth the in majority of the Central and North zone areas will be approximately 45 to 50 meters. From current thermister data it is known the entire pit is within permafrost and therefore, does not include any groundwater influences in the pit slope design. Final bench geometry of 24 meter bench height (70º bench face angle) with a minimum bench width of 10 meters can be successfully and safely mined. The average overall slope angle of the pit will varies depending on geotechnical parameters but averages 50 degrees. The selectivity for the open pit is constrained by the minimum mining width of 2 meters.

The Kupol open pit will be mined as a standard truck/loader operation, with the crusher located at the processing plant, and the waste dump located approximately 2 km to the south at the tailing impoundment. The strip ratio 12:1 is consistent over the life of the pit.

4.3 m^3 loaders will be the main loading units in waste and will be used for loading a high percentage of the ore. The loader units required for the open pit mining effort are presently on site at Kupol and are being used in the construction effort. 4.3m^3 excavators will be used in the ore grade control efforts in the pit. The excavators will clean the waste from the top of the shot vein under the guidance of grade control personnel. Next the excavators will pull the ore from the face of the bench either loading an available truck or placing to the side for later loading into a truck. The excavators will clean the ore until the footwall waste is reached. The loaders and excavators are matched with 35 tonne mining trucks that will be used for both waste and ore haulage. The trucks used for the construction effort will be used for ore and waste haulage from the pit. Based on cycle times for average haul distances to the crusher and waste dump, and on 340 scheduled days per year, this will require a fleet of 9 trucks. Drilling requirements in waste will be handled by one large diesel powered rig equipped with a drill capable of single-pass drilling of 6m benches with an additional .9 m for sub-grade, with a hole size of 140 mm. In addition two ECM diesel powered drills are available for the open it operations. These smaller drills are presently being used at site for the construction effort.

The production schedule for the open pit assumes mining commences in 2006 to provide construction materials for the tailing impoundment and other structures. Ore from the open pit will be mined for four years 2007 through 2010. Table 20.1, Open Pit Production Schedule shows the amount of material mined and the average grades per year from the open pit. The production schedule assumes that the pit operators work 340 days per year on two – 11 hour shifts.

Table 20.1: Open Pit Production Schedule

Year	2007	2008	2009	2010	Total
High Grade (tonnes)	178,628	229,895	268,454	273,222	950,199
Au Grade (g/tonne)	21.2	26.8	36.9	22.7	27.4
Ag Grade (g/tonne)	182	230	352	229	255
Low Grade (tonnes)	117,030	119,304	56,194	182,968	475,496
Au Grade (g/tonne)	6.3	6.3	6.8	6.5	6.5
Ag Grade (g/tonne)	57	64	69	79	69

20.1.2 Underground

The Kupol mineralization is typical of a high grade vein hosted deposit: single and multiple veins; globally continuous mineralized vein with localized discontinuous segments and variable thickness. Veins on surface are extremely well exposed. Over 75% of the veins at the surface is washed clean and mapped extensively. Two approximately 70 meter sections of the vein are drilled on a very tight spacing (10 meters along strike and 5 meters along dip). A 2004 detailed drilling program and detailed surface mapping and sampling quantified local vein geometry. From this information ore loss and dilution criteria are developed for the underground mine.

The initial stope layout is done using an 8 g/t undiluted grade and a 20 gram/meter (grade thickness) criteria. During the 2004 drilling program several thermister were installed. Permafrost exists to at least a depth of 250 meters, well below the depth of the feasibility mine plan. Based on the geometry of the mineralization and the results of the geotechnical studies, longhole stoping is the mining method:

- Sills are driven on 15 meter spacing approximately 4 meters high
- Longhole stopes (panels) are drilled between the sills (approximately 11 meters)
- Stope are filled with waste rock when required
- No sill pillars are left between mining fronts. A concrete sill pillar is constructed on the first (lowest) sill cut of a mining front if there is an expectation ore will be mined up to this sill from below.

Average sill dilution is approximately 27.9% and average dilution for the panels is approximately 23%.

Two major declines driven 5 meters wide by 5.5 meters high exploit the underground reserves, one in the south end of the mine and one in the north. Development of the south decline begins in 2006 and the north decline system begins in 2008. Table 20.2, Underground Production Schedule shows the amount of material mined and the average grades per year.

Table 20.2: Underground Production Schedule

Mining Location S=South, N=North	**Production (tonnes)**	**Au Grade (g/tonnes)**	**Ag Grade (g/tonne)**
2007 (S)	71,879	23.3	290
2008 (S)	234,821	24.6	312
2009 (S & N)	648,173	19.3	190
2010 (S & N)	902,172	16.7	235
2011 (S & N)	1,043,364	16.7	223
2012 (S & N)	1,004,288	15.7	219
2013 (S & N)	1,043,003	12.6	213
2014 (S & N)	713,504	13.2	193

The waste cross-cut access off the main decline will be 5m by 5m to the vein therefore allowing the use of trucks to deliver backfill into the stope. Development in ore will be 4m to 5m high and the full width of the ore. All development will use 2 boom jumbos.

The backfill cycle is an integral part of the production cycle and on an annual basis approximately 1500 tonnes per day of backfill is required to be placed to maintain the underground production schedule. It must be noted the last sill cut and associated panels at the top of a mining front will likely not need to be filled. Backfill will be a combination of run-of-mine waste either directly from underground development (50% of backfill requirements), with open pit waste (preferentially taking acid generating or potentially acid generating material) or waste from a borrow source located on surface (unlikely). The waste for backfill that is obtained from the surface sources may need to be sized to below 0.3m. Waste from the surface sources will be trucked with the open pit mine trucks to a stockpile area near the portal, then reloaded onto the underground mine trucks (40 tonnes) and back-hauled to the stopes requiring backfill. Backfill requirements first exceed waste from underground development in 2010. In 2010 the open pit operation is coming to an end and surface trucks are therefore available

Ore and waste haulage will be accomplished using 40 tonne articulated trucks. Development of declines and access ramps will be completed using 8 yd^3 LHD's.

Appendix B contains is an excerpt from the detailed underground mine design contained in the Feasibility Study.

20.1.3 Waste rock

The total amount of waste produced is approximately 18.6 million tonnes. Mine waste rock will be generated primarily by the open pit (16.9 million tones or 91%). Underground mine development waste totals 1.7 million tones. Up to 3.3 million tones of the total waste may be used to fill underground stoping areas.

The tailing impoundment will be constructed of non acid and potentially acid generating waste. Potentially generating waste rock may be used only in the core of the dam where thermal modeling indicates very fast freezing of this material. In addition the liner on the face of the dam greatly reduces the possibility of any water reaching the potentially acid generating material used in the core.

Acid generating mine waste will be placed in the tailing impoundment basin (upstream of the impervious liner). The mine waste will eventually become covered by tailings or water and freeze very soon after the end of mine life.

In order to characterize the wastes, a program for geochemical characterization of the wastes was prepared. Approximately 276 samples were selected for acid generation potential and other geochemical testing. The full report on the geochemical characterization is presented in Appendix C.2 of the Preliminary Assessment. The results of the geochemical characterization indicated that approximately 42% of the material may be acid generating, approximately 47% of the material may be non-acid generating, and the remaining portion (~11%) is potentially acid generating and must be classified in the field. The non-classified material has been assumed to be non-acid generating for this study.

The assay will be done by relatively fast index sulfur testing such as the Leco S test and tests for neutralization potential. Following testing, blasted waste materials will be flagged to identify their ARD classification, and haul trucks directed to the appropriate dump location in the tailings impoundment. Non acid generating waste will be used, for example, for any general site fill requirements during construction and operation.

20.2 Metallurgy

Metallurgical testwork for the Kupol Project process development was conducted in two phases. Phase I focused on flowsheet development, evaluation and selection, while Phase II was geared toward process/flowsheet optimization and evaluation of ore metallurgical variability. Phase I included testing to evaluate whole ore leach and flotation/leach process options. Phase II included testing to:

1) Evaluate various whole ore leach flowsheet options;
2) Optimize recovery, reagent consumption and process economics; and
3) Assess ore variability and finalize recovery estimates.

Design data for final equipment sizing and selection was also collected in Phase II testing.

The culmination of the Phase I metallurgical studies, was the selection of a whole ore leach flowsheet, with gravity separation, for treatment of Kupol ores. The selection was based on the results of extensive testwork to evaluate both whole ore leach and flotation/leach options, and economic and technical considerations. The whole ore leach flowsheet had significantly lower capital and operating costs in comparison to flotation/leach. Precious metals lab recoveries were similar for both flowsheets at approximately 94-95% for gold and 85-86% for silver.

The completion of the Phase II scope of work has resulted in an optimized feasibility flowsheet with final sizing and selection of key equipment for the whole ore leach process. The Kupol flowsheet will include the following major unit operations:

1) Primary jaw crushing
2) SAG/ball mill grinding
3) Gravity Separation
4) Agitated leaching
5) CCD thickener washing
6) Merrill-Crowe zinc precipitation
7) Smelting of precipitate to dore bars
8) AVR cyanide recovery
9) Hypochlorite cyanide destruction

Some of the more important studies during the Phase II testing included the lab-scale optimization of the leach circuit parameters, the evaluation of cyanide destruction options and the assessment of AVR cyanide recovery. The cyanide concentration for the economic optimum leach conditions was found to be silver grade dependent with higher grade supporting higher cyanide leach concentrations. The economic optimum leach conditions were used to evaluate the metallurgical response of more than 50 ore variability samples comprised of single and multiple hole composites from the core drilling program. Gold recoveries were mostly consistent across the zones in the Kupol deposit, but silver recovery was significantly more variable. Final recovery estimates based on the combined Phase I and II test results are 93.8% for gold and 78.8% for silver.

20.3 Process Description

The Kupol mill is a conventional gold/silver cyanidation plant that will incorporate a CCD thickener washing circuit and Merrill-Crowe zinc precipitation because of the high silver ore grade. There will be a cyanide recovery circuit using AVR (acidification-volatilization-recovery) and cyanide destruction using calcium hypochlorite.

The mill is designed to have a maximum throughput of 3,191 tonnes per day (at 100% availability) at a grind size of 80% passing 53 microns (averages 3,000 tonnes per calendar day at 94% availability). This equates to an annual throughput of 1,095,000 tonnes per year. An overall flowsheet for the process is shown in Figure 20.1, Kupol Mill Process Flow Sheet.

The estimated reagent consumption for the mill is provided in Table 20.3, Estimated Mill Reagent Consumption.

Table 20.3: Estimated Mill Reagent Consumption

Reagent	Consumption (kg/tonne milled)	Consumption [A] (tonne/year)
Sodium cyanide [B]	1.31	1434
Lime	4.18	4580
Flocculant (Ciba 1011)	0.23	252
Lead nitrate	0.26	285
Antiscalant	0.03	33
Sulfuric Acid	2.20	2409
Sodium Hydroxide	2.12	2321
Hydrochloric acid	0.01	11
Calcium Hypochlorite [B]	1.44	1577
Diatomite	0.38	416
	kg/kg doré produced	**tonne/year[C]**
Zinc Dust	1.00	230
Borax	0.26	60
Soda Ash	0.07	16
Sodium Nitrate	0.33	76

[A] Based on an average daily throughput of 3000 tonnes/day

[B] Assumes AVR cyanide recovery

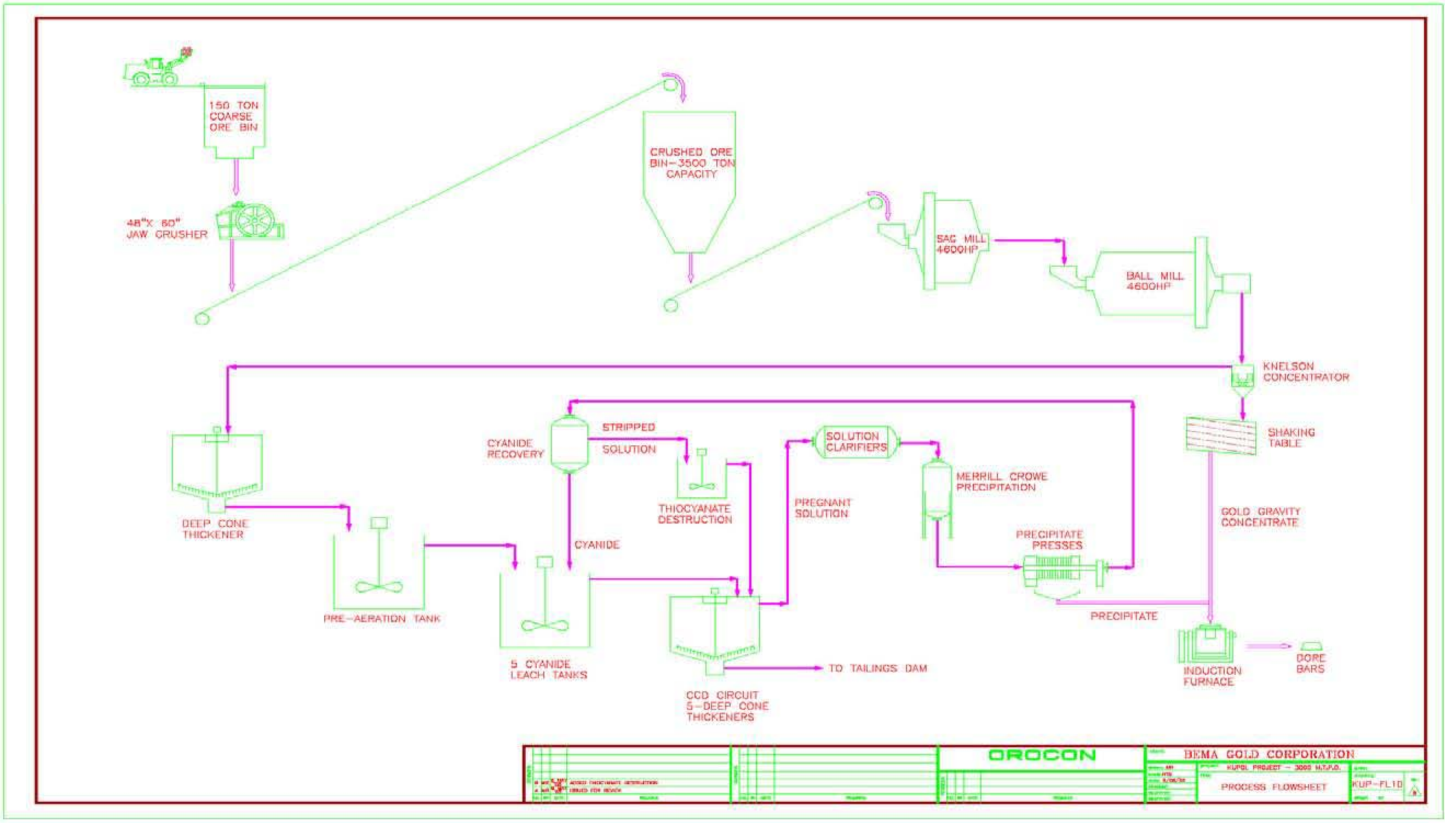
[C] Based on an annual production of 230 tonnes of dore.

The maximum fresh water make-up for the mill processing needs is approximately 137.5 m3/hour. This is based on a maximum dry feed rate of 3191 tonnes per day. Assumptions in the water balance include:

- Ore feed contains 3% moisture;
- Solids specific gravity is 2.65;
- Water retained by the tailings is 30%;
- CCD wash ratio is 3:1; and,
- The percent solids to the dam is 50%.

Process water for the mill water balance can either be sourced from the tailings dam or from process water wells located downstream of the tailings facility.

Figure 20.1: Kupol Mill Process Flow Sheet



20.4 Tailings Facilities

Feasibility design of the tailings disposal facility was completed by AMEC Earth & Environmental (AMEC). The feasibility design is based on a total tailings volume – 12,000,000 tonnes. At this time the indicated mineable reserves total approximately 7 million tonnes.

Feasibility level engineering/design was performed on two tailing impoundment alternatives. Both alternatives are for conventional deposition of tailing material. Investigations into dry stacking showed the tailing material was too difficult to filter and dry stacked tailing disposal was not a viable alternative. The preferred impoundment location alternative is contingent upon the results of condemnation and geotechnical drilling that is being performed in May and June of 2005.

The conventional tailings impoundment will be initially constructed with a starter dam to provide storage for the first 3 years of tailing production, storage of acid generating rock and a portion of the potentially acid generating rock, and the necessary process water. The starter dam will be constructed of non-acid generating and potentially acid generating material from pre-production open pit and underground mine development. It is expected the open pit and underground mines will provide sufficient construction materials, if not, non-acid generating material will be quarried. The dam will be raised in the downstream direction during the first three years of operation with material from the open pit mining activity. Storage capacity will be increased by extending the geomembrane up the slope and also extending the upstream foundation seepage cutoff up the abutments.

The seepage barrier will consist of a bituminous geomembrane liner placed on the upstream face and anchored to bedrock at the toe of the dam. Bituminous membranes are considered to have much greater permanence than plastic liners. Bituminous membranes have high durability at low temperatures, can tolerate a wide range of pH values and can be installed at temperatures down to 25°C.

Tailings, waste rock (acid generating and potentially acid generating) and water management will be carried out using procedures that allow effective operation in cold climates. The primary strategy will be to maintain a water cover over the tailings, manage the impoundment to provide reclaim water during the winter and deposit the acid generating and potentially acid generating material in the impoundment basin such that the waste rock will ultimately be covered by tailings.

To avoid water surplus in the pond, diversions will be installed to divert most of the catchment area upstream of the tailings facility.

20.5 Water Supply

The preliminary assessment for the water supply was completed by AMEC Earth & Environmental (AMEC). It was assumed that the estimated water requirements at the Kupol facilities will be approximately 400,000 to 600,000m^3/yr (1,100 to 1,600 m^3/day) for process water, and 35,000 to 50,000m^3/yr (100 to 140m^3/day) for the potable water. There were several options investigated during this study to include:

- Option 1 - Alluvial deposits in Kaiyemraveyeem Creek downstream (south) of the deposit (chosen);
- Option 2 - Alluvial deposits in Sarichnaya River (north) of the deposit (determined to have insufficient capacity);
- Option 3 - Surface water storage in the tailings basin (will be used for process water needs);
- Option 4 - Water from the shallow lake located north of the mine (determined to have insufficient capacity); and,
- Option 5 - Water supply dam on Kaiyemraveyeem Creek (determined to be uneconomical.

Option 1: Alluvial Deposits in Kaiyemraveyeem Creek Downstream (South) of the Mine

This option has been identified as the preferred option for the feasibility study. Downstream of the site, at a distance of approximately 4 km (and farther south) the Kaiyemraveem Creek flows in a wide valley with relatively steep slopes and a flat bottom. In the valley bottom the creek terraces are very well developed and consist of sand and gravel. In the summer and fall streamflow is present at the ground surface. In places the riverbed is dry, but in these locations the water flows below ground, within the sand and gravel deposits. At a distance of approximately 17 km south of the mine site groundwater discharge has been observed, resulting in extensive naled formation in winter. This naled indicates that groundwater flows in the alluvial deposits all year round, which in turn confirms the existence of talik conditions under the stream bed. At a distance of approximately 5 km from the mine and campsite the gravel deposits are over 10 m thick.

The positive features of this process water option are: 1) proven conditions of groundwater existence in the alluvial deposits, and 2) high extraction rates available due to the permeable nature of alluvial strata. Also, a relative proximity and easy access to the mine site make this option attractive. Downstream (south) of the test location groundwater resources are even more abundant, as the drainage network recharging the aquifer becomes more extensive. The downside of moving the water intake (wells) southward would be a greater distance from site, and therefore an increased length of the water pipeline.

The main disadvantage of this option is the location of groundwater intakes (wells) downstream of the mine site, tailings facility and camp(s), which could potentially contribute to surface and groundwater contamination. There is no confining, low-permeability layer, which would naturally protect the shallow alluvial aquifer; therefore it has to be assumed that groundwater could be process-affected in the future. Because the feasibility study has determined this to be the most feasibility potable and process water source, a water treatment plant will be incorporated into the potable water supply to ensure that the water will always be of drinking water quality.

A long-term test (1 month) was completed in April/May 2005. The 30 day test was conducted a pumping rate of 250 gallons/minute and, over the course of the test, the water level in the well remained constant (approximately 0.15 meters below initial values). The initial calculations for water volume indicate that this well has in excess of 6.4 million cubic meters of water available. CMGC is in the process of installing a permanent well in this location.

20.6 Existing Facilities

At the writing of the Feasibility Study, there exists substantial constructed infrastructure at the Kupol site and other locations such as Pevek, Bilibino and Magadan. The following is not meant to be totally inclusive of all activities and Infrastructure

Pevek

- Truck shop for summer road and winter road operation
- Personnel and offices for logistic management (ship off-loading, container shipments and fuel shipments)
- Living facilities for truckers etc.

Bilibino

- Logistics support (fixed wing and helicopter support)
- Personnel movement to and from Magadan, Kupol, Anadyr/Nome, Alaska and Pevek

Magadan

- Overall management of logistics and personnel movement
- Management of country issues
- Accounting
- Human Resources
- General Administration (IT, translations etc)

Also, other consultants and contractors work with Magadan personnel assisting with shipments from Everett, Washington, and Vladivostok and Vanino, Russia.

Current and 2005 Kupol Site infrastructure and activity includes, but is not limited to:

- Construction camp expanded from 250 person camp to 460 person camp
- Completed road to airstrip
- Successful construction and operation of the winter road from Kupol to Pevek, movement of approximately 650 TEU's and over 10,000 tonnes of fuel
- Construction of 7,200 cubic meters of fuel storage
- Earthworks for the Permanent Camp
- Operation of crushing and screening plant for concrete aggregate
- Erection of batch plant

20.7 Logistics

Logistics are scheduled for the Kupol project using the various offices operated by Chukotka Mining and Geological Company (CMGC) in Magadan, Pevek, Bilibino, and Moscow in the Russian Federation, and Seattle, WA in the U.S. Most supplies will be delivered to Pevek port during the summer period. The Pevek port is located on the East Siberian Sea and is typically accessible from July until mid-September. CMGC operates a staging facility in Pevek and Dyoynoye to store and prepare supplies until the winter road is passable to site. Fuel is stored by the distributor in Pevek in an existing 90,000 m3 facility that is currently under-utilized (approximately 1/3 of capacity currently).

Supplies and fuel will be transported to site using all wheel, all terrain vehicles. A trip to and from the Kupol site in supply trucks takes approximately 3 days. Trip time during the first and the last two weeks of winter road operation are slower due to road conditions. Supplies will also be brought to Kupol on a regular basis by fixed wing aircraft. When the airport is operational AN-12 and AN-38 planes will be used. In general an AN-12 is capable of carrying approximately 17 passengers or 2 tonnes of supplies and an AN-38 is capable of carrying approximately 26 passengers or 2.5 tonnes of cargo. Flights will be used to transport food, doré, and other supplies to and from Magadan. Additionally, if necessary, supplies can be delivered to Keperveyeem and Pevek in IL-76 transport planes during winter months. A helicopter is also available as needed between Keperveyeem, Anadyr, and site.

Russian personnel working at the mine will be scheduled on a three-week-on / three-week-off turnaround. Expatriates working at the mine will be scheduled on a six-week-on / four-week-off turnaround. Personnel will normally be transported to site by fixed wing aircraft. The flight will fly between Pevek, Keperveeyem, and Magadan. At the present time the company has a weekly charter from Nome, Alaska to Keperveyem that is used for personnel and freight. (See Figure 20.2 below)

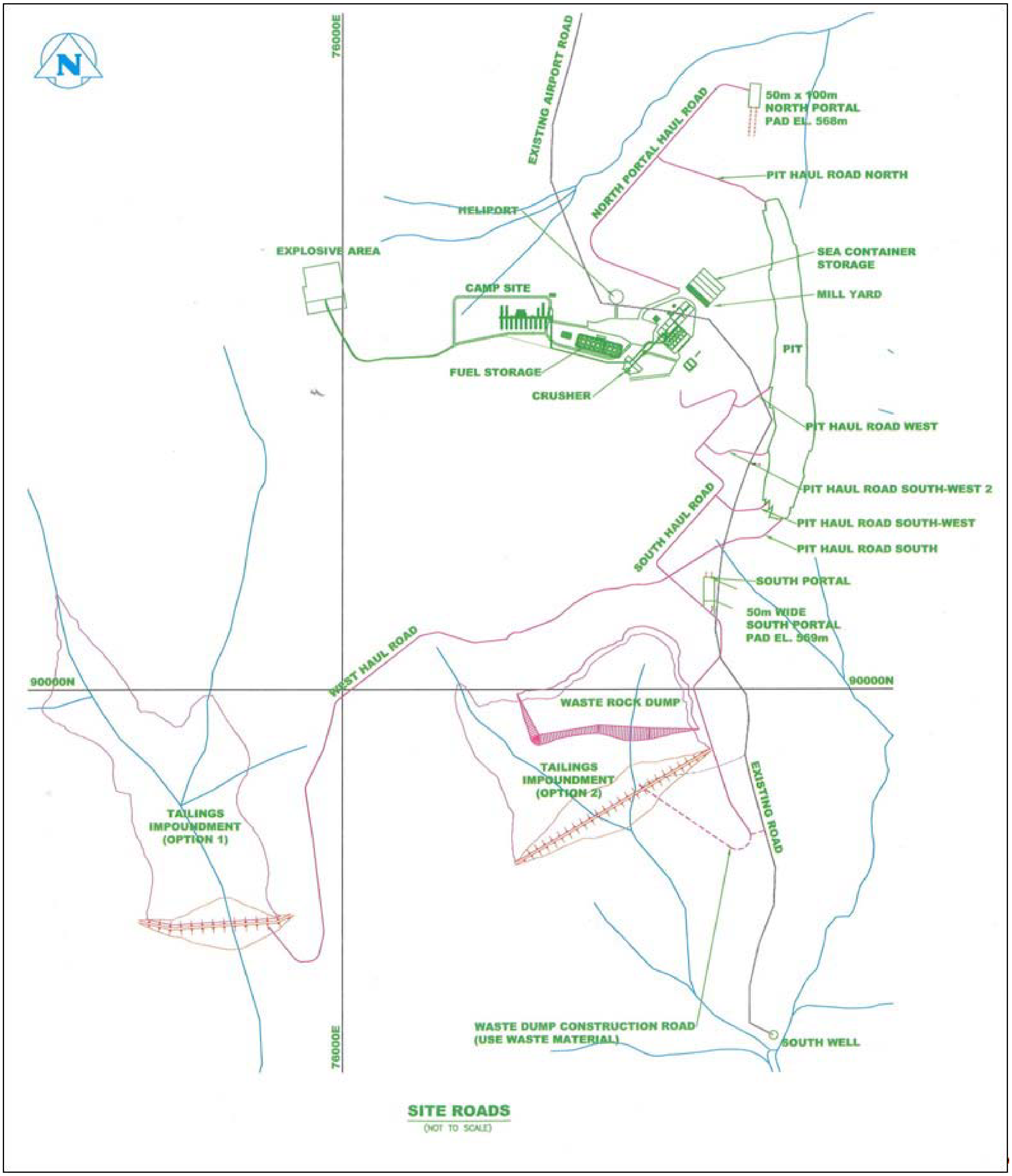
Figure 20.2: Logistics Shipping Routes Map

20.8 Ancillary Facilities

Due to its remote location, the Kupol project must include all ancillary support facilities to include access roads, airport facilities, permanent camp, power generation, fuel storage and distribution, wastewater treatment plant and other necessary facilities.

Figure 20.3 is a general arrangement drawing of the Kupol site, excluding the airport.

Figure 20.3: General Arrangement Sketch of the Kupol Site



20.8.1 Access Roads

The main access road in the winter of 2003 – 2004 was a 765 km route from Pevek through Bilibino to Kupol. During the summer of 2004 a route using an all season road, Pevek to Dvoynoye, then a winter route due south to the Kupol site was investigated. (See Figure 4.2 above)

Bema Gold established management facilities, a staging area, and a truck shop in Pevek in the second half of 2004 and was extremely successful in shipping approximately 650 containers and over 10,000 tonnes of fuel to the site. At the present time the route for the majority of the supplies required at Kupol will be:

- Summer and Fall
    - Ship by sea to Pevek (port open from mid-July to mid-September)
    - Truck non perishable freight to staging yard in Dvoynoye (~530 km)
- Winter (January through April)
    - Truck freight from staging yard in Dvoynoye to Kupol (~125 km)
    - Truck fuel from Pevek to Kupol by traveling due south of Pevek along a winter road (~430 km)
    - Truck any remaining freight in Pevek to Kupol along the winter road (~430 km)

20.8.2 Airport facilities

The airstrip has been designed and construction began in mid-2005. It is being constructed approximately 10 kilometers north of the mine site along a plateau in Starichnaya valley. The airstrip will initially be 1500 meters long (including approaches) by 150 meters wide and is expected to be expanded to accommodate larger planes (AN26, AN12). The airstrip will also have a 1000 m^3 fuel tank for aviation fuel and a 50 m^3 fuel tank for surface vehicles. There will also be a small building for shift change inspection and airstrip support.

20.8.3 Mill and Services Building

The Mill and Services building combines six distinct areas:

- Mill Area
- Power House
- Mine rescue
- Service Complex
- Tank Building

- Truck Shop

These distinct areas are contained in directly adjoining pre-engineered buildings. The Mill and Services Building is being fabricated at this time in North America.

20.8.4 Permanent Camp

The 11,250 m² camp has been designed as a "Permanent Camp". The size was established at 606 persons nominal and can be comfortably expanded to 656 persons. All areas of the camp will be heated using waste heat from the mill central Power House supplemented when necessary by the mill boiler system. The living quarters include VIP units, single occupancy rooms, double occupancy rooms and senior staff quarters. Due to the extreme weather conditions at Kupol great care has been taken to provide adequate recreational facilities for use after work such as games rooms, Gymnasium, and Exercise Room. The Kitchen and dining area has a 3.5 meter ceiling. The camp has a centralized fire alarm system, with the control panel located in the permanently staffed camp security office.

The Permanent Camp is being manufactured in North America at this time.

20.8.5 Power Generation

General

The Kupol power house will operate as an "Island Installation", producing electricity but not connected to an external power grid. The installed generating capacity is approximately 25 MW with an anticipated demand of 15.5 MW. The installed capacity is based upon an N + 2 engine design. Generating voltage will be 4160V, 3 phase, 60 Hz. Plant voltage will be 480V, 3 phase, 60 Hz. All generator units are equipped with jacket water and exhaust heat recovery systems producing a heated water/glycol medium at approximately 91°C. The waste heat system will recover the equivalent of approximately 15MW. The hot medium will be used to heat the mill building complex and the camp facility.

Generating Units

A total of eight generating units will be installed. Four medium speed units rated 4,940 kW at 720 RPM will be the base load engines. It is anticipated three units will be operating at all times, the fourth unit being a spare. Four medium speed Cat 3516 generating units rated 1,450 kW continuous at 1200 RPM will provide power equivalent to a second spare base load engine. One or two of the Cat 3516 generator sets will be used for peak shaving allowing a close match between site operational power requirements and working generating capacity.

20.8.6 Fuel Storage and Distribution

The tank far will hold a combined 30,000 m³ of diesel duel, 800 m³ of aviation fuel and 300 m³of gasoline. All the fuel for the site will be trucked from Pevek over the winter road. The fuel handling facility, located adjacent to the tank farm, will allow the unloading of two fuel trucks simultaneously. Fuel distribution to the power house day tank, camp day tank and fueling station will be through welded, arctic grade steel pipe.

The tank farm is surrounded by a 1.1 m high compacted earth berm with .8 m high internal berms. Berms and enclosed area are lined with a 2 mm thick high density polyethylene liner. Surface water and possible contaminants will be collected in lined sumps and pumped to a water treatment facility for cleaning.

20.8.7 Wastewater treatment plant

The wastewater treatment plant has a treatment capacity for up to 160 m³/day an estimate of .265 m³/day per person for 600 people. The system can also accommodate a temporary increase for 300 people such as summer exploration.

20.9 Legal Matters

The rights to explore and develop the Kupol deposit are currently owned by the Closed Stock Company Chukotka Mining and Geological Company (CMGC). These rights were granted pursuant to the Exploration and Production License (АНД 11305 БЕ, and former License АНД 00746) that was issued by the Ministry of Natural Resources and the Administration of the Chukotka Autonomous Okrug on October 4, 2002. CMGC was initially set up as a wholly owned subsidiary of the Chukotka Unitary Enterprise (a wholly owned government enterprise).

Based on agreement reached between Bema Gold Corporation and Chukotka Unitary Enterprises (Framework Agreement dated December 18, 2002 and amended August 7, 2003), Bema Gold Corporation (through their fully owned subsidiary, Kupol Ventures Limited) can acquire up to 75% minus one share ownership of CMGC. The remaining

25% will remain with Chukotka Unitary Enterprises. Kupol Ventures Limited (KVL) can acquire up to 75% of CMGC in the following stages (see Figure 20.4):

1. Initial 20% - paying $US 8 million in cash (*paid December 2002*) and expending a minimum of $US 5 million in exploration by December 2003 (*completed*);
2. 10% - paying $US 12.5 million in cash by December 2003 (*completed*);
3. 10% - paying $US 10 million in cash (*paid December 2004*) and expending an additional $US 5 million by December 2004 (*completed*);
4. 35% - completing a bankable feasibility study and paying $US 5.00 per ounce for 75% of the gold identified in the proven and probable reserves categories in the feasibility study within 90 days of completing the study.

20.9.1 License For Exploration and Production

The Exploration and Production License (АНД 11305 БЕ) is owned by CMGC. It was registered October 4, 2002 and expires on March 16, 2024. The validity term of the license may be extended by the government of Chukotka, if the license holders provide a substantiated application for an extension of the license terms 6 months before the expiry date.

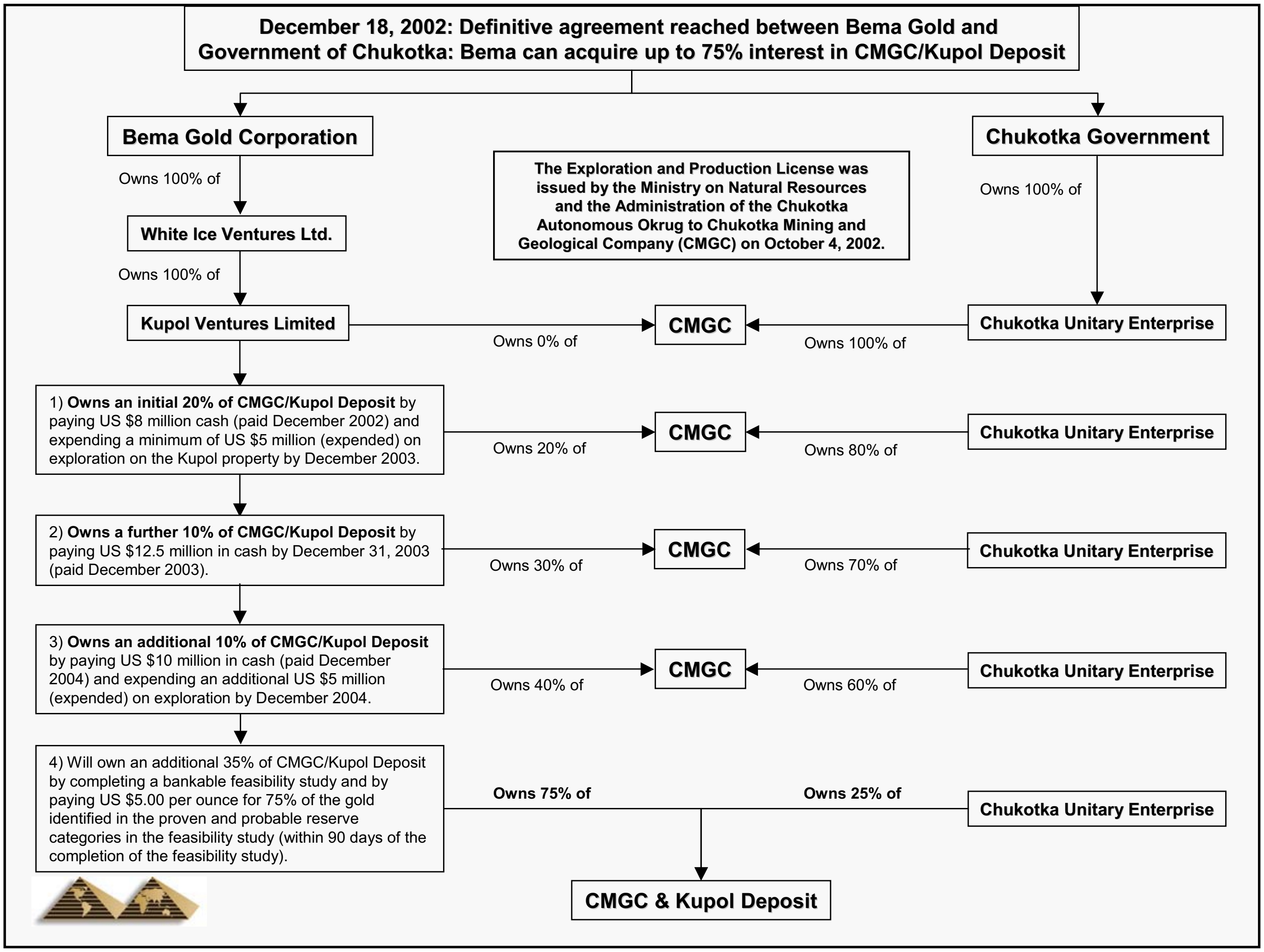
Under the license agreement, CMGC must make regular payments for the right of subsoil use as provided by the existing regulatory and legal instruments to include:

(1) During exploration:

- 1.0% of the costs of the geological and exploration works; and,
- 2.0% of the value of metal mined during the geological exploration works

It is believed that this license condition is no longer valid and has been superceded (2003) by the mineral extraction tax of 6% for gold sales and 6.5% for silver sales. For the purpose of the feasibility study, the 6% and 6.5 % sales tax for gold and silver, respectively have been assumed.

Figure 20.4: Bema Gold Kupol Deposit Acquisition Flow Chart



(2) During operations:

- an amount established during state geological examination (GKZ Expertiza); and,
- when metal losses exceed acceptable norms, double the normal payment rate.

Additionally, Amendment 1 to the license agreement (dated August 7, 2003) requires CMGC to:

- complete exploration works and submit a report with results and calculations of gold and silver C1 & C2 reserves to the State Geological Commission of Experts (GKZ Expertiza) no later than December 31, 2005;
- start development of the deposit, after holding the State Geological Commission of Experts (reserves Expertiza) no later than 2006; and,

- have an annual throughput of no less than 40,000 tonnes per year with a total gold recovery of 85% or more, and a total silver recovery of 70% or more.

20.9.2 Environmental Covenants

Pursuant to the conditions set forth in the license, CMGC must:

- present pre-project and project documents concerning environmental protection measures, calculations of environmental quality requirements, documents authorizing the use of water, information on ecological validation of the allotment, calculations of fees for disposal of wastes, and release of pollutants to the Department for State Ecological Inspection;
- present pre-project and project documentation of the production of minerals, performance of works in water basins and protected areas to the Department and Chukotka District Centre of State Sanitary and Epidemiological Inspectorate for regulatory inspection and approval;
- present to the Department documents required to obtain the license for water use;
- perform construction and reconstruction of production and other facilities on the site of the deposit, subject always to the plans approved by the Department;
- promptly make payments for any acts of environmental pollution under the regulatory and legal acts of the Russian Federation, compensate any damage to hunting, agrarian, fishing and water industries;
- promptly report to the Department any escape of pollutants;
- envisage ways and means of mitigating environmental pollution in the event of a release;
- implement measures of technical and biological reclamation of lands provided to the lessee for temporary use. Indicate the amount of funds necessary for effective reclamation and restoration of the land to a state suitable for further effective economic use;
- construct facilities for mitigating industrial, household, and surface water runoff discharges as well as air pollution mitigation measures;
- present to the Department, together with the feasibility study for the deposit, an Environmental Impact Assessment;
- perform monitoring of water sources, including laboratory monitoring as approved by the Department; and,
- comply with the requirements set forth by the government agencies of the Chukotka Autonomous Okrug concerning solutions for the social and economic problems of the territory to include relations with the reindeer husbandry industry.

It should also be noted that reclamation and closure of the facility is only deemed complete upon the signing of the Act of Liquidation or Abandonment by the Ministry of Natural Resources and Rosgorteknadzor.

20.9.3 Occupational Health and Safety

Pursuant to the conditions set forth in the license, CMGC must:

- maintain all workplaces in accordance with the requirements, norms and standards of occupational health and safety set forth in the Russian Federation; and,
- perform the development of the deposit in accordance with the Uniform Rules of Protection of Subsoil in Hardrock Mining (Moscow, 2003), and standards (norms and rules) for occupational safety and environmental protection.

20.9.4 Taxation

Taxes payable by gold producers in Russia to federal, regional, and local budgets include export duty, profits tax, value added tax, tax for extracting minerals, and other miscellaneous taxes.

- Export duty - Current rates of export duty are 0% of the sales value of gold (reduced on February 14, 2002 from 5%) and 0% of the sales value of silver (reduced on August 15, 2002 from 6.5%).
- Profits tax - On January 1, 2002, the profit tax was reduced from 35% to 24%, of which 5%, 17%, and 2% is payable to federal, regional, and local governments, respectively. Russian legislation provides for certain exemptions from profits tax, which include charter capital contributions, use of funds for capital investments, and transfer of funds free of charge from foreign investors to finance capital investments of production designation, conditional upon their use within one year of receipt.
- Value added tax (VAT) - Current rate of VAT is 18%, which was wholly transferable to the federal budget in the year 2002. VAT is also charged by custom authorities for the import of goods into the territory of Russia. The rates of import VAT are the same as the rates of VAT applicable to the sale of goods in Russia. The most important exemptions from import VAT include the import by foreign investors of technological equipment as charter capital contributions to a Russian company. Import VAT is offset once the goods are accounted for on the books of the recipient. VAT charged on goods used for construction is refundable or can be used as a credit against other federal taxes payable once product produced is sold.
- Tax for extracting minerals – The current rate of tax for extracting minerals is 6% of the sales value for gold and 6.5% of the sales value for silver.
- Miscellaneous taxes - There are also some other regional and local taxes and royalties that include property tax (2.2%) based on the value of fixed assets recorded in the books of the Russian company.

20.9.5 On-going Legal Matters

20.9.5.1 Restricted Right to Sell Gold in Russia

Currently under Russian legislation, the Russian Federal Ministry of Finance (known as *Gokhran* in Russia) and regional authorities have a preferential right to purchase gold and silver bullion bars directly and/or from the producers of precious metals. In the case of sales by producers to the Ministry of Finance and/or regional authorities, such state agencies buy at prices that are based on international market prices. In the case where entities that have a first priority right have not used such right, producers may sell gold and silver bars only to authorized Russian banks.

In addition to selling gold and silver bars on the domestic Russian market, and provided that the above-noted first priority right is waived annually, producers may export their gold and silver bullion bars through Russian-authorized banks and/or directly to offshore purchasers.

20.10 Environment

20.10.1 Baseline Information

The Kupol Property is located within the watershed of the Anui and Anadyr upland regions, in the east foothills of the Anui mountain ridge. This area is characterized by prevailing low, rounded hills with occasional flat, midland areas. The watersheds are flat-shaped or convex-plane, with rounded hilltops elevated from 100 to 200 meters above the riverbeds. The tops are divided by wide (200 to 300 m) but shallow (20 to 30 m) saddles. The absolute elevations of the hills surrounding the property do not exceed 700-1050 m (1034.2 m for Malakhai Mountain and 815.0 m for Kupol Mountain).

Permafrost is distributed throughout the Kupol Property area. Depending on geomorphology, thickness of permafrost layer goes down from surface to 200 to 320 meters and reaches its maximum deep under riverbeds (and may reach 450 meters based on SRK criteria). Thickness of seasonal melting varies from 0.02-1.5 meters in river valley terrains to 12.4 meters on watersheds. Annual temperature range line never goes 20 – 30 meters under surface. Rock temperature within this range can vary from -14.0˚C at the bottom to -5.8˚C at the 2-meter depth. Temperature gradient within permafrost rock is 0.023˚C/m.

In accordance with the closest weather stations, the average annual air temperature at the Kupol site, with only minor variances, is near -13°C. The total amount of precipitation does not exceed 277 mm. The absolute minimum average monthly temperatures occur in January and February (-58°C). During the warmest months (June-August), the average air temperatures are 8.3; 11.3; and 10°C; respectively. Snow cover appears in the mountainous regions in the middle of September and achieves a maximum depth in March. The average depth of snow cover is 38-45 cm.). The duration of a stable snow cover is approximately 237 days. Wind patterns for the region around the Kupol site are defined primarily by the trade winds that are characterized by atmospheric circulation. The average annual wind speed is 2.1-2.6 m/s with a maximum wind speed of 20 m/s. The maximum wind speed recorded at the weather stations is 24 m/s. The maximum wind speed recorded by the weather station installed at site (2003) is 30 m/s.

The territory around the Kupol site belongs to the Watersheds of the Srednyi Kayemraveem and Malyi Anyui Rivers. The Srednyi Kayemraveem drains into the Mechkereva River. The Mechkereva River is a right tributary of the Anadyr River. The Anadyr River is one of the largest waterways on the Chukotka peninsula and the waters flow from the West to the East thru the middle part of Chukotka and drains into the Bering Sea in the Pacific Ocean. The Malyi Anyui River is a right tributary of the Kolyma River. The Kolyma is one of the largest waterways of the Far Northeastern part of Russia and flows form the South to the North to the Eastern Siberian Sea in the Arctic Ocean. The Srednyi Kayemraveem River runs north to south and is situated just east of the deposit. It has an average bed width of 5-10 meters and an average depth of 0.3 meters. The river is primarily fed from surface water runoff (90%) that includes rain, snowmelt, and seasonal thawing of the active permafrost layer. Based on analogous rivers, the maximum amount of flow occurs in June and July. During the spring and summer, the river will experience 97% of its annual flow (3% occurs in the fall months).

Based on geobotanical classification, the Kupol deposit is located in the Anyusk geobotanical district of the sparsely forested area of the western part of the Anyusk-Chukotka foothills. The forest composition of the Kupol deposit is represented by 73 species that are typical for the Omolon and Anyusk geobotanical regions. The area is not populated with any rare or protected species. There is no commercially valuable vegetation located within the site boundaries. The existing vegetation can be used as reindeer pastures. The value as a reindeer pasture is no higher than the multitude of other areas surrounding the site with similar plant densities and speciation.

The area around the Kupol deposit is populated by wildlife that is typical for the cold-weather, topography, and mountainous terrain that surrounds the deposit. Animals that are typically found in western Chukotka include 20 species of mammals, 1 amphibian, and 34 species of nesting birds. The territory can be divided into 3 distinct regions for the mammals: tundra, taiga, and mountains. There are a total of 34 bird species, belonging to 8 orders, within the Srednyi Kayemraveem and Myechyerova River basins. Of these species, the Passeriformes and the Charadriiformes are the dominant species encountered and have the largest populations.

Among rare and endangered species that can be located within the area include:

1. mammals protected under the Russian Red Book for Far Eastern Russia:
    a. Yakutian snow sheep; and,
    b. Wild northern reindeer.
2. birds protected under the Russian Red Book for Far Eastern Russia:
    a. Peregrine Falcon; and,
    b. Gyrfalcon.

The site was surveyed during the 2003 field season by an archeologist. This included walking the site and surrounding areas to look for any identifying artifacts or areas that may have archeological significance. There were 4 areas that were identified for further investigation. None of these areas were located within the construction footprint.

There are no nature preserves or protected environmentally sensitive areas in the vicinity of the Kupol site.

20.10.2 Impacts

Impacts describe the potential effect that a risk source may have on one or more environmental receptors. Receptors can include affected humans (mine personnel, local communities) as well as natural ecosystems. Potential environmental impacts have been assessed by characterizing the natural receiving environment, mine employees and local communities affected by the mine and combining this with theoretical knowledge of typical effects of exposure to the identified risk sources. The potential environmental impacts from the project are listed in Table 20.4, Impact Matrix for the Kupol Project.

Table 20.4: Kupol Project Impact Matrix

Component	Impact Description	Mitigation Measures	Risk
Air quality	Short term, episodic, and localized impacts	Baghouses for dust, regular maintenance, catalytic converters where feasible, watering of fugitive dust sources, minimizing land disturbance	HIGH for stationary sources, MEDIUM for mobile sources, and HIGH for fugitive sources
Topography and land disturbance	Changes in topography, removal of vegetation, reduction in surface water quality, and changes in hydrobiological characteristics	Minimizing land disturbance, interim reclamation, and stockpiling of topsoil	INTERMEDIATE for topography and land disturbance
Soil	Compaction, soil structure loss, potential chemical changes due to chemical composition	Minimize land disturbance, interim reclamation, extensive testing of potential ARD and implementation of waste rock management plan.	Potential impacts from ARD, if not properly managed are HIGH
Surface water quality	Changes in river productivity, increased sedimentation, no long term impacts	Proper erosion control measures, minimization of land disturbances, treatment of domestic wastewater, minimize potential of filtration from tailings impoundment	Potential impacts from erosion and run-off are LOW. Potential impacts, if tailings facility leaks, are HIGH. None of the impacts are considered long term.
Vegetation	Reduction in ground cover, loss of habitat and feeding area for wildlife	Minimize land disturbance, interim reclamation	Potential impacts from vegetation loss are LOW. Impacts are considered to be short term.
Wildlife impacts	Loss of wildlife through poaching and disturbance of natural habitat	Minimize land disturbance and implement strong anti-poaching policies at the mine.	Potential impacts for wildlife loss are LOW. Impacts are considered to be SHORT term.
Aesthetics and visual resources	Loss of aesthetics and visual resource through land disturbance and creation of new land formations	Minimize land disturbance and conduct interim reclamation	Potential impacts are LOW. Impacts for land formation are considered LONG term.
Socioeconomics	Increase revenue for the region through taxation and job creation. Overall positive impacts for the region.	Maximize opportunities for IP and women. Maximize number of employees hired from the region. Maximize potential for local purchases and potential development of small to medium enterprises to support the operations.	Potential positive socioeconomic impacts are considered HIGH and LONG term.
Archeology	Loss of cultural monuments during land disturbance	Conducted survey of area. Create an cultural monument response plan	Potential impacts are LOW. Impacts are considered to be LONG term.

20.11 Project Schedule

The Kupol Project Development Schedule (schedule) is a Critical Path Method (CPM) schedule. The schedule was prepared using input from members of the project management team and the EPC contractor with activity definition, durations, start dates, and logical relationships. The schedule depicts activities, in varying level of detail, from the feasibility study completion in the 2nd quarter of 2005 through the production of dore´ in the 3rd quarter of 2008.

Kupol Project development will occur over the three years following publication of feasibility study with project operations scheduled to occur in June 2008. The Kupol Gold Project Schedule is located in *Appendix B*.

The schedule is partitioned into the following major project areas;

- Major milestones

- Engineering and design
- Transportation
- Site Preparation and Infrastructure
- Concrete and Structural
- Process Facilities
- Utilities and Ancillary facilities
- Tailing system
- Mine development

The Kupol project is unique in that much of the infrastructure and a large portion of the process facilities engineering and procurement is well advanced. Key work completed at the publication of this feasibility study includes:

- Mill procurement and the majority of refurbishment
- Procurement and mobilization of the primary earthworks construction fleet
- Procurement and construction of the temporary man camp supporting infrastructure construction
- Process plant clearing and grubbing and earthwork
- Airport road construction
- General site access construction
- Mobilization of infrastructure construction force
- Procurement and construction of temporary site fuel facilities for over 10,000 cubic meters (8500 tonnes) of diesel fuel
- Setup and operation of a large scale logistical support operation
- Construction and operation of a winter road
- Engineering, procurement, and majority fabrication of the permanent 600 person man camp
- Engineering, procurement and majority fabrication of the primary process building, maintenance facility, office building, dry, and power generation facility
- Selection of the power generation vendor

Succeeding the completion of the Feasibility Study in 2005 is the preparation of an Environmental Impact Assessment for presentation to Russian authorities. The Environmental Impact Statement and the feasibly study are the basis of an application for a construction permit for the main project facilities. Ongoing work is being conducted under temporary approvals and permits.

20.11.1 Critical Activities

The Critical Path of the Kupol Project schedule is driven by the longest related path to the start up of the mill and production of doré.

The major criticality flow generally through;

- Engineering activities;
- long lead procurement of the 4.5 MW generators;
- the availability of the winter road for transportation of all goods, equipment, and consumables;
- construction of the generators;
- Construction of the mills, and the generators

20.12 Execution Plan

The Kupol Project execution plan encompasses project management by Bema Gold Corporation and utilizes a delivery method comprised of a combination of Engineering, Procurement, and Construction (EPC) combined with multiple prime engineering contracts and some self preformed owner construction.

Engineering and Procurement will be managed from the Bema corporate offices in Vancouver, British Columbia, Canada with construction management occurring from the Kupol Site. The majority of procurement is sourced from North American or European suppliers and is ocean shipped to the North Siberian Seaport of Pevek with subsequent overland delivery utilizing dedicated winter roads constructed and maintained by the project.

The site development will take place year round, utilizing a work force of experienced Russian nationals, trained and supervised by Russian and expatriate supervision, many of whom have worked on Bema's previous Russian projects. Personnel will be "fly in-fly out" on a standard rotational basis.

Logistics will be supported from the large existing Bema support structure in place and operating in Russia out of Magadan.

The project anticipates utilizing professionally assisted commissioning in many areas to facilitate mill startup in June of 2008.

20.12.1 Goals and Objectives

The Kupol Gold Project philosophy is to develop the minimum facility required to maximize the recovery of the resource at the highest possible rate of return, in the shortest possible time period without adverse affect to human health, safety, or the environment.

20.12.2 Scope

The project encompasses a combined open pit and underground mine feeding a 1,095,000 tonne per year processing facility. The process will produce gold and silver doré via the Merrill Crowe process following milling, gravity circuit and whole ore leaching. Tailings will be deposited in an onsite tailings impoundment.

20.12.3 Cost

Refer to the Economic Evaluation Section for Project Cost information.

20.12.4 Schedule

The general construction sequence encompasses four (4) construction seasons, as follows; 2005, 2006, and 2007, with final construction occurring in late spring of 2008.

Earthwork and infrastructure commences and completes in 2005 and 2006 with mechanical erection, tailings dam construction, and mine development, commencing in 2006 and 2007 and completing in the 2nd quarter of 2008. Pre commissioning commences in the first quarter of 2008 with commissioning and startup occurring in the last half of 2nd quarter 2008. Doré production is expected in the 2nd quarter of 2008.

20.12.5 Management Approach

The development of the Kupol project will be managed and overseen by an experienced management team comprised of Bema Gold employees and contract personnel. The team reports to the Vice President of Russian Operations in Magadan and the Senior Vice President of Operations in Vancouver. The Management team's primary responsibilities include:

- Manage the overall execution of the project according to the corporate goals, objectives, and project philosophy
- Assure oversight of the engineering and procurement process from the Vancouver office
- Perform review, comment, approval of all engineering and design drawings necessary for project completion
- Assure management and oversight of the EPC contractor in during engineering, procurement, and construction

- Contract and manager for all other engineering, procurement, and technical assistance required to fulfill the project objectives
- Assure management of all Bema self performed work in the field
- Provide logistical support to the EPC contractor and the management team overseeing Bema self performed work
- Prepare internal and external reporting to satisfactorily support Bema management needs, project management needs, lender, regulatory needs, and other needs as defined by the corporate sponsor

20.12.6 Method of Development and Construction

Two major methods of construction will be utilized for development of the Kupol Project, Engineering, Procurement, and Construction (EPC), and multiple prime engineering contracting, supporting Bema self performed construction. The primary process facility will be engineered, procured, and constructed utilizing the engineering, procurement, and construction (EPC) approach. The EPC contract is between Bema Gold and Orocon, Inc., a Vancouver, British Columbia firm experienced in successfully developing process facilities in Russia. The contract is a basic form, cost plus fixed fee contract with the addition of a cost and schedule performance fee.

This EPC contracting approach will be supplemented by engineering and design contracts on a separate prime basis for all other facilities and development work managed by the Bema team. Currently anticipate prime engineering contracts include:

- Surface and mine final engineering and final design
- Final infrastructure design
- Tailings impoundment final design
- Waste dump final design
- Site Wide Water Management and treatment design
- Underground electrical distribution final design.

The Bema management team will manage all contracts and self performed construction of the earthworks, tailings dam, underground development, surface mine development, and winter roads.

20.12.7 Logistics Support

Logistics for the Kupol project will be provided by offices operated by Chukotka Mining and Geological Company (CMGC) in Magadan, Pevek, Bilibino, Anadyr, and Moscow in the Russian Federation, and Seattle, WA in the U.S. Most supplies will be delivered to Pevek port during the summer period. The Pevek port is located on the

East Siberian Sea and is typically accessible from July until mid-September. CMGC operates a staging facility in Pevek and Dyoynoye to store and prepare supplies until the winter road is passable to site.

20.12.8 Battery Limits

Orocon is generally responsible for the engineering, procurement, and construction of the processing facility, power generation, mechanical infrastructure, and surface facilities, including the man camp. The owner is responsible for earthwork, tailings pond construction, site wide water management; surface and underground mine development including underground electrical distribution from the portals.

20.12.9 Risk Management & Insurance

The Kupol management team will implement standard North American risk management practice and procedures during development and implementation of the project. Risk is continually assessed and its impact on project objectives and financial and technical performance will be mitigated via design, construction and project operations procedures.

Insurance to cover construction, marine, and land shipment, storage, general liability is being implemented. Additionally, less traditional insurances are being considered for implementation after risk assessment and cost benefit analysis is completed.

20.12.10 Organization

The Kupol Project Management Team reports to the Vice President of Russian Operations and the Senior Vice President Of Bema Gold Corporation in Vancouver,

The project management team is lead by Corporate Project Sponsor, (Senior Vice President of Operations), and the Vice President of Russian Operations.

20.13 Capital and Operating Cost Estimates

20.13.1 Pre-Production Capital Costs

The Preproduction Capital Cost Estimate for the Kupol Project's processing plant and infrastructure was compiled by the project management team supported by the primary EPC contractor, Orocon Incorporated, and other Feasibility Study engineering contractors that contributed significantly to the capital cost estimate form and basis. Orocon Incorporated provided the plant process and infrastructure costs. Orocon performed the engineering, procurement, construction and management for the process plant and facility at Bema Gold's Julietta operation located in Far East Russia. Wardrop Mining and Engineering provided significant portions of the form and basis of the infrastructure engineering, surface and underground mining capital costs and ancillary facilities and support equipment costs

The project capital cost estimate is far advanced from the usual Feasibility Study Cost estimated due to two factors:

- Bema's long term operating presence in Russia
- The advanced state of Kupol Projects procurement commitment and the advanced state of infrastructure construction

The grinding mills and crushing equipment, the man camp, the primary surface buildings have been secured; partial payment has been made; and they are undergoing fabrication and shipping.

The project total costs include costs expended to date in 2004 and 2005.

The total estimated preproduction capital cost estimate is USD 407 million, which includes working capital for supplies, taxes and owners costs. (Table 20.5)

Table 20.5: Preproduction Capital Costs

Description	Costs
Inventory Costs	
Working Capital (Supplies Inventory)	$ 5,644,999
Plant & Infrastructure (Spares Inventory)	$ 7,763,000
Sub-total	$ 13,407,999
Owners Preproduction Capital Costs	
Feasibility & Design Engineering	$ 4,228,140
Surface Mining Costs	$ 19,280,676
Underground Mining Costs	$ 21,934,653
Processing Costs	$ 2,969,852
Site Services Costs	$ 2,492,872
General & Administrative Costs	$ 6,846,477
Sub-total	$ 57,752,670
Preproduction Capital Costs	
Plant & Infrastructure	$ 158,787,625
Surface Mine Capital Equipment	$ 1,828,600
Site General Capital Equipment	$ 50,000
Owners' Site Construction Capital	$ 3,317,309
Processing Capital Equipment	$ 77,353
Underground Mine Capital Equipment	$ 12,592,848
Owners Site Construction	$ 113,095,630
Sub-total	$ 289,749,365
Total Capital & Inventory Expenses	**$ 360,910,034**
Taxes	$ 4,850,249
VAT	$ 41,528,707
Total	**$ 407,288,988**

20.13.2 Operating Costs

The operating cost estimate was prepared by Bema Gold and engineering contractors supporting the feasibility study and is based on actual or estimated supply costs, actual and estimated logistic costs, engineered productivity / production rates, and equipment operating and maintenance costs from other operating mines and equipment vendors. (Table 20.6)

Table 20.6: Operating Cost Estimate

Cost/Tonne Milled	Activity or Cost Center
$7.52	Surface Mining
$21.69	Underground Mining
$24.54	Processing
$3.23	Site Services
$3.13	General and Administrative
$1.27	Reclamation
$35.50	Taxes
$135.68	**Total Operating Cost**

20.14 Economic Evaluation & Analysis

20.14.1 Metal Prices & Exchange Rates

The metal prices used in the Kupol Feasibility Study economic model are USD $400 per ounce of gold and USD $6.00 per ounce of silver.

The recent historical exchange rate of 30 Rubles (RUR) to 1 United States Dollar (USD) is used in the Feasibility Study which is supported by using The Bank of Canada's average daily RUR to United States Dollar exchange rate from May 31, 2003 to May 31, 2005 of 29.01 RUR to the USD.

20.14.2 Marketing and Refinery Agreement

The milling facility located at the Kupol site will produce approximately 225 tonnes per year (625 kg per day) of doré (gold and silver product in bar form). Doré bars will be transported weekly by security personnel on an AN12 aircraft from the site to Magadan. The doré will then be transported by security in an armored vehicle from the Magadan Airport to the Kolyma Refinery located near Magadan. The refinery then refines the doré into gold and silver bullion bars meeting international standards. Per Russian regulations, the Central Bank of Russia has the first right of refusal to purchase the gold bullion and silver bullion, if they elect not to then the bullion can be exported for sale.

The Feasibility Study uses the current refinery agreement between Kolyma Refinery and Bema's Julietta Mine as the basis for calculating the Net Production Value. The current agreement is the Refinery Contract No. 90/2001, dated August 7th, 2001 with Amendment No. 1, dated November 25, 2002. The Kolyma Refinery has the capacity to refine the gold from Kupol but will need to expand silver refining capability in order to process the large amount of silver (6.3 million ounces per year) from Kupol.

This is not expected to be an issue since the refinery has previously expanded silver refining capability to meet the increased capacity required due to the production form Bema's Julietta Mine. The Kolyma Refinery has several years to engineer and construct the silver refinery expansion prior to receiving doré from Kupol.

20.14.3 Economic Model Basis

The economic model for the Feasibility Study is an Activity Based Costing model. This model identifies the detail activities required for each work process. The resources and appropriate resource drivers were identified for each activity. The resources and resource driver requirements were calculated and actual supply, labor and logistic costs were used or estimated.

This modeling method is appropriate for determining operating costs for the Kupol Project due to its uniqueness within the mining industry. Specific activity costs and equipment costs were modeled and then compared to actual operating costs at similar mining operations, such as Kinross Gold's Kubaka Surface Mine located in Far East Russia, Cameco Corporation's Kumtor Surface Mine located in Kyrgyzstan, Kennecott Corporation's Greens Creek Mine located in Alaska, Bema Gold's Julietta Mine located in Far East Russia, and other mines located in Nevada (United States) and Canada. Equipment operating costs were also obtained from Caterpillar, Hitachi and Ingersoll Rand and Western Mine Engineering indices to provide a comparison and check with reported actual costs. Actual supply costs, processing reagent costs, grinding media, fuel, ocean and land freight costs where obtained from potential or existing suppliers or from actual project experience and used in the economic model.

20.14.4 Analysis

Exhibit 20.1 is a summary of the annual cash flows in the Kupol Economic Model.

20.14.4.1 Payback Period

The preliminary economic analysis indicates the Kupol project has a payback period for the preproduction capital investment and operating costs of less than 2 years before Net Profit Tax.

20.14.4.2 Net Present Value at Varying Discount Rates

The Net Present Value before Net Profit Tax, using a per ounce gold price of $400 and a silver price of $6.00, at a 0% discount rate is $730 million, at 5% discount rate is $430 million and at 8% discount rate is $308 million.

20.14.4.3 Return on Investment

The Discounted Cash Flow Return on Investment is 30.8% before Net Profit Tax when calculating from 2005 forward (Table 20.7).

Table 20.7: Cash Cost Per Gold Ounce

Production Period	Operating Cash Cost	Total Cash Cost	Gold Oz/Year	Silver Oz/Year (Millions)
Full Years 1-2	$41/oz	$83/oz	701,000	6.3
Full Years 1-5	$43/oz	$84/oz	600,000	6.1
Feasibility ~6.5 years	$47/oz	$88/oz	552,000	5.9

20.14.4.4 Sensitivity Analysis

The Kupol project sensitivity analysis was performed on the following parameters with summary returns as indicated in the following table.

The project shows normal sensitivities, but none of the negative modifications cause returns to modify the robust project financial analysis. The following sensitivity graphs were completed using ranges of values for key economic model inputs shown in Table 20.8. The sensitivity analysis results are shown in Figures 20.5 and 20.6.

Table 20.8: Ranges of Values for Sensitivity Analysis

Parameter	Low	Feasibility	High
Gold Price Silver Price	$350/oz $5.50/oz	$400/oz $6.00/oz	$450/oz $6.50/oz
Fuel ~$/Barrel	$30/barrel	$40/barrel	$50/barrel
Ruble:USD Exchange Rate	27:1	30:1	33:1
Russian Labor	-25%	Current	+25%
Logistics Cost	$242/Tonne	$323/Tonne	$404/Tonne

Figure 20.5: Preproduction Capital Cost Sensitivity

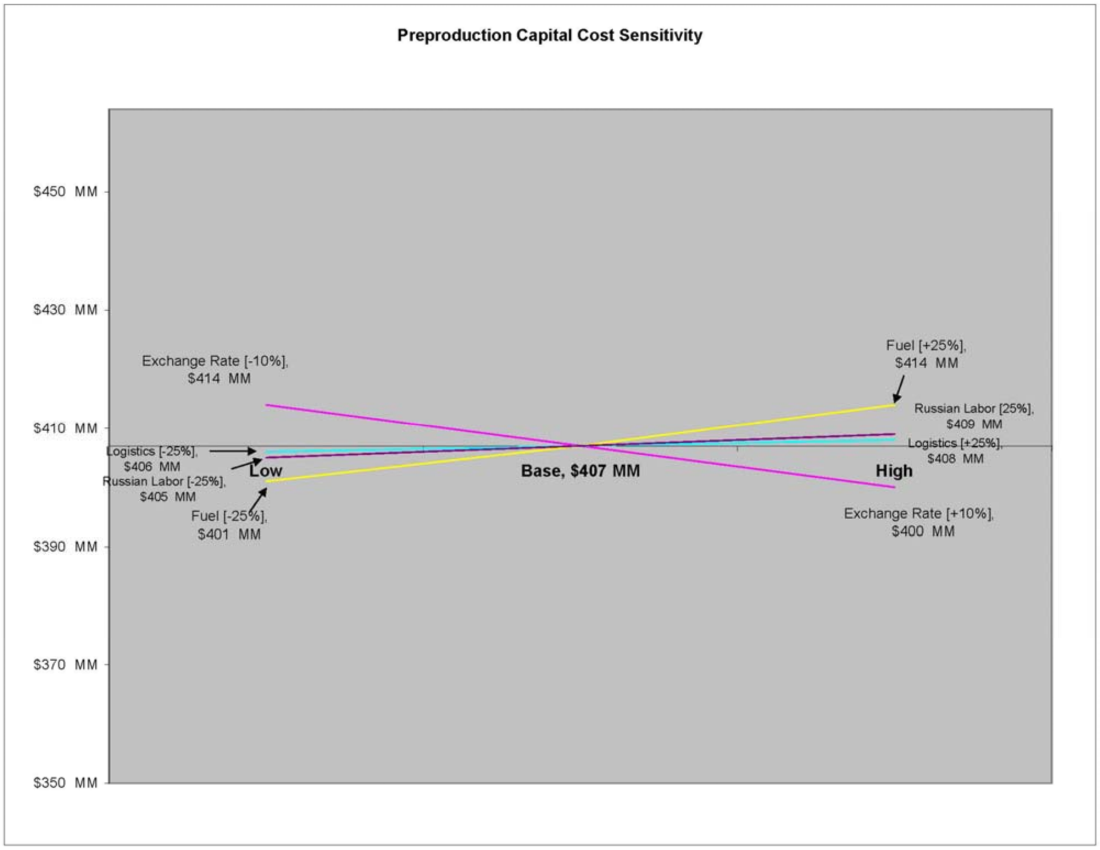


Figure 20.6: Cumulative Cash Flow Sensitivity

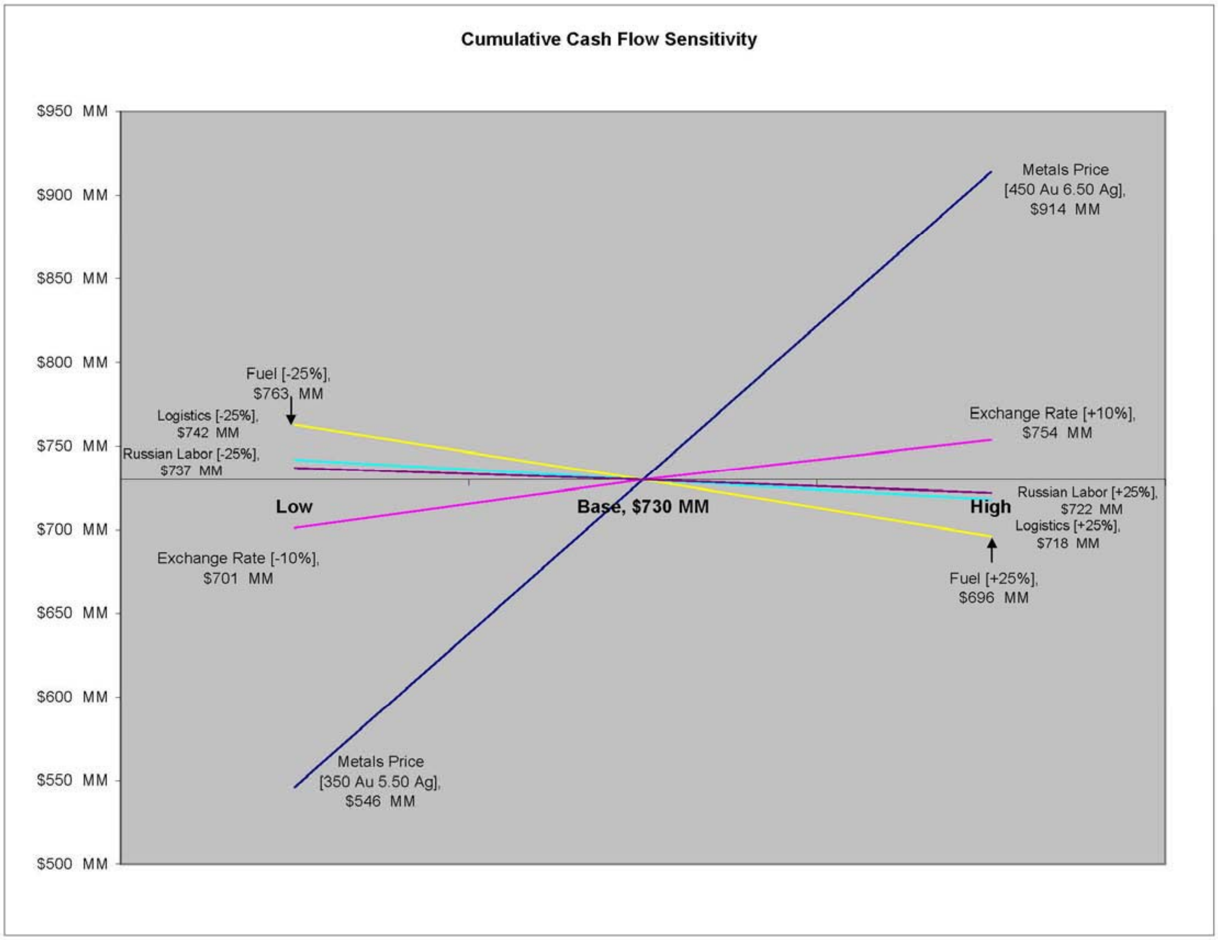


Exhibit 20.1: Project Proforma Cash Flow Statement

EXHIBIT 20.1
Revised:
1-June-05

KUPOL PROJECT
CJSC-CMGC - Chukotka Mining and Geological Company
FEASIBILITY PRE-PRODUCTION & OPERATIONS ECONOMIC MODEL
PROJECT PROFORMA STATEMENT - 100% *[EXCLUDES SEPARATION OF PREPRODUCTION CAPITAL AND OPERATING COSTS]*

	ACTUAL	FORECAST + BUDGET-R7											
	YR 2004	YR 2005	YR 2006	YR 2007	YR 2008	YR 2009	YR 2010	YR 2011	YR 2012	YR 2013	YR 2014	YR 2015	TOTAL PROJECT
PRODUCTION VALUE													
Gross Production Value	$0	$0	$0	$0	$170,265,087	$345,793,262	$288,041,586	$257,469,816	$236,375,797	$200,985,930	$177,964,949	$0	$1,676,896,426
Less: Refining, Shipping, Fees	0	0	0	0	(2,734,616)	(5,481,792)	(4,823,663)	(4,339,114)	(4,017,083)	(3,556,511)	(3,030,513)	0	($27,983,291)
Net Production Value	$0	$0	$0	$0	$167,530,472	$340,311,470	$283,217,922	$253,130,702	$232,358,714	$197,429,419	$174,934,436	$0	$1,648,913,135
COSTS													
Design Engineering	$0	$1,363,327	$1,012,671	$960,691	$891,451	$0	$0	$0	$0	$0	$0	$0	$ 4,228,140
Owners Site Construction	$24,873,736	$45,414,966	$24,638,290	$11,322,142	$6,846,495	$0	$0	$0	$0	$0	$0	$0	$ 113,095,630
Orocon Site Construction	$13,343,750	$49,670,629	$71,739,130	$19,387,540	$12,409,576	$0	$0	$0	$0	$0	$0	$0	$ 166,550,625
Surface Mining Costs	$0	$0	$2,179,464	$11,901,188	$12,480,057	$12,415,648	$11,323,394	$623,735	$694,870	$685,077	$984,887	$0	$ 53,288,320
Underground Mining Costs	$0	$0	$4,117,873	$13,069,403	$11,393,704	$18,052,907	$21,322,582	$21,810,126	$21,091,590	$23,570,622	$19,319,131	$0	$ 153,747,940
Processing Costs	$0	$0	$0	$281,434	$16,130,509	$28,506,532	$28,521,210	$24,988,037	$25,048,407	$25,162,958	$25,243,318	$0	$ 173,882,406
Site Services Costs	$0	$0	$734,112	$822,186	$2,247,777	$3,188,873	$3,163,170	$3,192,131	$3,169,122	$3,167,502	$3,239,597	$0	$ 22,924,469
General & Administrative Costs	$0	$333,500	$2,478,951	$2,932,380	$2,643,950	$3,079,973	$2,186,320	$2,160,224	$2,138,681	$2,134,363	$2,121,638	$0	$ 22,209,979
Change in Ore Stockpile				($799,062)	($78,294)	$84,763	($649,170)	$0	$189,958	$157,396	$1,094,407	$0	$0
Total Costs	$38,217,486	$96,782,422	$106,900,491	$59,877,902	$64,965,226	$65,328,696	$65,867,506	$52,774,253	$52,332,628	$54,877,919	$52,002,980	$0	$ 709,927,508
OTHER COSTS													
Taxes	$6,352,029	$15,886,005	$11,175,911	$9,419,022	$26,364,524	$41,172,447	$35,698,857	$31,438,593	$27,809,372	$24,439,562	$20,638,804	$1,195,962	$ 251,591,087
TOTAL OPERATING COSTS	$44,569,515	$112,668,427	$118,076,402	$69,296,924	$91,329,750	$106,501,142	$101,566,363	$84,212,846	$80,142,000	$79,317,481	$72,641,784	$1,195,962	$ 961,518,596
OPERATING INCOME / (LOSS)	**($44,569,515)**	**($112,668,427)**	**($118,076,402)**	**($69,296,924)**	**$76,200,722**	**$233,810,327**	**$181,651,559**	**$168,917,856**	**$152,216,714**	**$118,111,938**	**$102,292,652**	**($1,195,962)**	**$687,394,539**
NON-OPERATING COSTS													
Reclamation	$0	$0	$0	$0	$0	$100,000	$100,000	$100,000	$250,000	$250,000	$100,000	$8,100,000	$9,000,000
INCOME / (LOSS) Excluding Depreciation	($44,569,515)	($112,668,427)	($118,076,402)	($69,296,924)	$76,200,722	$233,710,327	$181,551,559	$168,817,856	$151,966,714	$117,861,938	$102,192,652	($9,295,962)	$678,394,539
Net Profit Tax	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
NET INCOME / (LOSS) Excluding Depreciation	($44,569,515)	($112,668,427)	($118,076,402)	($69,296,924)	$76,200,722	$233,710,327	$181,551,559	$168,817,856	$151,966,714	$117,861,938	$102,192,652	($9,295,962)	$678,394,539
CAPITAL & INVENTORY EXPENSES													
Change in Working Capital (supplies, logistics)	$0	$0	$391,665	$5,253,334	$4,119,733	($2,692,088)	($3,707,440)	($2,013,412)	$872,351	($3,990,173)	($18,889,949)	$0	$ (20,655,978)
Underground Mining Costs (Primary & Secondary Development)	$0	$0	$0	$0	$6,446,454	$7,918,013	$7,114,987	$11,367,431	$4,879,918	$290,775	$0	$0	$ 38,017,580
Capital Expenses	$0	$1,508,536	$9,067,137	$6,364,503	$9,259,334	$4,216,050	$330,000	$150,000	$150,000	$150,000	$0	$0	$31,195,561
Total Capital & Inventory Expenses	$0	$1,508,536	$9,458,802	$11,617,837	$19,825,521	$9,441,975	$3,737,547	$9,504,019	$5,902,269	($3,549,397)	($18,889,949)	$0	$ 48,557,162
VAT Tax Refundable	$0	$0	$0	$0	($47,624,653)	($10,438,997)	($10,308,868)	($8,972,663)	($7,882,948)	($7,673,839)	($7,025,772)	($0)	$ (99,927,740)
ANNUAL CASH FLOW	**($44,569,515)**	**($114,176,964)**	**($127,535,204)**	**($80,115,699)**	**$104,078,147**	**$234,622,586**	**$188,772,049**	**$168,286,501**	**$153,757,435**	**$128,927,778**	**$127,013,965**	**($9,295,962)**	
PRE-PRODUCTION CUMULATIVE CASH FLOW	($44,569,515)	($158,746,479)	($286,281,683)	($367,995,505)	($407,288,988)	$0	$0	$0	$0	$0	$0	$0	($407,288,988)
TOTAL PROJECT CUMULATIVE CASH FLOW	($44,569,515)	($158,746,479)	($286,281,683)	($366,397,382)	($262,319,235)	($27,696,649)	$161,075,400	$329,361,901	$483,119,335	$612,047,114	$739,061,079	$729,765,117	$729,765,117
Gold Price	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	USD 400.00 /t-oz	
Silver Price	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	USD 6.00 /t-oz	
Operating Cash Costs per Ounce from mid-2008 thru 2014													
Operating Cash Cost per Gold ounce Produced (less silver					$35	$42	$49	$37	$43	$59	$76		$47
Total Cash Cost per Gold ounce Produced (less silver credits)					$75	$82	$90	$77	$82	$99	$112		$88
Project Cash Costs per Ounce from 2004 thru 2014													
Operating Cash Cost per Gold ounce Produced (less silver					$128	$42	$49	$37	$43	$59	$76		$140
Total Cash Cost per Gold ounce Produced (less silver credits)					$173	$82	$90	$77	$82	$99	$112		$182
DCF - ROI [from 2005 thru 2014]													30.8%
NPV [% Discount] 0.00%													$729,765,117
NPV [% Discount] 5.00%													$429,946,500
NPV [% Discount] 8.00%													$307,885,166
PRODUCTION STATISTICS													TOTAL
Gold Produced					376,394 t-oz	767,814 t-oz	619,349 t-oz	551,520 t-oz	504,021 t-oz	417,564 t-oz	379,371 t-oz		3,616,032 t-oz
Silver Produced					3,284,551 t-oz	6,444,629 t-oz	6,717,017 t-oz	6,143,642 t-oz	5,794,562 t-oz	5,660,065 t-oz	4,369,449 t-oz		38,413,915 t-oz
Tonnes Ore Milled	0 t	0 t	0 t	0 t	536,850 t	1,095,000 t	1,095,000 t	1,095,000 t	1,095,000 t	1,095,000 t	1,075,049 t	0 t	7,086,898 t
Gold Grade	0.00 g/t	0.00 g/t			24.97 g/t	24.16 g/t	18.43 g/t	16.54 g/t	15.10 g/t	12.34 g/t	10.91 g/t		16.92 g/t
Silver Grade	0.0 g/t	0.0 g/t			254.9 g/t	237.8 g/t	241.1 g/t	218.8 g/t	208.0 g/t	205.9 g/t	151.4 g/t		214.0 g/t
Tonnes Surface Ore Mined	0 t	0 t	0 t	295,658 t	349,199 t	324,648 t	456,190 t	0 t	0 t	0 t	0 t	0 t	1,425,695 t
Gold Grade				15.33 g /t	19.83 g /t	31.71 g /t	16.20 g /t						20.44 g/t
Silver Grade				132.8 g/t	173.2 g/t	303.3 g/t	169.2 g/t						193.2 g/t
Tonnes Underground Ore Mined	0 t	0 t	0 t	71,879 t	234,821 t	648,173 t	902,172 t	1,043,364 t	1,004,288 t	1,043,003 t	713,504 t	0 t	5,661,203 t
Gold Grade				23.34 g /t	24.57 g /t	19.35 g /t	16.68 g /t	16.69 g /t	15.68 g /t	12.63 g /t	13.16 g /t		16.03 g/t
Silver Grade				290.3 g/t	312.0 g/t	189.8 g/t	235.1 g/t	222.9 g/t	218.7 g/t	212.7 g/t	193.3 g/t		219.3 g/t

COST PER TONNE MILLED

	YR 2004	YR 2005	YR 2006	YR 2007	YR 2008	YR 2009	YR 2010	YR 2011	YR 2012	YR 2013	YR 2014	YR 2015	TOTAL
PRODUCTION VALUE													
Gross Production Value					$317.16 /t	$315.79 /t	$263.05 /t	$235.13 /t	$215.87 /t	$183.55 /t	$165.54 /t		$236.62 /t
Less: Refining, Shipping, Fees					($5.09)	($5.01)	($4.41)	($3.96)	($3.67)	($3.25)	($2.82)		($3.95)
Net Production Value					$312.06 /t	$310.79 /t	$258.65 /t	$231.17 /t	$212.20 /t	$180.30 /t	$162.72 /t		$232.67 /t
COST OF PRODUCTION													
Surface Mining Costs					$23.25 /t	$11.34 /t	$10.34 /t	$0.57 /t	$0.63 /t	$0.63 /t	$0.92 /t		$7.52 /t
Underground Mining Costs					$21.22 /t	$16.49 /t	$19.47 /t	$19.92 /t	$19.26 /t	$21.53 /t	$17.97 /t		$21.69 /t
Processing Costs					$30.05 /t	$26.03 /t	$26.05 /t	$22.82 /t	$22.88 /t	$22.98 /t	$23.48 /t		$24.54 /t
Site Services Costs					$4.19 /t	$2.91 /t	$2.89 /t	$2.92 /t	$2.89 /t	$2.89 /t	$3.01 /t		$3.23 /t
General & Administrative Costs					$4.92 /t	$2.81 /t	$2.00 /t	$1.97 /t	$1.95 /t	$1.95 /t	$1.97 /t		$3.13 /t
Change in Ore Stockpile					($0.15)	$0.08 /t	($0.59)	$0.00 /t	$0.17 /t	$0.14 /t	$1.02 /t		$0.00 /t
Total Cost of Production					$121.01 /t	$59.66 /t	$60.15 /t	$48.20 /t	$47.79 /t	$50.12 /t	$48.37 /t		$100.17 /t
OTHER COSTS													
Taxes					$49.11 /t	$37.60 /t	$32.60 /t	$28.71 /t	$25.40 /t	$22.32 /t	$19.20 /t		$35.50 /t
TOTAL OPERATING COSTS					$170.12 /t	$97.26 /t	$92.75 /t	$76.91 /t	$73.19 /t	$72.44 /t	$67.57 /t		$135.68 /t
OPERATING INCOME / (LOSS)					$141.94 /t	$213.53 /t	$165.89 /t	$154.26 /t	$139.01 /t	$107.86 /t	$95.15 /t		$97.00 /t
NON-OPERATING COSTS					$0.00 /t	$0.00 /t	$0.00 /t	$0.00 /t	$0.00 /t	$0.00 /t	$0.00 /t		$0.00 /t
Reclamation					$0.00 /t	$0.09 /t	$0.09 /t	$0.09 /t	$0.23 /t	$0.23 /t	$0.09 /t		$1.27 /t
INCOME / (LOSS) Excluding Depreciation					$141.94 /t	$213.43 /t	$165.80 /t	$154.17 /t	$138.78 /t	$107.64 /t	$95.06 /t		$95.73 /t
PRODUCTION STATISTICS													
Gold Produced					376,394 t-oz	767,814 t-oz	619,349 t-oz	551,520 t-oz	504,021 t-oz	417,564 t-oz	379,371 t-oz		3,616,032 t-oz
Silver Produced					3,284,551 t-oz	6,444,629 t-oz	6,717,017 t-oz	6,143,642 t-oz	5,794,562 t-oz	5,660,065 t-oz	4,369,449 t-oz		38,413,915 t-oz
Tonnes Ore Milled	0 t	0 t	0 t	0 t	536,850 t	1,095,000 t	1,095,000 t	1,095,000 t	1,095,000 t	1,095,000 t	1,075,049 t	0 t	7,086,898 t
Gold Grade	0.00 g/t	0.00 g/t			24.97 g/t	24.16 g/t	18.43 g/t	16.54 g/t	15.10 g/t	12.34 g/t	10.91 g/t		16.92 g/t
Silver Grade	0.0 g/t	0.0 g/t			254.9 g/t	237.8 g/t	241.1 g/t	218.8 g/t	208.0 g/t	205.9 g/t	151.4 g/t		214.0 g/t
Tonnes Surface Ore Mined	0 t	0 t	0 t	295,658 t	349,199 t	324,648 t	456,190 t	0 t	0 t	0 t	0 t	0 t	1,425,695 t
Gold Grade				15.33 g /t	19.83 g /t	31.71 g /t	16.20 g /t						20.44 g/t
Silver Grade				132.8 g/t	173.2 g/t	303.3 g/t	169.2 g/t						193.2 g/t
Tonnes Underground Ore Mined	0 t	0 t	0 t	71,879 t	234,821 t	648,173 t	902,172 t	1,043,364 t	1,004,288 t	1,043,003 t	713,504 t	0 t	5,661,203 t
Gold Grade				23.34 g /t	24.57 g /t	19.35 g /t	16.68 g /t	16.69 g /t	15.68 g /t	12.63 g /t	13.16 g /t		16.03 g/t
Silver Grade				290.3 g/t	312.0 g/t	189.8 g/t	235.1 g/t	222.9 g/t	218.7 g/t	212.7 g/t	193.3 g/t		219.3 g/t

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The Bema Gold Kupol Feasibility Study summarized herein is based on numerous technical reports, memos and drawings authored by individuals and firms identified in Table 1.1 of this Report.

Appendix A

Certificates and Consent Letters of Qualified Persons

CERTIFICATE AND CONSENT

To Accompany the Technical Report Summarizing the Kupol Project Feasibility Study, Chukhotka Okrug, Russia

I, William J. Crowl, residing at 8036 S Ammons Street, Littleton, Colorado 80128-5539, USA, do hereby certify that:

1) I am a Vice President with the firm of Gustavson Associates, LLC ("Gustavson") with an office at Suite D, 5757 Central Ave, Boulder, Colorado 80301, USA.
2) I am a graduate of the University of Southern California with a Bachelor of Arts in Earth Science (1968), and an MSc. in Economic Geology from the University of Arizona in 1979, and have practiced my profession continuously since 1973;
3) I am a registered Professional Geologist in the State of Oregon (G573) and am a member in good standing of the Australasian Institute of Mining and Metallurgy.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI-43-101.
5) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the assets of Bema Gold Corporation.
6) I am independent of the issuer as defined in Section 1.5 of NI 43-101.
7) Gustavson was retained by Bema Gold Corporation to prepare a Technical Report Summarizing the Kupol Project Feasibility Study, Chukhotka Okrug, Russia (the "Report").
8) I was the co-author of the Report along with Mr. Fred W. Stahlbush, a Qualified Person and Engineering Manger, Bema Gold Corporation. I am responsible for the assembly and preparation of the Report and the Qualified Person for the estimation and classification of the Mineral Reserves as presented in this report. Mr. Stahlbush, as the Kupol Project Feasibility Study Manager was the Qualified Person responsible for the preparation of the Kupol Feasibility Study. Mr Tom Garagan, P.Geo, Bema Gold Corporation's Vice President o Exploration, is the Qualified Person for the estimation and classification of the Kupol Project Mineral Resources as presented in this Report and in a Technical Report dated 31 March 2005.
9) I am not aware of any material fact or material change with respect to the subject matter of the Report which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
10) I have had no prior involvement with the Kupol Project. In the past I have worked as a consultant to Bema Gold Corporation on other mining projects.
11) I have read NI 43-101 and Form 43-101F1 and this Report has been prepared in compliance with this Instrument and Form 43-101F1.
12) I visited the subject property in May 15 - 18,2005.
13) I hereby consent to use of this Report and our name in the preparation of any documents for submission to any Provincial regulatory authority.

"SEAL"
"William J. Crowl"

Boulder, Colorado, USA
July 4, 2005

William J. Crowl
Vice President, Mining

GUSTAVSON ASSOCIATES
GEOLOGISTS · ENGINEERS · APPRAISERS

CONSENT OF QUALIFIED PERSON

British Columbia Securities Commission

Alberta Securities Commission

Saskatchewan Securities Commission

Manitoba Securities Commission

Ontario Securities Commission

Autorité des marchés financiers

New Brunswick Office of the Administrator

Office of the Attorney General
Registrar of Securities, Prince Edward Island

Nova Scotia Securities Commission

Securities Commission of Newfoundland

The undersigned hereby:

1. states that the undersigned is one of the authors of the report entitled "*Technical Report, Summarizing the Kupol Project Feasibility Study*" dated July 4, 2005 (the "Report") prepared on behalf of Bema Gold Corporation (the "Issuer"), portions of which are summarized in the press release dated June 3, 2005 (the "Press Release"), and the material change report dated June 23, 2005 (the "Material Change Report"), of the Issuer;

2. consents to the references to and summary of the Report in the Press Release and Material Change Report;

3. certifies that the undersigned has read the disclosure in the Press Release and Material Change Report and has no reason to believe that there are any misrepresentations in the information contained therein that are derived from the Report or that the disclosure in the Press Release or Material Change Report contains any misrepresentation of the information contained in the Report; and

4. consents to the filing of the Report in the public files of the securities commissions in each of the provinces of Canada.

Dated: July 4, 2005

Signed, Sealed and delivered by:
"William J Crowl"

5757 Central Ave Suite D Boulder, CO 80301 USA +1 (303) 443-2209 FAX +1 (303) 443-3156
http://www.gustavson.com gustavson@gustvson.com

CERTIFICATE OF AUTHOR

Fred Stahlbush, Mining Engineering Manager, Kupol Feasibility Study
Bema Gold Corporation
Suite 3100, Three Bentall Centre, 595 Burrard Street, P.O. Box 49143
Vancouver, British Columbia, Canada
Telephone: (604) 681-8371
Fax: (604) 681-1242

Email: fstahlbush@bemagold.com

1. I, Fred Stahlbush, do hereby certify that I am Mining Engineering Manager, Kupol Feasibility Study for Bema Gold Corporation and reside at 3783 Westhills Place, Bellingham, WA 98226 USA.

2. I graduated with a Bachelor of Science degree in Mining Engineering from the Colorado School of Mines in 1980.

3. I am a member of the Australasian Institute of Mining and Metallurgy registered as member #223407.

4. I have worked as a mining engineer for a total of 25 years since 1980. I have been involved in the mining industry in Canada, USA, South America, Mexico and Russia.

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

6. I was the co-author of the report along with William J. Crowl, a Qualified Person and consulting geologist for Bema Gold Corporation titled "Technical Report Summarizing the Kupol Project Feasibility Study, Chukotka Okrug, Russia dated July 4, 2005 (the "Technical Report") . Additionally, William J. Crowl was responsible for the assembly and preparation of the Technical Report and the Qualified Person for the estimation and classification of the Mineral Reserves as presented in the Technical Report. Further, I confirm that as the Kupol Project Feasibility Study Manager, I was the Qualified Person for the preparation of the Kupol Feasibility Study. Mr. Thomas Garagan, P. Geo. Bema Gold Corporation's Vice President of Exploration, is the Qualified Person for the estimation and classification of the Kupol Project Mineral Resources as presented in this Technical Report and in a Technical Report dated March 31, 2005.

7. I visited the property on July 1, 2004 to July 7, 2004 and for 6 days in October ,2004.

8. I have read NI 43-101 and certify that the Technical Report has been prepared in compliance with NI-43-101 and Form 43-101F1.

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

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

11. I am not independent of the issuer. Per section 5.3.2 of National Instrument 43-101 an independent qualified person was not required for the writing of the Technical Report on the Kupol Property.

Dated at Vancouver, this 4th day of July, 2005.

BEMA GOLD CORPORATION

Per:

(Signed, sealed and delivered by:
"Fred Stahlbush")

BEMA GOLD CORPORATION
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CONSENT OF QUALIFIED PERSON

British Columbia Securities Commission

Alberta Securities Commission

Saskatchewan Securities Commission

Manitoba Securities Commission

Ontario Securities Commission

Autorité des marchés financiers

New Brunswick Office of the Administrator

Office of the Attorney General
Registrar of Securities, Prince Edward Island

Nova Scotia Securities Commission

Securities Commission of Newfoundland

The undersigned hereby:

1. states that the undersigned is one of the authors of the report entitled "*Technical Report, Summarizing the Kupol Project Feasibility Study*" dated July 4, 2005 (the "Report") prepared on behalf of Bema Gold Corporation (the "Issuer"), portions of which are summarized in the press release dated June 3, 2005 (the "Press Release"), and the material change report dated June 23, 2005 (the "Material Change Report"), of the Issuer;

2. consents to the references to and summary of the Report in the Press Release and Material Change Report;

3. certifies that the undersigned has read the disclosure in the Press Release and Material Change Report and has no reason to believe that there are any misrepresentations in the information contained therein that are derived from the Report or that the disclosure in the Press Release or Material Change Report contains any misrepresentation of the information contained in the Report; and

4. consents to the filing of the Report in the public files of the securities commissions in each of the provinces of Canada.

Dated: July 4, 2005

Signed, Sealed and delivered by:
"FRED STAHLBUSH"

CERTIFICATE OF AUTHOR

Tom Garagan, P. Geo, Vice President, Exploration
Bema Gold Corporation
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Vancouver, British Columbia, Canada
Telephone: (604) 681-8371
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Email: tgaragan@bemagold.com

1. I, Tom Garagan, P. Geo, do hereby certify that I am Vice President of Exploration for Bema Gold Corporation, and reside at 1672 beach grove delta B.C.

2. I graduated with a Bachelor of Science (Honours) degree in Geological Sciences from the University of Ottawa in 1980.

3. I am a member of the Association of Professional Geoscientists and Engineers of British Columbia, Association of Professional Engineers, Geologists and Geophyseists of Alberta and a fellow of the Geological Association of Canada.

4. I have worked as a geologist for a total of 25 years since by graduation from the university. I have been involved in gold exploration and mining in Canada, USA, Russia, South Africa, Ethiopia, Chile, Argentina, Venezuela and Mexico.

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

6. William J. Crowl , a Qualified Person and consulting geologist for Bema Gold Corporation along with Mr. Fred Stahlbush, a Qualified Person and Engineering Manager for Bema Gold Corporation were the co-authors of the report titled "Technical Report Summarizing the Kupol Project Feasibility Study, Chukotka, Okrug, Russia dated July 4, 2005 (the "Technical Report"). Additionally, William J. Crowl was responsible for the assembly and preparation of the Technical Report and the Qualified Person for the estimation and classification of Mineral Reserves as presented in the Technical Report. Mr. Fred Stahlbush, Kupol Project Feasibility Study Manager was the Qualified Person for the preparation of the Kupol Feasibility Study. Further, I confirm that I am the Qualified Person for the estimation and classification of the Kupol Project Mineral Resources as presented in this Technical Report and in a Technical Report dated March 31, 2005.

7. I visited the property in excess of 15 times over 4 years.

8. I have read NI 43-101 and certify that the Technical Report has been prepared in compliance with NI-43-101 and Form 43-101F1.

9. I am Vice President of Exploration for Bema Gold Corporation and visited the project several times prior to acquisition.

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

11. I am not independent of the issuer. Per section 5.3.2 of National Instrument 43-101 an independent qualified person was not required for the writing of the Technical Report on the Kupol Property.

Dated at Vancouver, this 4th day of July, 2005.

BEMA GOLD CORPORATION

Per:

(Signed, sealed and delivered by:
"Tom Garagan, VP Exploration")

g:bemakupolcertificate of QP.tomg

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4. consents to the filing of the Report in the public files of the securities commissions in each of the provinces of Canada.

Dated: July 4, 2005

(Signed, Sealed and delivered by :
« TOM GARAGAN, VP Exploration »)

Appendix B

Details of Underground Mine Design

6.1 Underground Mine Design

6.1.1 Mine Design Assumptions

Mining takes place from the bottom of the open pit downward and outward to the maximum extent of economic Indicated resources. Twin declines from the two portal locations, South and North (Figure 6.1 below); provide access to the orebodies approximately 150 meters on the footwall side of the vein zone. Spiral ramps (olive color) branch off of the ramps at strategic locations to provide vertical development of each orebody. Big Bend has one racetrack-shaped spiral, the South, Central, and North each have one spiral, and the North Upper has two. The spiral standoff is approximately 60 m from the orebodies which allows positioning and room for attack ramps that cross to the veins. The attack ramps (red color in Figure 6.1 below) split into an upper and lower ramp which enters the vein one exactly over the other with a vertical spacing of 15 m. The following sections discuss the development details, constraints, and stoping required to meet an underground productive capacity of as much as 3000 tpd.

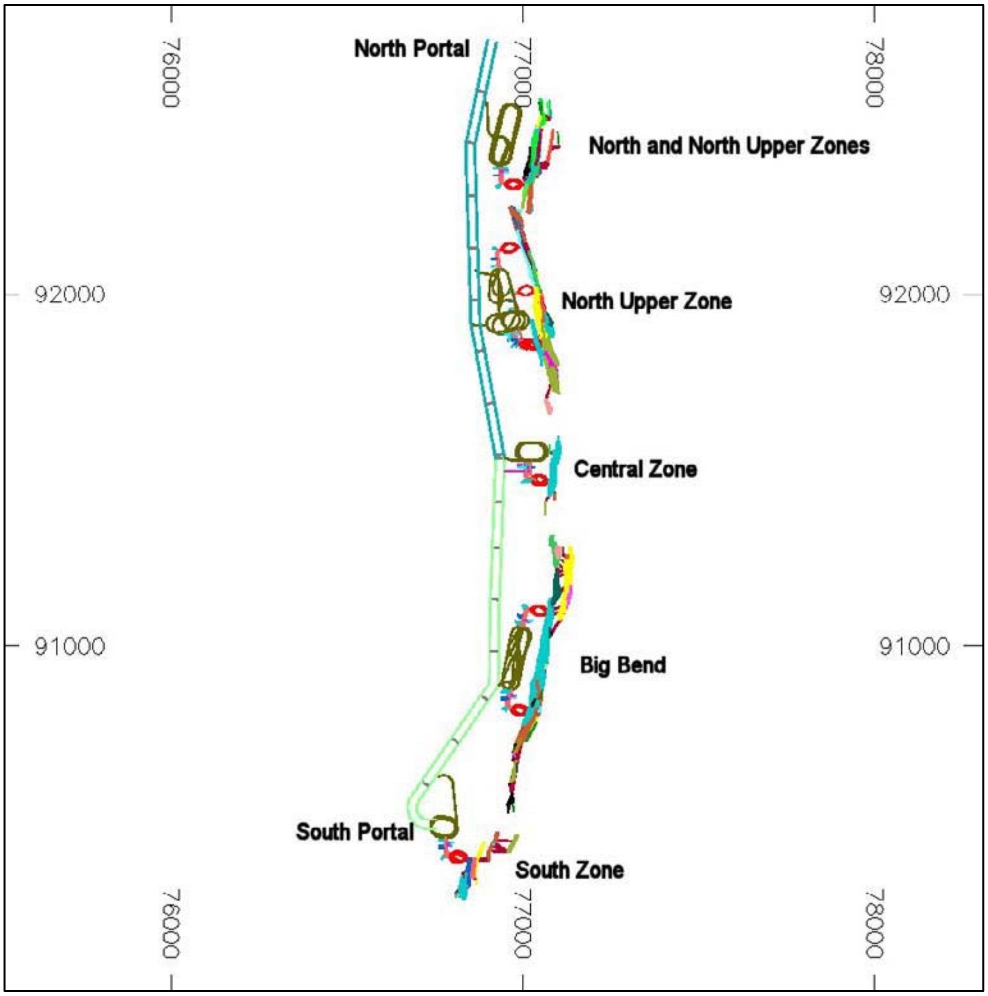
Underground mine planning uses the resource block model, re-blocked and composited across the strike of the vein. The orebody names in the mine plan generally follow the geologic resource, but the limits are modified based on mining access and continuity. The first screen to identify mining region is a review of the block model using an 8 g/t cutoff with a 20 gm-m filter. Portions of the Indicated resource that do not pass the filters are excluded from the mine design.

Tight ore control will be available to guide subdrift and panel markups. Sublevel control will include detailed geologic mapping and face sampling. Geologists will compile individual maps to area posting sheets at various scales, and will use the information to update cross-sections and long sections. Assay and geologic information will be compiled in an electronic database. Geologists will track and control dilution and ore loss based on this information and subdrift/stope surveys. Panel layouts will be based on the underground mapping information, supplemented by surface and definition drill holes on approximately 12.5 X 15 m centers.

6.1.2 Minimum Mining Width

Minimum sub-level size is determined by the mobile equipment dimensions, and a safe clearance. The minimum sub-level cross section is 3.5 m wide x 4 m high for a Toro 006 LHD. A shanty-shaped back will be required in minimum width sublevels to allow room for ventilation tubing. The Toro 1400D will require a minimum sub-level cross section width of 4.7 m. The Solo 07 - 10c longhole drill requires between 0.5 – 1.2 m of clearance beyond the vein contact, depending on the dip of the vein (Figure 6.6 below). The minimum width of longhole stopes is 1.5 m. Waste pillars between veins that are left in place will be a minimum of 3 m wide.

Figure 6.1. Kupol Underground Mine Plan and Access.



6.1.3 Dilution

Three-dimensional wireframe models of the sublevels incorporate designed dilution. Sub-level dilution is established by equipment size, vein width, dip, and complexity. The dilution varies between 0.75 m and 1.0 m per rib. *Appendix L* details the logic and calculation of each dilution component. Minimum mining width dilution is a major component, driven by the mucking and longhole drilling equipment.

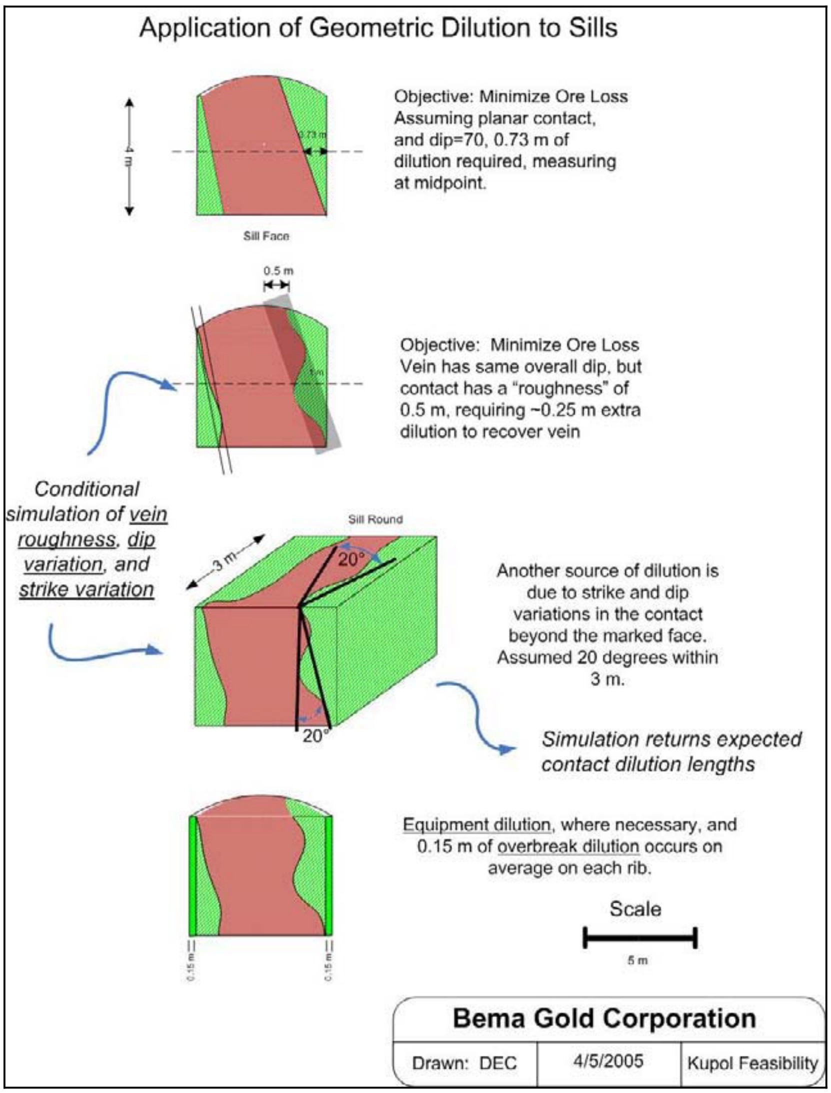
Geometric dilution, defined as dilution that occurs as mining attempts to follow the vein contacts within the drift profile (Figure 6.2), locally exceeds equipment dilution requirements. The Kupol veins dip between 65 and 90 degrees, depending on the area. Contacts are sinuous along the dip and along strike, typical of epithermal veins; small-displacement faults and cross-cutting dikes contribute to apparent sinuosity. Table 6.1 below summarizes sill dilution applied to the Kupol resource model:

Table 6.1 Sill Dilution Applied to Kupol Underground Resource.

General Region	Mine Coordinates	Dilution per Rib
South	90,000 to 90880	1.0 m
Big Bend	90,880 to 91,270	0.90 m
Central	91,270 to 91940	1.0 m
North	91,940 to 92,600+	0.75 m

Table 6.1 above quantities include 0.15 m of overbreak dilution per rib. Dilution per rib equals the greater of equipment or geometric dilution, plus the overbreak amount. Veins narrower than 1.5 - 2.0 m carry dilution exceeding that in the table above because of LHD clearances. These compose approximately 5% of the mineable resource blocks.

Figure 6.2. Schematic of Sill Dilution



Sill dilution is applied by expanding the vein wireframe a fixed distance outward in plan from each contact (Table 6.1 above), resulting in variable dilution percentage. Dilution tons are the volume of dilution material inside 3-D sill wireframes multiplied by a bulk density factor of 2.48. Dilution grade is the weighted average of stockwork resource blocks and any unclassified material inside the sill wireframes. Zero grade is attributed to unclassified material.

Longhole dilution is determined by drawing the planned stope outline in Mine 2-4d scheduling software and adding a fixed dilution percentage by area, discussed below. The dilution assumptions are:

- Geometric dilution between the panel limits depends on the sill spacing, minimum panel width (1.5 m), vein sinuosity and displacements between the sills, investigated with conditional simulations that use close-spaced diamond drill holes;
- The study assumes that geology will provide a sill back markup of the vein contacts and a topsill vein contact projection based on sublevel geologic mapping; and,
- The definition drilling program will apply a dilution constraint by providing ore position and grade data between the sills on a nominal grid of 15 m (vertical) X 12.5 m (along strike). This information will permit modifications to the production drilling pattern; e.g., tightening it around pinch-outs, or drilling blind holes where warranted to recover bulges in the vein.

Panel dilution grade is assigned 1 g/T Au and 13 g/T Ag based on the global average of stockwork-vein contacts. The geometric component of dilution is a fixed percentage, based on similar simulation results in two separate areas, one with relatively narrow veins, and the other with relatively thick veins.

Resource adjustments for combined sill and panel dilution average 22%, applied as percentages listed Table 6.2 below:

Table 6.2 Underground Dilution Detail[1].

Factor	Sills	Panels
Interburden[2]	4%	4%
Design, including overbreak	21%	13%
Other Planned(Re-handling, Backfill, Misclassification)	0%	4%
Subtotal	25%	21%
Total (Tons-Weighted)	**22%**	

1) Dilution = Dilution Tonnes/(Dilution Tonnes + Vein Tonnes)

2) Sill interburden is a minimum based on panel amount.

Sill dilution is calculated from the 3-D wireframes that have dilution applied as a fixed length, described above. Panel dilution in Table 6.2 above is calculated by applying fixed percentages of dilution to the *in situ* vein within the panel. Total dilution is weighted by the tonnages of panel and sill in the mine plan.

6.1.4 Ore Loss

The mine plan applies an ore loss factor to the stope panels. The sources of ore loss are:

1. Geometric ore loss at vein contacts of 4% in panels; and
2. Other panel losses of 2% due to miscellaneous operating and workmanship issues, including scheduling, misclassification, poor ground conditions, losses to backfill, and access problems.

The conditional simulations described in Section 6.1.3 above determine the geometric ore loss component. Panel ore loss is thus 6% (1-(0.96 x 0.98)) and sill ore loss is 0%. The ore loss components, applied to sill and panel tonnages yields a weighted average ore loss of 4%.

6.1.5 Trade-Off Studies for Portal Location and Number of Portals

Trade-off studies included the following items:

- Access by single ramp versus ventilation/secondary escapeway(s) provided by vertical opening (bored raises or Alimak raise); and
- Ramping initiated close to surface camp versus ramping for rapid access to the ore zones.

The studies resulted in selection of the twin ramp system with separate portals closer to the ore zones. Advantages of using a twin ramp versus single ramp arrangement are:

- The twin headings provide all the required, intake, exhaust and second egress to commence production when the access development intersects the ore zones;
- Better traffic control of haulage, service, supervision and logistics equipment;
- If required, dedicated up and down ramp, therefore no traffic delays;
- Superior ventilation flexibility;
- Elimination of “hard” ventilation line;
- More efficient development with multiple faces than with single face, always carrying fresh airbase with the primary development, and eliminating a need for “ventilation to catch-up with the development activity”;
- Flexibility to bypass any ramp section(s) for equipment breakdown or ramp maintenance; and,
- Development flexibility with multiple faces allows more consistent advance during the shift.

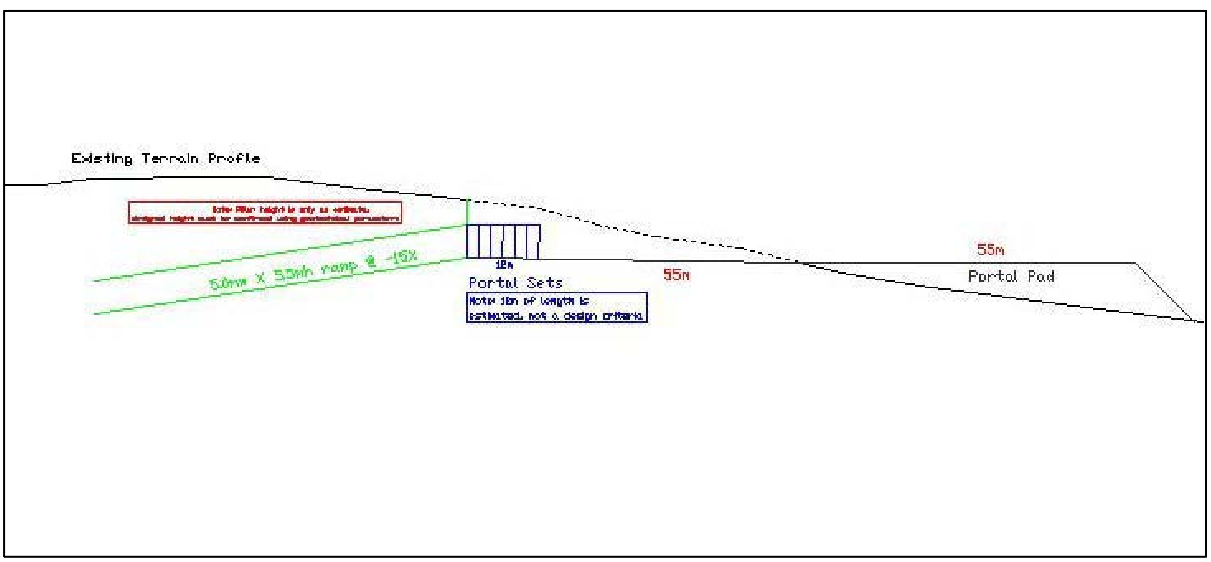
The principal disadvantage of the twin ramp system is the larger amount of waste development required. Mining from two portals instead of a single, centrally located one requires that underground mine equipment and personnel have to travel significant distances over surface roads in order to reach the portals. Planning began with a single portal concept, but this was abandoned in order to bring the underground mine up to its maximum sustained production rate earlier. The approximately 3 km strikelength of the Kupol orebodies requires two separate developments to achieve the production before exhausting the open pit.

6.1.6 Access

Access development will be a two-portal arrangement – one North Portal, and one South Portal. These portals are located to minimize the development distances to the higher grade ore zones particularly the Big Bend area in the South. The portals are remote from the buildings, requiring surface road connections.

The portal locations are checked against the contours and the closest exploration drill holes, but the specific ground conditions in the selected portal areas are not yet evaluated in terms of the expected ground conditions, depth of overburden, and ground support design. Pilot holes will be drilled along the line of each ramp to establish the conditions to be encountered and determine the detailed ramp and portal ground support design (Figure 6.3 below). A higher level of support will be required for the portal sections that are subject to freeze-thaw. In the case of Kupol, the ultimate depth of the pilot holes should correspond to an elevation where the permafrost conditions prevail. This may typically extend +/- 200m in from the portal because of the ventilation system and seasonal freeze-thaw.

Figure 6.3. General Portal Cross-Section.



Both the South Portal ramp and the North Portal ramp will be twin headings 5.5 m-high by 5 m-wide driven at -15% grade. The twin headings will be 20 m apart with connecting crosscuts every 150 m. Steel square sets lagged with timber will be used to form the portal protection and limit the impact of blowing snow and ice in the portal collar area.

Surface roads have to be kept cleared and sanded to allow the underground equipment to travel on them. The surface roads will be lined with cable barriers, markers and high intensity, coloured lights to allow use in white-out conditions.

6.1.7 Ventilation

The main exhaust fan, twin headings, and a connection through the last crosscut provide the circuit for the primary ventilation. A secondary ventilation fan and flexible ducting is required for each face. The twin headings and short secondary ventilation duct length of 150 m allow the ventilation system to easily meet the regulations. A spare primary fan installed in the intake portal with an independent power supply provides backup required by regulation.

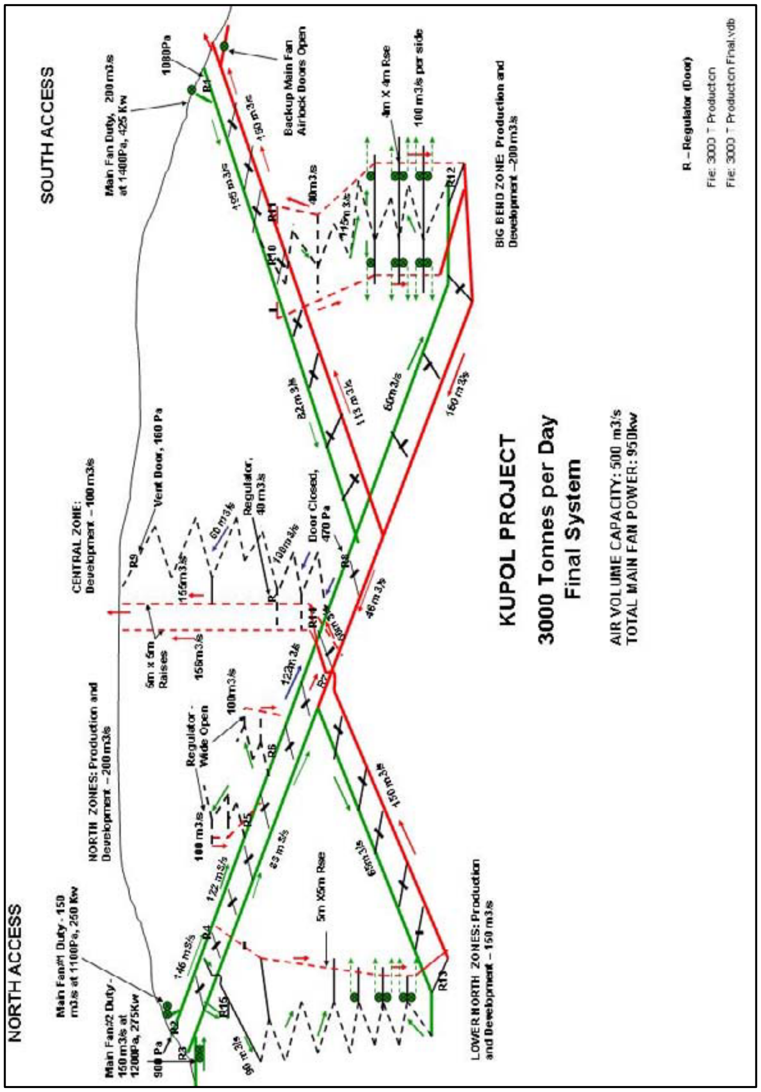
At the initial production stage, the regulated maximum air velocity of 8 m/s permitted in the ramps limits the total intake and exhaust. The maximum contaminants regulation limits the amount of equipment that can be used with this volume of airflow. The contaminants in the exhaust must be kept below the limit to allow use of both headings. The mine will be equipped with central blasting, so that blasting can be undertaken between shifts and at lunch time when no men are in the mine.

The portal ventilation doors, if required, will be removed after access development intersects the Central Zone, and when ventilation raise(s) and secondary egress are developed to surface. All four portals will then be downcast, intake air will be utilized at each working Zone and then exhausted at the Central Zone via a single ramp/portal and two ventilation raises.

The ventilation system for the Kupol Project, described in detail in Appendix M, permits a 3000 tonne per day production rate. Twin main declines are driven from the South and North Portal locations to the ore zones (Figure 6.4 below). These openings are the primary intake and return air routes. An additional exhaust route through the Central Zone is also required to augment the total air volume requirements. All main mine fans are on surface at the portal entrances or on top of main return air raises. Backup main fans are required for all installations.

An air volume capacity equal to 500 m^3/s is the final system requirement for all options. An air volume to diesel equipment ratio of 2.7 cubic meters per minute per brake horsepower (2.7 m^3/min/bhp, or 0.06 m^3/s/kW) determines the overall air volume requirements, and in the design of the auxiliary ventilation systems. Russian regulators at the Julietta Mine are using a ventilation factor of 2.5 m^3/min./bhp, or 0.056 m^3/s/kW.

Figure 6.4. Ventilation Schematic Section.



Control of air flow in all ramps is by the use of ventilation regulator doors because regulations prohibit underground booster fans without special variance. Special consideration to their design must be undertaken to ensure workability at high pressure.

The main intake fan operating pressures and volumes will vary throughout the life of the operations. The highest air volume and fan kW requirements occur at the full production stages. The total connected load for the main fans is 950 kW.

6.1.8 Development and Stoping

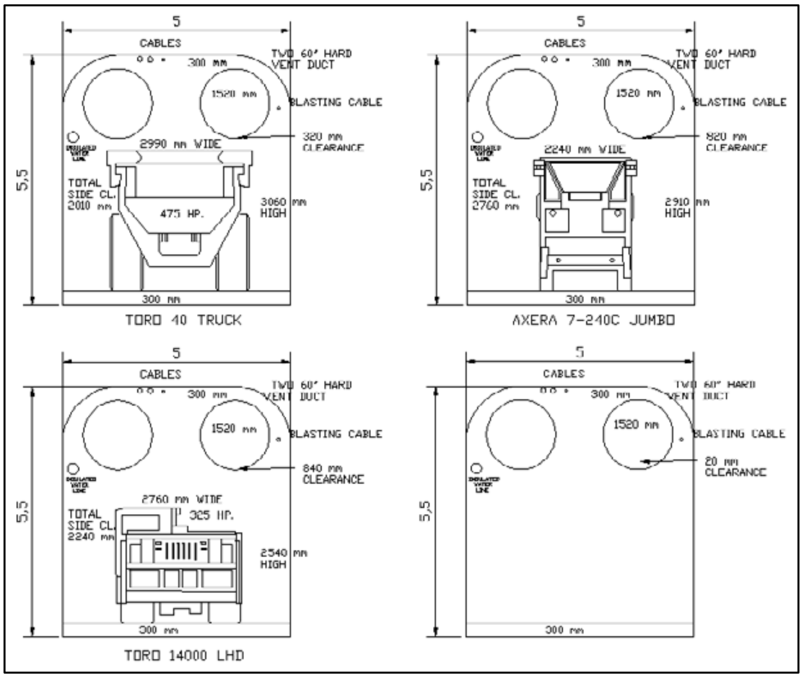
6.1.8.1 Development Headings

Three types of development heading are planned at Kupol Mine:

- Ramps for 40 tonne trucks;
- Level access and sub level sills; and
- Drop/inverse rising.

Figure 6.5 below shows typical development cross-sections with services and equipment.

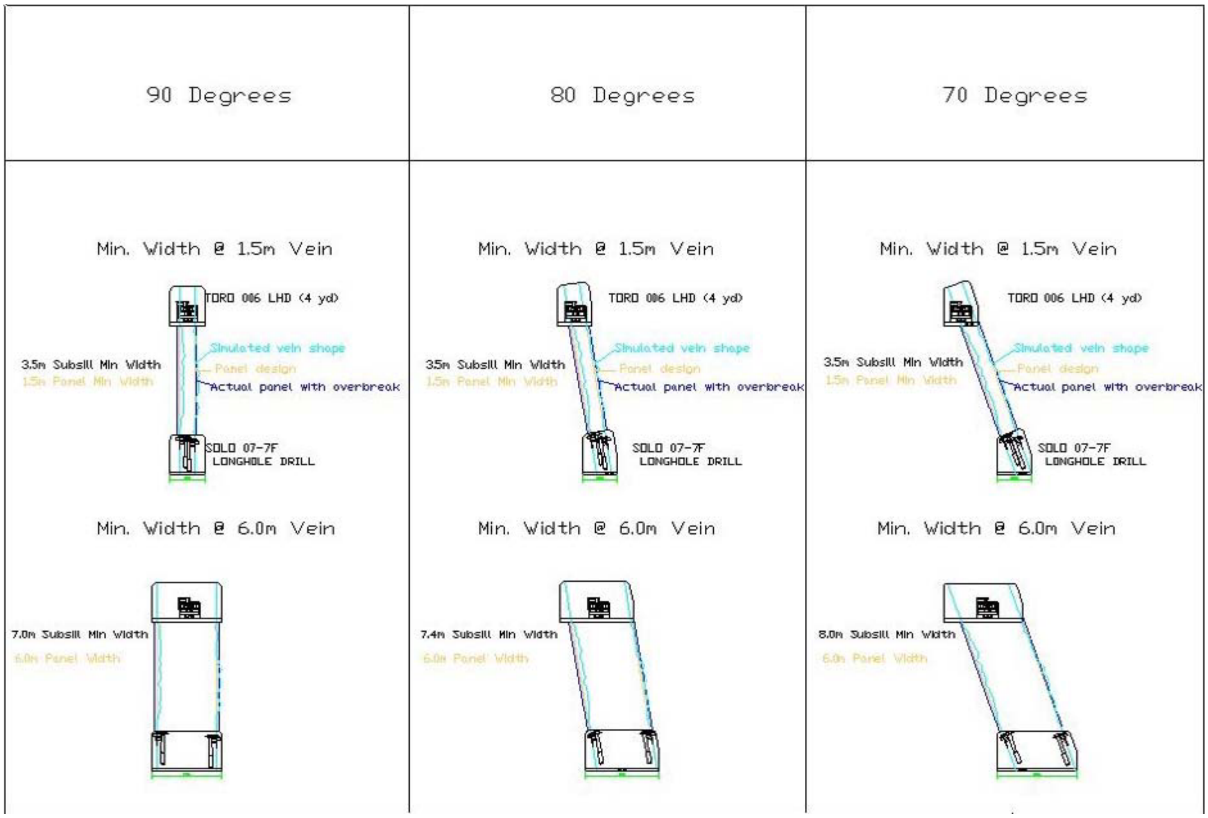
Figure 6.5. Typical Development Cross-Section.



Sill drift design is 4.5 m high where widths are less than 6 m, with a minimum width of 3.5 m. Sills greater than 6 m wide will be excavated 4 m high. The maximum expected sill width is 15 m based on geotechnical constraints discussed in Section 6.2 - Figure 6.6 below.

Figure 6.6 below shows LHD and drill equipment in narrow and average vein cross-sections.

Figure 6.6. Subsill and Panel Minimum Widths Shown at Different Vein Dips.



Minimum widths vary according to vein dip, as discussed in Figure 6.6 above.

6.1.8.2 Stoping

The stoping method is drifting on sill and longhole stoping. The method addresses issues of vein geometry, ground conditions, dilution, permafrost mining, productivity, and general safety. The vertical sublevel interval chosen is 15 m.

.

Longhole drilling will employ a 1.8 m by 1.8 m pattern using 76 mm diameter blast holes. Blast holes will be drilled upwards from the lower sublevel which will allow pre-drilled holes and hence permit increased longhole drill productivities. After blasting, three lines of unloaded holes must remain by regulation. Pre-drilled lines avoid drilling new holes near bootlegs or miss holes. The holes will be angled at 70° toward the stope, and drilled to breakthrough in order to avoid bridging below the upper sub-level.

The advantages and disadvantages in drilling upholes are summarized in Table 6.3 below:

Table 6.3 Comparison of Longhole Drill Orientations.

Advantages	Disadvantages
1.Drill Mist/Water drainage effective in permafrost.	1. Up hole loading near the stope is not as safe as downholes.
2. Holes are less likely to be blocked/plugged by debris compared to downholes.	2. Mucking and loading cycles conflict in the lower sub level
3. Maintaining pre-drilled holes is easier with upholes.	

Slot blast holes will be drilled up from the lower sublevel, broken through to the upper sublevel and then blasted as a drop raise. The short 11 m slot raises will be stage-blasted from the bottom up in two or three blasts to prevent blast freezing caused by lack of volume. Once the slot raise is opened up, then blasts of increasing volume extend the slot across the full width of the stope.

The open stope will be blasted up to a general maximum of 30 m, or to the limit of line of sight, whichever is less. The 11 m stope height will ensure that drilling accuracy should be excellent, once the operators are trained and experienced. Each blast will design for approximately 6 m (3 rows of holes at 1.8 m spacing). The final blast will occur after the stope is backfilled to minimize fly rock and maximize recovery.

Figure 6.7. Typical Longhole Stoping Schematic Section.

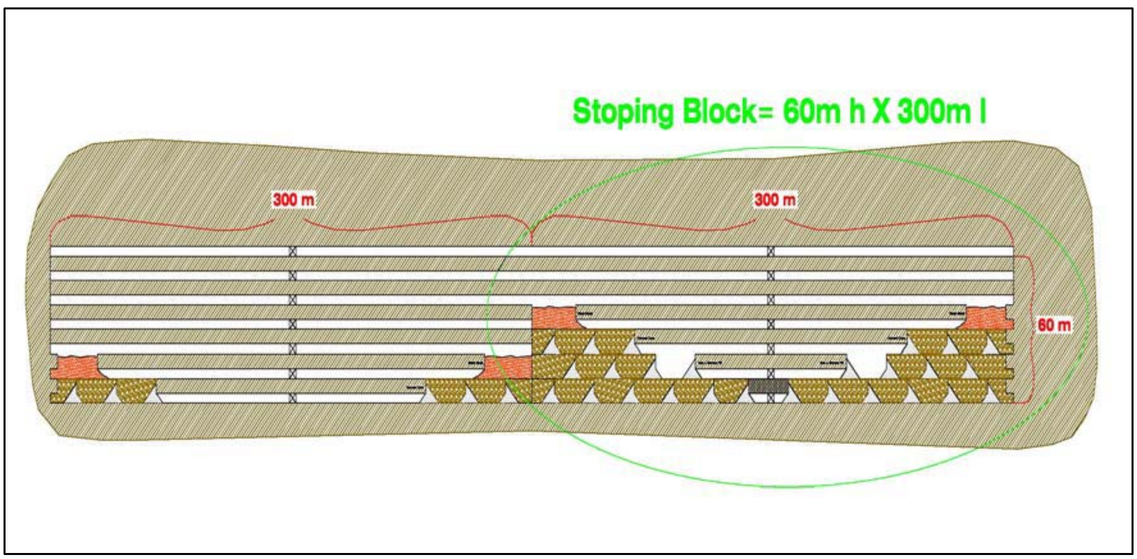


Figure 6.7 above is a typical longhole section that shows broken ore in red and rock fill in brown. Continuous stoping will be possible in some areas where there is access from both ends of a stope. Some of Big Bend orebody will be mined in this manner because there are two separate stope entry points along the strike of the orebody (**Error!**

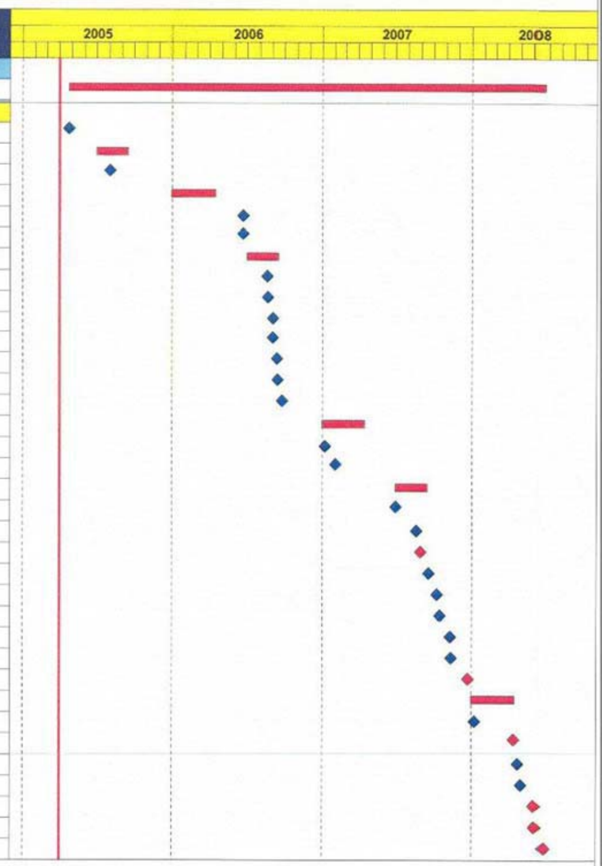
Reference source not found.). Multiple, subparallel veins characterize the South and North orebodies (**Error! Reference source not found.**). In some cases, stope strikelengths are shorter, but these areas will enjoy the benefit of shared access from the attack ramps (Figure 6.1 above).

Appendix C

Project Schedule

Feasibility Study Schedule

Act ID	Description	Original Duration	Early Start	Early Finish
Major Milestones				
		1155	23APR05	21JUN08
1080	Mill Foundations Start	0	23APR05 *	
1280	Ocean Freight to Pevek-2005	77	30JUN05 *	15SEP05
1090	Mill Foundations Completed	0		01AUG05
1300	Winter Road Transport from Pevek to Site-05 / 06	105	31DEC05 *	15APR06
2440	Air Strip Construction Completed	0		20JUN06
1100	Underground Mine Development Start	0	21JUN06	
2220	Ocean Freight to Pevek-2006	77	30JUN06 *	15SEP06
1140	Commence Mechanical Erection	0	19AUG06	
1720	Concrete Program Completed	0		19AUG06
1070	Man Camp Complete	0		01SEP06
1120	Surface Pit Pre-Stripping Commences	0	01SEP06	
1290	CAT 3516 Generators Start-up	0	12SEP06	
1160	Tailing Dam Phase I Start	0	13SEP06	
1470	Engineering Complete	0		23SEP06
1310	Winter Road Transport from Pevek to Site-06 / 07	105	31DEC06 *	15APR07
2450	Mill Installed	0		06JAN07
1130	Surface Minning Commences	0		01FEB07
2330	Ocean Freight to Pevek-2007	77	30JUN07 *	15SEP07
1170	Tailing Dam Phase I Completed	0		30JUN07
2370	Equipment Installation - Crushing Completed	0		19AUG07
2380	Equipment Installation - Grinding Completed	0		29AUG07
2400	Equipment Installation - Merrill Crowe Completed	0		18SEP07
2390	Equipment Installation - Refinery Completed	0		08OCT07
2430	Tailing Pond Ready for Water	0		15OCT07
2350	Service Bldg Structural Steel Completed	0		09NOV07
2410	Site Electrical Distribution Completed	0		11NOV07
2360	Equipment Installation - Leaching & CCD Complete	0		22DEC07
2340	Winter Road Transport from Pevek to Site-07 / 08	105	31DEC07 *	14APR08
2420	Service Building Equipment & Cranes Completed	0		07JAN08
1150	Major Mechanical Completion	0		11APR08
1110	UM Development Completed (South Portal)	0		21APR08
2320	Testing & Commissioning Main Generators Start	0	30APR08	
1980	Main Generators (4.5 MW) Start-Up	0	30MAY08	
1180	Mill Startup Commences	0	01JUN08 *	
1560	Dore Production	0	21JUN08	



Start date	01JAN05
Finish date	21JUN08
Data date	01APR05
Run date	10JUN05
Page number	1A

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Kupol Gold Project

Early bar
Progress bar
Critical bar
Summary bar
Critical point
Start milestone point
Finish milestone point