TECHNICAL REPORT ON MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES, ÇAYELI MINE, REPUBLIC OF TURKEY. PREPARED FOR INMET MINING CORPORATION

Report for NI 43-101

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1 SUMMARY

EXECUTIVE SUMMARY

Roscoe Postle Associates Inc. (RPA) was retained by Inmet Mining Corporation (Inmet) to prepare a NI43-101 compliant, independent Technical Report on Mineral Resources and Mineral Reserves of the Çayeli copper-zinc mine (the Project), located near the town of Çayeli, northeastern Turkey. RPA visited the mine in the course of completion of the Technical Report.

Çayeli is an underground copper and zinc mine wholly owned by Inmet. The mine is located east of the town and port of Rize, approximately 7.5 km from the Black Sea. Development began in early 1990 and commercial production commenced in November 1994. The deposit is a VMS Kuroko type, and the mining method is transverse sublevel retreat with paste and waste filling. Çayeli Bakir Isletmeleri A.S. (CBI), a wholly owned subsidiary of Inmet, has the surface rights to operate on the immediate mine property.

Currently, the major assets and facilities associated with the Project are:

- A massive sulphide orebody with a mineral reserve of 11.6M tonnes at an average grade of 3.65% Cu and 5.87% Zn as of December 31, 2005;
- A hoistroom, a headframe, and a 600 m deep shaft;
- A decline and a series of ramp-connected levels;
- An exhaust fan station;
- An explosive storage facility;
- An underground small maintenance shop;
- A 3,600 tpd mill;
- A process water pond;
- Two waste dumps;
- A crushing plant with eight ore storage bins;
- A maintenance and technical mine department building with office space, including a conference room, a warehouse, a dry;
- A canteen building with an adjacent infirmary;
- A laboratory building;
- A paste fill plant;
- An administration office;
- A pumping station.

CONCLUSIONS AND RECOMMENDATIONS

The proven and probable mineral reserves at December 31, 2005, are estimated to be 11.6M tonnes at an average grade of 3.65% Cu and 5.87% Zn. In 2005, the mill processed a total of 833,638 tonnes at an average grade of 3.84% Cu and 6.74% Zn. The copper concentrate production totaled 116,054 tonnes at an average grade of 22.81% Cu and a recovery rate of 82.62% copper. Zinc concentrate production totaled 83,903 tonnes at an average grade of 50.39% Zn and a recovery rate of 75.24% zinc.

RPA agrees with the methodology used for mineral resource and mineral reserve estimation. The mineral resources and mineral reserves are estimated using well-interpreted 3D mineralization solids, a good database, reliable assays, and block grade estimation techniques that are consistent with industry standards. All of the above has contributed to good reconciliation between mine and mill over the mine life. RPA confirms that the mineral resources and mineral reserves at Çayeli are estimated in accordance with National Instrument 43-101.

The major underground expansion program, which includes shaft deepening (completed), excavation of the associated shaft bottom infrastructure, ore handling system, and service ramp extension, is well advanced. The work associated with the development and installation of the shaft underground infrastructure, ore handling system, and service ramp extension was advancing on schedule at the time of the RPA visit. This program, once completed in the middle of 2006, will allow for sustained productivity from the lower parts of the deposit and will give access for exploration at depth.

In RPA's opinion, the Çayeli property has very good exploration potential, both on surface and underground. CBI retains large land holdings surrounding the mine, which host almost 15 km of the stratigraphic strike equivalent to the Çayeli ore horizon. Most of this ground remains essentially unexplored at this time.

The mine is sensitive to NSR parameters such as metal prices and operating costs. Based on long-term copper and zinc prices (US$1.10/lb and US$0.55/lb, respectively), the margin between the average NSR value and the 2005 operating cost per tonne of ore is US$16 ($72 per tonne vs. $56 per tonne). At current prices (Cu = US$2.30/lb, Zn = US$1.00/lb, Ag = US$9.50/oz, Au = US$550/oz), the NSR value is US$165, which gives a margin of approximately US$110 per tonne of ore.

Although RPA does not see problems that could represent material issues in the reserve estimation, we recommend the following:

1. **Land Tenure.** The mine and surrounding property comprise three separate leases. One of them, Licence OIR-10627, will expire in 2006. According to the mine geologists, Licence OIR-10627 is important for exploration. RPA recommends that, in this particular case, mine geologists be directly involved in the process of extending the expiration date of this lease because the Turkish government may require that geological reports be filed.

2. **RQD Database**. The RQD (Rock Quality Designation) values are not currently in a database. RPA recommends entering RQD values into a database, which, in combination with rock types, ore types, alteration zones, and microseismic data, may be very useful for development heading planning, stope design, and ground support.

   RPA also recommends using the RQD values to assess, on a stope by stope basis, if the ‘in-time’ drifting, production drilling, blasting, and mucking philosophy is applicable.

3. **Density**. Density determinations are not conducted for stope definition drill holes, channel samples, delineation drill holes in which sludge is recovered, or in the development sludge. In these cases, average values are assigned based on the rock code designation.

   RPA recommends that density be determined by an interpolation technique, such as kriging, rather than assigning a density value based on rock codes. Density would then be based on more local values rather than on average or regional values.

4. **QA/QC.** CBI reports that the lower of the original and checked values is used for grade estimation and that there is no field for check assays in the database. RPA recommends using the average of the original and checked values for grade estimation and having a field in the database for check assays. A database should be set up for check assays that would contain sample type, original values, checked values, and dates. The check assay results should be included in a Mineral Resource and Mineral Reserve Estimates Technical Report. RPA also recommends that results be presented on XY graphs.

RPA recommends that check assay programs carried out on geology samples be more closely followed by geologists. Check assays should also be requested by geologists, not only by the laboratory manager.

5. **Assay Statistics by Domain**. RPA recommends that statistics on raw data (prior to compositing) be presented for each wireframe (zone) without application of any cut-off. Statistical reports, histograms, and probability plots should be presented as backup in a Mineral Resource and Mineral Reserve Estimates report.

6. **Variography**. RPA recommends that variography be done for silver and gold.

7. **Capping of High Grade Samples.** RPA has reviewed the Cu-Zn probability plots that have been produced for each of the mineralized domains and recommends that further consideration be given to the use of grade capping, versus outlier limitation, as a method of addressing high grade issues.

8. **NSR vs. EqCu**. Until December 2005, CBI had been using an equivalent copper grade (EqCu %) as a common parameter from which the cut-off grade was derived. The EqCu formula was derived from NSR calculations, in which a fixed head grade (generally, the reserve grade) and a fixed recovery factor were assigned to each metal.

   RPA recommended using NSR values, rather than EqCu values, for resource and reserve reporting, cut-off determination, and stope design. NSR calculations consider variable metal recovery based on head grades.

9. **Cut-Off Grade**. The current 2006 Five Year Plan operating costs are forecast at $52 per tonne, and experience is $56. In this estimate, Inmet used a cut-off grade of $46, or an EqCu of 3.3%, which, in RPA's opinion, is appropriate on an incremental basis.

10. **Block Model Grade Estimation**. Copper and zinc grades in the block model are estimated using the Ordinary Kriging interpolation method. Silver, gold, lead, and density are estimated using the Inverse Distance weighting method.

    Each domain zone is treated based on "hard" boundary limitations, which means that block grades could only be estimated using samples located inside that specific zone (for example, the Main Zone Clastic Ore (zone 1) block grades were estimated using only samples located inside zone 1).

    RPA generally agrees with the 'hard' boundary approach, which is based on ore type, however, RPA recommends being very careful in this approach in the southern part of the deposit where there is no displacement of the mineralized zone by the Scissor Fault. RPA recommends that a 'box' be outlined in the southern part of the deposit in the vicinity of the Scissor Fault where the 'hard' boundary is not applicable. For example, RPA has observed significant differences in grades for two adjacent blocks that belong to the same ore type (Clastic Ore) but different domains (one block being in the Main Zone and the other block in the Deep Zone), and where the Scissor Fault shows no apparent displacement. In this particular case, the block on the hanging wall of the fault had an estimated grade of 12.62% Zn, while the adjacent block on the footwall side of the fault had an estimated grade of 4.10% Zn. Grades are overestimated on one side of the fault and underestimated on the other.

11. **Grade Reconciliation**. CBI reports that short-term grade reconciliation (daily, weekly, monthly) between the geology model and mill grades is essentially based on the adjustment of stope broken ore grades to mill grades.

    Short-term grade reconciliation is not easy when ore is stockpiled in different piles on surface before being sent to the mill, therefore, RPA recommends the following:

    - In order to be coherent in the process of grade reconciliation, the forecast diluted grade of stopes should be compared to the mill.

    - The broken ore stockpile mine/mill grade ratio should be used to back-calculate stope grades. Stockpile grades are currently not used for grade reconciliation.

    - A more detailed study on broken ore sampling should be conducted in order to see if the sampling quantity is appropriate, or whether additional sampling is necessary. Also, broken ore sample grades are generally higher than mill grades. Such results suggest that broken ore sampling should be revisited.

    Long-term grade reconciliation should be carried out between the mill heads and the following:

    - Broken ore samples.
    - Stopes that have been surveyed using a Cavity Monitoring System (CMS).
    - Diluted block model.
    - Undiluted block model.

12. **'In-time' Stope Preparation**. The stope preparation process is based on the "in time" drifting, production drilling, blasting, and mucking philosophy. The reason for this is that, due to the relatively poor ground conditions, the excavations often deteriorate quickly.

    RPA recognizes the need to respect the potential for rapid deterioration of prematurely excavated openings. The in-time preparation philosophy is an appropriate measure to deal with this problem, however, it slows stoping productivity and ties mine resources to a given extraction area in a very unproductive way.

RPA recommends utilizing stope delineation drilling for testing RQDs and assessing, on a stope by stope basis, if a given stope qualifies for an exemption from the "in time" development and can then be prepared and extracted in a sequential order, without interruption.

13. **Ground Support**. RPA agrees that the ground support measures adopted at the Çayeli mine are working well and, in general, are appropriate.

14. **Five Year Mine Plan**. The Çayeli operation does not have a life of mine plan. The operation developed a preliminary five-year mining plan aimed at mining one million tonnes per year of ore for the next five years. RPA recommends updating the life of mine plan for the operation based on mineral reserves.

TECHNICAL SUMMARY

PROPERTY DESCRIPTION AND LOCATION

The Çayeli mine is located eight kilometres south of the town of Çayeli, a fishing and tea farming village with a population of 25,000, located on the south coast of the Black Sea.

LAND TENURE

The mine and surrounding property comprise three separate leases. The total area of the property is approximately 13,000 hectares.

HISTORY

The work which led to the present mine was started in 1967 by the Turkish Mineral Research and Exploration Institute (MTA). MTA carried out a geophysical survey and drilling program, and drove an adit into massive sulphide ore located just south of the current deposit.

In 1981, CBI was formed as a company, with Etibank (now Eti Holding AŞ), Phelps Dodge, and Gama Endüstri AŞ as shareholders, to develop the orebody. Phelps Dodge sold its 45% share to Metall Mining (now Inmet Mining) in 1988. Further underground work and metallurgical testing was done in the period from 1988 to 1991. Positive results were obtained and a production decision was made. Site construction started in 1992, basic mine infrastructure commenced in 1993, and the first concentrate was produced in August 1994. The total capital cost of the operation was approximately $200M.

In 2004, Inmet purchased Eti Holding AŞ's 45% interest and became the 100% owner of the Project.

GEOLOGY

The Çayeli orebody is situated between hangingwall pyroclastites and flows and footwall rhyolite. Mineralization, which is typical of a volcanogenic massive sulphide environment, is known over a strike length of 920 m. Massive sulphides form the majority of the ore in the deposit. The ore comprises varying amounts of pyrite, chalcopyrite, and sphalerite, with minor dolomite and barite. The measured resource has a strike length of approximately 600 m, a vertical depth of over 600 m, and varies in thickness from a few metres to 80 m, with a mean of approximately 20 m. The average dip is 65° NNW in the upper part of the deposit and approximately 50° at depth. Below the massive sulphides, there are varying widths of stringer mineralization that can often exceed the cut-off grade due to chalcopyrite veining.

MINERAL RESOURCES AND MINERAL RESERVES

Mineral resource and mineral reserve estimates are based on all available drilling, sampling, and mapping data as of December 31, 2005. The resource and reserve estimations were completed using the MineSight software system.

The deposit is segregated into three distinct geologic domains based primarily on the style of mineralization. Each of these domains is further divided by a fault (the Scissor Fault) which separates the Main and Deep Ore zones. Three-dimensional wireframes were developed for all domains.

Copper and zinc grades in the block model are estimated using the Ordinary Kriging interpolation method. Silver, gold, lead, and density estimations are done using the Inverse Distance weighting method.

The December 31, 2005 Mineral Resource and Mineral Reserve estimates for the Çayeli Mine, according to NI43-101 standards, are summarized in the tables below.

TABLE 1-1 MINERAL RESOURCES
Inmet Mining Corporation – Çayeli Mine

Category	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	$NSR/t
Measured	1.91	3.08	3.09	22	0.50	0.18	57
Indicated	2.65	3.05	2.55	20	0.44	0.15	54
Meas.+ Ind.	**4.56**	**3.06**	**2.78**	**21**	**0.46**	**0.16**	**55**
Inferred							
Çayeli	0.62	3.38	2.85	18	0.26	0.12	58
Russian Adit	0.45	3.29	11.08	N/A	N/A	N/A	N/A
Total Inferred	**1.07**	**3.34**	**6.29**	**N/A**	**N/A**	**N/A**	**N/A**

Notes:
1. CIM definitions were followed for mineral resources.
2. Mineral resources are estimated at a cut-off grade of $35 NSR/tonne of ore
3. Mineral resources are estimated using prices of US$1.10 per pound Cu, US$0.55 per pound Zn, US$5.60 per ounce Ag, $US450 per ounce Au
4. Measured and indicated mineral resources are exclusive of mineral reserves.
5. Russian Adit inferred resources: Au and Ag have never been assayed.

TABLE 1-2 MINERAL RESERVES
Inmet Mining Corporation – Çayeli Mine

Category	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	$NSR/t
Proven	4.70	3.77	5.85	44	0.59	0.30	73
Probable	6.90	3.57	5.88	52	0.53	0.36	69
Total	**11.60**	**3.65**	**5.87**	**49**	**0.56**	**0.34**	**70**

Notes:
1. CIM definitions were followed for mineral resources.
2. Mineral reserves are estimated at a cut-off grade of $46 NSR/tonne of ore
3. Mineral reserves are estimated using prices of US$1.10 per pound Cu, US$0.55 per pound Zn, US$5.60 per ounce Ag, $US450 per ounce Au.

MINING OPERATIONS

The mine design is based on underground bulk mining methods with the use of delayed backfill to extract the ore in a sequential manner. A single production shaft located on the footwall side of the orebody and a service ramp provide access to the mine.

The primary mining method selected for mining the Çayeli orebody is retreat transverse and longitudinal long hole stoping with paste fill and loose or consolidated waste rock backfill. The stopes are mined in primary, secondary, and tertiary sequencing. The primary and secondary stopes are mined as transverse and the tertiary as longitudinal stopes.

The main levels are developed off the service ramp along the strike of the orebody at 45 m to 100 m vertical intervals. From the top of the mine down to the 800 m level, levels are located on the hangingwall side and, from the 800 m level down to the bottom of the mine, they are located on the footwall side of the orebody. The main levels divide the orebody into mining blocks.

Sublevels are developed within the orebody, along the contact with the hanging wall or in the centre of the orebody in the upper parts of the mine. In the lower parts of the mine, sublevels are developed along the contact with the footwall. The sublevels are part of the stopes. The ore within sublevel drift configurations is recovered after the primary and secondary stopes in a block have been mined out and backfilled. Extraction of the ore from the sublevel drifts is called tertiary stoping and is done in a retreat scenario.

The sublevel vertical distance is dictated by the stope height. In the upper parts of the mine, it is 20 m, allowing development of a 15 m high by seven metres wide stope bench for production drilling. In the lower part of the mine, the sublevels will be developed at a 15 m vertical distance, allowing development of a 10 m high by 10 m wide bench for production drilling.

MINERAL PROCESSING

During mining, the ore is separated into three different types and hoisted to surface, where it is transferred into one of the eight storage compartments. Each compartment has a 3,500 tonnes storage capacity. Before being sent to the concentrator for processing, the ore from surface storage bins is blended in order to achieve optimal metallurgical results.

The concentrator facility includes conventional crushing, grinding, differential flotation, and pressure filtration.

In 2005, the average feed grades were 3.84% copper and 5.74% zinc. The concentrate grade for copper averaged 22.81% at an 82.64% recovery and the concentrate grade for zinc averaged 50.39% at a 75.24% recovery. In the last five years, average recovery has been 86.05% for copper and 72.12% for zinc.

Based on the requirements of the concentrator, the mine carries out segregation of three ore types, Clastic, Black, and Yellow. Clastic ore is material that, based on visual estimates, contains more than 10% sphalerite fragments in massive sulphides. These fragments are generally fine-grained and form intergrowths with chalcopyrite, which may affect the copper recoveries.

The Black and Yellow ore types are based on the contained zinc grade and are mined separately to allow for optimal grade blending from the stockpile bins on surface. These two ore types are referred to as "Spec ore". Black ore is defined as material with more than 4.5% Zn and a Cu/Zn ratio of less than one. Yellow ore consists of copper-rich, zinc-poor sulphides. Yellow ore is comprised of approximately one half massive sulphides and one half stockwork material (stockwork mineralization generally contains very little sphalerite).

Two copper concentrates are produced:

- Spec concentrate. In general, a tonne of Yellow Ore combined with approximately 0.42 t of Black Ore yields Spec concentrate.

- Non-Spec concentrate. This concentrate is comprised of Clastic Ore and the balance of Black ore. Approximately 10% of copper concentrate is generated from this type of ore. The concentrate is high in precious metals and also contains higher zinc. The number of customers for this type of product is rather limited.

In the 2006 Five Year Milling Plan, 40% of ore mined will yield Spec concentrate and 60% will yield Non-Spec concentrate. The average grades of Spec ore feed are 4.3% Cu and 5.1% Zn. The average grades of Non-Spec ore feed are 3.4% Cu and 7.2% Zn. The average head grades for both types of feed are 3.7% Cu and 6.4% Zn. The NSR per tonne of ore for Spec ore feed is $80, while the NSR per tonne of ore for Non-Spec ore feed is $67 (Cu = US$1.10/lb, Zn = US$0.55/lb). Based on the above ratio, the average NSR per tonne of ore is $72.

Copper and zinc concentrates are loaded onto trucks for transportation to the Rize port. Each truck is weighed at the plant gate prior to transporting the concentrate.

The table below summarizes milling production between 2001 and 2005.

TABLE 1-3 MILL PRODUCTION – 2001-2005
Inmet Mining Corporation - Çayeli Mine

Year	Milling Tonnes	Copper Concentrate				Zinc Concentrate		
		Tonnes	% Cu	g/t Au	g/t Ag	Tonnes	% Zn	g/t Ag
2001	816,480	135,957	24.25	0.70	66.36	50,129	50.52	121.63
2002	895,423	132,133	24.65	0.80	78.62	64,778	51.06	126.42
2003	927,892	135,923	24.64	0.77	73.99	65.853	51.03	116.40
2004	765,329	112,915	23.71	0.85	85.35	65,713	50.77	121.44
2005	833,638	116,054	22.81	1.19	96.36	83,903	50.39	126.20
Total	**4,238,763**	**632,982**	**24.06**	**0.85**	**79.45**	**330,376**	**50.74**	**122.65**
Ave./Year	847,753	126,596	24.01	0.86	80.14	66,075	50.75	122.42

TAILINGS DISPOSAL

A submarine tailings disposal system is used whereby tailings are transported by pipeline along the Büyükdere River for discharge into the anoxic environment of the Black Sea, 3,030 m offshore at a depth of 250 m.

ENVIRONMENTAL CONSIDERATIONS

The site has an environmental plan in place to control dust and effluent emissions in accordance with the Turkish Mining and Environmental regulations. The plan requires the operation to monitor on a regular basis the effluents at the mine site and its surroundings. The results of monitoring are reported to the authorities on a quarterly basis.

In RPA's opinion, the site maintains proper records in regard to controlling the effluents and is in compliance with the regulations. At the time of RPA's visit, there were no reported violations of the regulations with regard to dust control, water discharge from the mine, maintenance of the acid generating rock storage area, noise control at the site, and tailings discharge in the Black Sea.

At the present time, CBI is not required to prepare a closure plan for the mine, however, CBI is committed to decommission the mine in accordance with risk-based criteria and applicable regulation and permits.

In 1999, CBI commissioned SRK UK Ltd. to generate a closure plan for the mine. The final report was received in November 1999. The closure activities have been broken down into five main component areas:

- Underground mine;
- Process plant and general site infrastructure;
- Waste rock dumps;
- Port facilities;
- Tailings disposal system.

For each component area, the main decommissioning and closure issues are highlighted along with their respective performance criteria, closure investigations and test work, available closure options, closure actions, provisions for the demonstration of closure performance, and the costs of the closure actions. The plan anticipates a decommissioning and closure period of approximately 12 months for the site infrastructure and underground mine, and five years for the site rehabilitation and environmental compliance monitoring. The decommissioning and closure costs are estimated to be $6.9 million.

CAPITAL AND OPERATING COST ESTIMATES

The mine typically has an annual sustaining capital budget between $6 million and $8 million. In 2003, a number of projects were initiated to improve recovery at the processing plant. In 2004, a major underground expansion program was undertaken at a total estimated cost of $17 million. This program included shaft deepening, excavation of the shaft new infrastructure, ore handling system, and service ramp extension. The shaft deepening was completed in October 2005. The remaining elements of the program are expected to be complete in the middle of 2006.

The tables below summarize the mine capital expenditures spent between 2001 and 2005 and the 2006 budgeted capital expenditure breakdown by department.

TABLE 1-4 CAPITAL EXPENDITURES - 2001-2005
Inmet Mining Corporation – Çayeli Mine

Years	Capital Expenditures US$ M
2001	4.697
2002	7.965
2003	5.910
2004	16.634
2005	18.722

TABLE 1-5 CAPITAL EXPENDITURE BUGDET IN 2006
Inmet Mining Corporation – Çayeli Mine

Department	Capital Expenditures US$ M
Mine	2.4
Shaft Project	10.8
Mill	2.3
General and Administration	0.2
Deferred development	4.8
Total	**20.6**

The deferred development capital covers primary underground development such as ramps, main levels, and underground infrastructure.

The operating costs from 2001 to 2005 are summarized in the table below.

TABLE 1-6 OPERATING COSTS PER TONNE MILLED - 2001-2005
Inmet Mining Corporation – Çayeli Mine

	2001	2002	2003	2004	2005
Tonnes Milled	816,480	895,423	927,892	765,329	833,638
Mine	$12.37	$13.60	$17.20	$23.97	$26.48
Mill	$8.51	$9.16	$10.59	$12.98	$12.86
Administration	$8.05	$8.70	$12.54	$16.65	$17.27
Total Cost	**$28.93**	**$31.46**	**$40.33**	**$53.60**	**$56.61**

In the last three years, operating costs have significantly increased at the operation. This is mostly attributed to a Turkish Lira appreciation against the US dollar and major wage and salary increases that took place at the same time.

In 2004 and 2005, CBI engaged Proudfoot Consultants to carry out a continuous improvement program aimed at enhancing communications and streamlining of the operations. The cost of executing this program added approximately US$5 per tonne to

US$6 per tonne to the unit cost for 2005 as a one time charge. Additinally, an increase in energy, material, and part costs contributed to the operating cost growth.

Total manpower at the Çayeli Mine is 467 employees.

RPA reviewed the net smelter return (NSR) parameters and estimates that the average value of ore per tonne is $70 by using the Five Year Mine Plan head grades and long-term metal prices (Cu = US$1.10/lb, Zn = US$0.55/lb).

2 INTRODUCTION AND TERMS OF REFERENCE

Roscoe Postle Associates Inc. (RPA) was retained by Inmet Mining Corporation (Inmet) of Toronto, Canada, to prepare an independent Technical Report on the Çayeli copper-zinc mine which is located near the town of Çayeli in northeastern Turkey. The purpose of this report is to audit the mineral resource and mineral reserve estimates and prepare a technical review of the Çayeli operations. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects. RPA visited the property on December 6-9, 2005.

Çayeli is an underground copper and zinc mine wholly owned by Inmet. The mine is located east of the town and port of Rize, approximately 7.5 km from the Black Sea. Development began in early 1990 and commercial production commenced in November 1994. The deposit is a VMS Kuroko type and the mining method is transverse sublevel retreat with paste and waste filling. Çayeli Bakir Isletmeleri A.S. (CBI), a wholly owned subsidiary of Inmet, has the surface rights to operate on the immediate mine property (IR-7540).

Currently, the major assets and facilities associated with the mine are:

- A massive sulphide orebody with a mineral reserve of 11.6 M tonnes at an average grade of 3.65% Cu and 5.87% Zn as of December 31, 2005;
- A hoistroom, a headframe, and a 600 m deep shaft;
- A decline and a series of ramp-connected levels;
- An exhaust fan station;
- An explosive storage facility;
- An underground small maintenance shop;
- A 3,600 tpd mill;
- A process water pond;
- Two waste dumps;
- A crushing plant with eight ore storage bins;
- A maintenance and technical mine department building with office space, including a conference room, a warehouse, a dry;
- A canteen building with an adjacent infirmary;
- A laboratory building;
- A paste fill plant;

- An administration office;
- A pumping station.

Electric power is provided via a 31.5kV line running from the Turkish national grid system. Process makeup water and additional site water needs are provided from a series of wells drilled on the property. The majority of the tailings produced are utilized as paste backfill in the mining sequence, and any excess is discharged via a 20 km pipeline at a depth of –250 m into the Black Sea.

Access to the mine, from Çayeli, is via paved road for a distance of eight kilometres towards the small village of Madenli. The Karadeniz Highway provides transportation in the east-west direction along the Black Sea coast from Samsun to the Georgian border located approximately 100 km east of Çayeli.

SOURCES OF INFORMATION

A site visit was carried out from December 6 to December 9, 2005, by Bernard Salmon, RPA Consulting Geological Engineer, and Andrew Hara, RPA Associate Mining Engineer. Both the surface infrastructure and underground workings were visited.

Discussions were held with the Çayeli mine personnel as follows:

- Mr. Joe Boaro, P.Eng., Technical Manager
- Mr. Haci Karakus, Geological Engineer, Chief Geologist
- Mr. Ertan Uludag, Geological Engineer, Mine Resource Geologist
- Mr. Mine Nihat Soyer, Mine Engineer, Mine Planning Engineer

In preparation of this report, RPA reviewed technical information and documents on the Project, including "Technical report on the Mineral Resource and Mineral Reserve Estimates at the Çayeli Mine, Turkey - Statement as of December 31, 2004" and associated reports, drawings, and spreadsheets. Digital files of the Çayeli resource estimation block model were also provided to RPA. Checks on basic data were performed in order to judge whether drilling, sampling, assaying, and other data were acquired under mining industry accepted standards and under Mining Standards Task Force Final Report (TSC/OSC, 1999) best practice guidelines.

Project review and preparation of the NI-43-101 Technical Report was carried out under the direction of Graham G. Clow, P. Eng., RPA Principal Mining Engineer. Mineral resources and mineral reserves were reviewed and audited by Bernard Salmon, Eng. Andrew Hara, P. Eng., reviewed the mining aspects, ore processing, cost estimation and environmental issues.

The documentation reviewed, and other sources of information, are listed at the end of this report in Item 20 References.

LIST OF ABBREVIATIONS

Units of measurement used in this report conform to the SI (metric) system. All currency in this report is US dollars (US$) unless otherwise noted.

List of Abbreviations:

μ	micron	km^2	square kilometre
°C	degree Celsius	kPa	kilopascal
°F	degree Fahrenheit	kVA	kilovolt-amperes
μg	microgram	kW	kilowatt
A	ampere	kWh	kilowatt-hour
a	annum	L	liter
m^3/h	cubic metres per hour	L/s	litres per second
cfm	cubic metres per minute	m	metre
Bbl	barrels	M	mega (million)
Btu	British thermal units	m^2	square metre
C$	Canadian dollars	m^3	cubic metre
cal	calorie	min	minute
cm	centimeter	MASL	metres above sea level
cm^2	square centimeter	mm	millimetre
d	day	mph	miles per hour
dia.	diameter	MVA	megavolt-amperes
dmt	dry metric tonne	MW	megawatt
dwt	dead-weight ton	MWh	megawatt-hour
ft	foot	m^3/h	cubic metres per hour
ft/s	foot per second	opt, oz/st	ounce per short ton
ft^2	square foot	oz	Troy ounce (31.1035g)
ft^3	cubic foot	oz/dmt	ounce per dry metric tonne
g	gram	ppm	part per million
G	giga (billion)	psia	pound per square inch absolute
Gal	Imperial gallon	psig	pound per square inch gauge
g/L	gram per litre	RL	relative elevation
g/t	gram per tonne	s	second
gpm	Imperial gallons per minute	st	short ton
gr/ft^3	grain per cubic foot	stpa	short ton per year
gr/m^3	grain per cubic metre	stpd	short ton per day
hr	hour	t	metric tonne
ha	hectare	tpa	metric tonne per year
hp	horsepower	tpd	metric tonne per day
in	inch	US$	United States dollar
in^2	square inch	USg	United States gallon
J	joule	USgpm	US gallon per minute
k	kilo (thousand)	V	volt
kcal	kilocalorie	W	watt
kg	kilogram	wmt	wet metric tonne
km	kilometre	yd^3	cubic yard
km/h	kilometre per hour	yr	year

RELIANCE ON OTHER EXPERTS

This report has been prepared by Roscoe Postle Associates Inc. (RPA) for Inmet. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to RPA at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by Inmet and other third party sources.

For the purpose of this report, RPA has relied on ownership information provided by Inmet. RPA has not researched property title or mineral rights for the Çayeli Project and expresses no legal opinion as to the ownership status of the property.

3 PROPERTY DESCRIPTION AND LOCATION

The Çayeli mine is located eight kilometres south of the town of Çayeli, a fishing and tea farming village with a population of 25,000 located on the south coast of the Black Sea (Figure 3-1).

LAND TENURE

The mine and surrounding property consists of three separate leases as listed in Table 3-1 and shown in Figure 3-2. The total area of the property is approximately 13,000 hectares.

TABLE 3-1 LIST OF PROPERTIES
Inmet Mining Corporation – Çayeli

Licence #	Registration #	Size (ha)	Exp. Date	Status
IR 7540	67923	334.4	29/7/2044	Operation Licence
IR 6649	36375	8568.9	6/2/2011	Operation Licence
OIR 10627	54243	4286.6	9/10/2006	Pre-operating Licence

In 2006 Licence OIR-10627 will expire. At the time of the site visit, discussions with Mr. Collin Burge, P.Geo, Project Geologist, and Messrs. Haci Karakus and Ertan Uludag, mine geologists, indicated that this piece of land is considered important for exploration. Although all mining permits and leases are under the responsibility of the Land Manager based at the mine, RPA recommends that, in this particular case, mine geologists be directly involved in the process of extending the expiration date of this lease because the Turkish government may require that geological reports be filed. At the time of the visit, Mr. Haci Karakus, Chief-Geologist, confirmed to RPA his leading role in the process.


Inmet Mining Corporation
Çayeli Project
Republic of Turkey
Location Map
Figure 3-1
March 2006
0 50 100 150 200
Kilometres
Legend:
National Capital
City / Town
Road
Railroad
International Border
ÇAYELI MINE
Black Sea
Mediterranean Sea
Sea of Marmara
Straits of Bosporous
Dardanelles
Gulf of Antalya
Russia
Georgia
Armenia
Az.
Iran
Iraq
Syria
Lebanon
Cyprus
Bulgaria
Greece
Crete
Rhodes
Khios
Lévos
Damascus
Beirut
Nicosia
Ankara
Rize
Artvin
Trabzon
Ordu
Carsamba
Samsun
Sinop
Inebolu
Kastamonu
Karabük
Zonguldak
Eregli
Izmit
Adapazari
Üsküdar
Istanbul
Edirne
Kirklareli
Kesan
Canakkale
Eceabat
Balikesir
Bursa
Manisa
Izmir
Bodrum
Marmaris
Denizli
Kütahya
Eskisehir
Afyon
Burdur
Isparta
Antalya
Gazipasa
Konya
Karaman
Silifke
Mersin
Tarsus
Eregli
Adana
Iskenderun
Antioch
Nigde
Kayseri
Kirikkale
Cankiri
Amasya
Tokat
Sivas
Malatya
Kahramanmaras
Gaziantep
Urfa
Akcakale
Diyarbakir
Elâzig
Erzincan
Erzurum
Kars
Karaköse
Van
Tatvan
Mus
Batman
Kurtalan
Mardin
Nusaybin
Kura
Aras
Coruh
Murat
Van Gölü
Euphrates
Tigris
Great Zab
Yesilirmak
Kizilirmak
Kizil Irmak
Seyhan
Ceyhan
Sakarya
Tuz Gölü
Beysehir Gölü
Büyük Menderes
42°
36°
30°
40°
N
Pazar
Ardeşen
Çamlı
Çayeli
Güneysu
İkizdere
Kalkandere
Derepazarı
İyidere
Hayrat
Çaykara
Dernekpazarı
Sürmene
Köprübaşı
Araklı
Arsin
Yomra
TRABZON
Akçaabat
Maçka
Düzköy
Çarşıbaşı

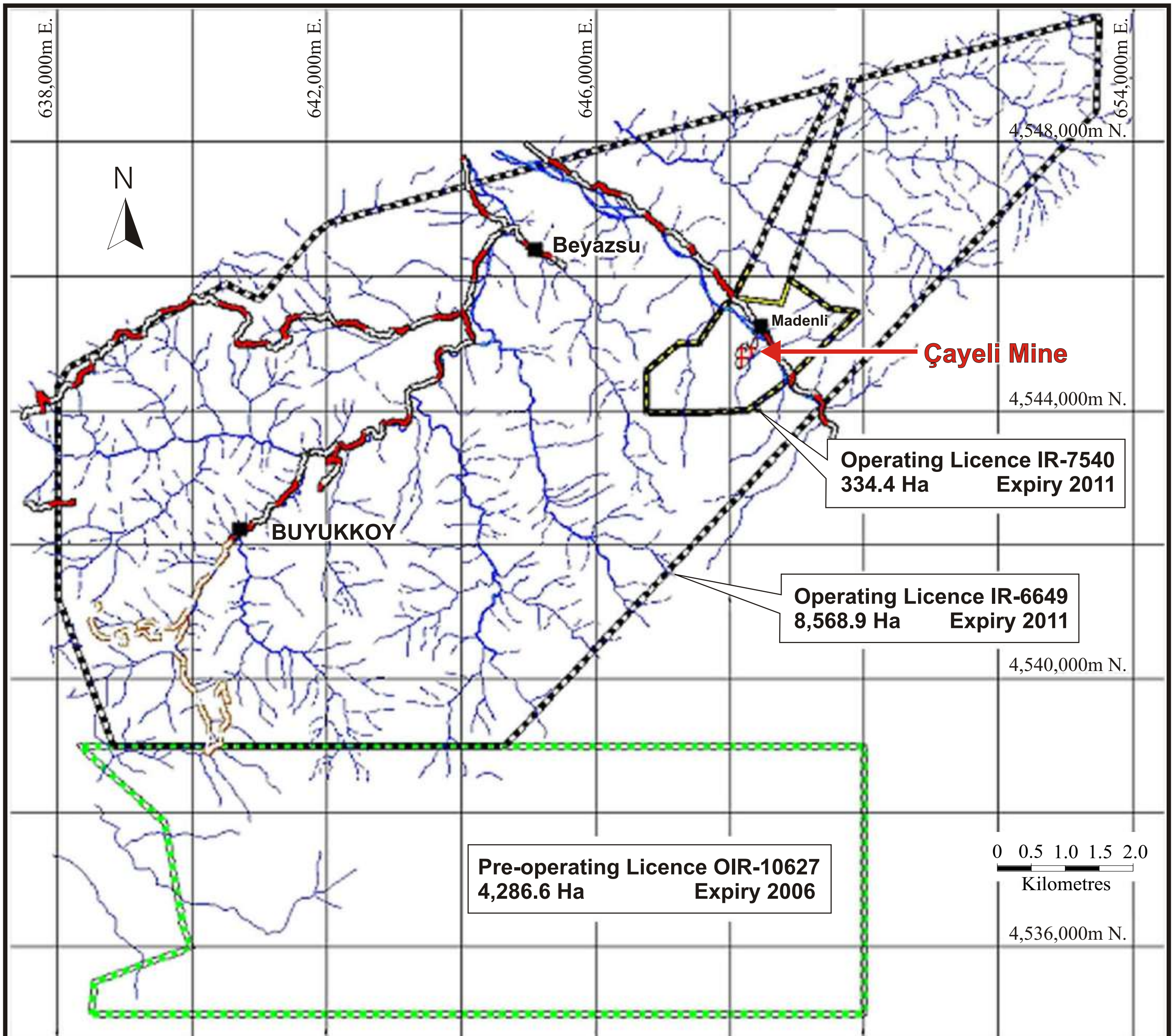


Figure 3-2

Inmet Mining Corporation

Çayeli Mine

Republic of Turkey

Property Boundary Map

March 2006

Source: Sim, 2004.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

Access to the mine from Çayeli is via paved road for a distance of eight kilometres towards the small village of Madenli (Figure 4-1). The Karadeniz Highway provides transportation in the east-west direction along the Black Sea coast from Samsun to the Georgian border located approximately 100 km east of Çayeli. The port city of Rize, with a population of approximately 80,000, is located 20 km west of Çayeli. An international airport is located in the city of Trabzon (pop. 250,000), approximately 100 km west of Çayeli.

CLIMATE

Climate is moderated by the proximity to the sea, with hot humid summers and generally wet winters. Rainfall is abundant in this area, averaging over 2.5 m per year. As a result, the area is lush, with vegetation including a wide variety of flowering plants such as rhododendrons and azaleas.

Tea (Cay) farming is by far the most common cash crop in the area as the short, bushy plant can be seen almost everywhere. Local gardens are also common with a variety of vegetables for personal consumption.

LOCAL RESOURCES

The mine employs approximately 467 people in its operations, the large majority of whom have been hired locally. Other skilled and professional workers have migrated from other parts of the country. Only 1.5% of the workforce is expatriates.

SITE INFRASTRUCTURE

CBI has the surface rights to operate on the immediate mine property (IR-7540). Surface and underground infrastructures consist of the following:

- A massive sulphide orebody with a mineral reserve of 11.6 M tonnes at an average grade of 3.65 Cu, 5.87% Zn as of December 31, 2005;
- A hoistroom, a headframe, and a 600 m deep shaft;
- A decline and a series of ramp-connected levels;
- An exhaust fan station;
- An explosive storage facility;
- An underground small maintenance shop;
- A 3,600 tpd mill;
- A process water pond;
- Two waste dumps;
- A crushing plant with eight ore storage bins;
- A maintenance and technical mine department building with office space, including a conference room, a warehouse, a dry;
- A canteen building with an adjacent infirmary;
- A laboratory building;
- A paste fill plant;
- An administration office;
- A pumping station.

Electric power is provided via a 31.5kV line running from the Turkish national grid system. Process makeup water and additional site water needs are provided from a series of wells drilled on the property. The majority of the tailings produced are utilized as paste backfill in the mining sequence and any excess is discharged via an eight kilometre pipeline at a depth of 275 m in the Black Sea.

PHYSIOGRAPHY

The mine is located in the foothills of the Kackar Mountains which extend along the eastern portion of the southern Black Sea coast. This is a rugged mountain range with peaks reaching over 4,000 m elevation.

The mine is located at an elevation of 100 m within rugged local hills up to 1,000 m high.

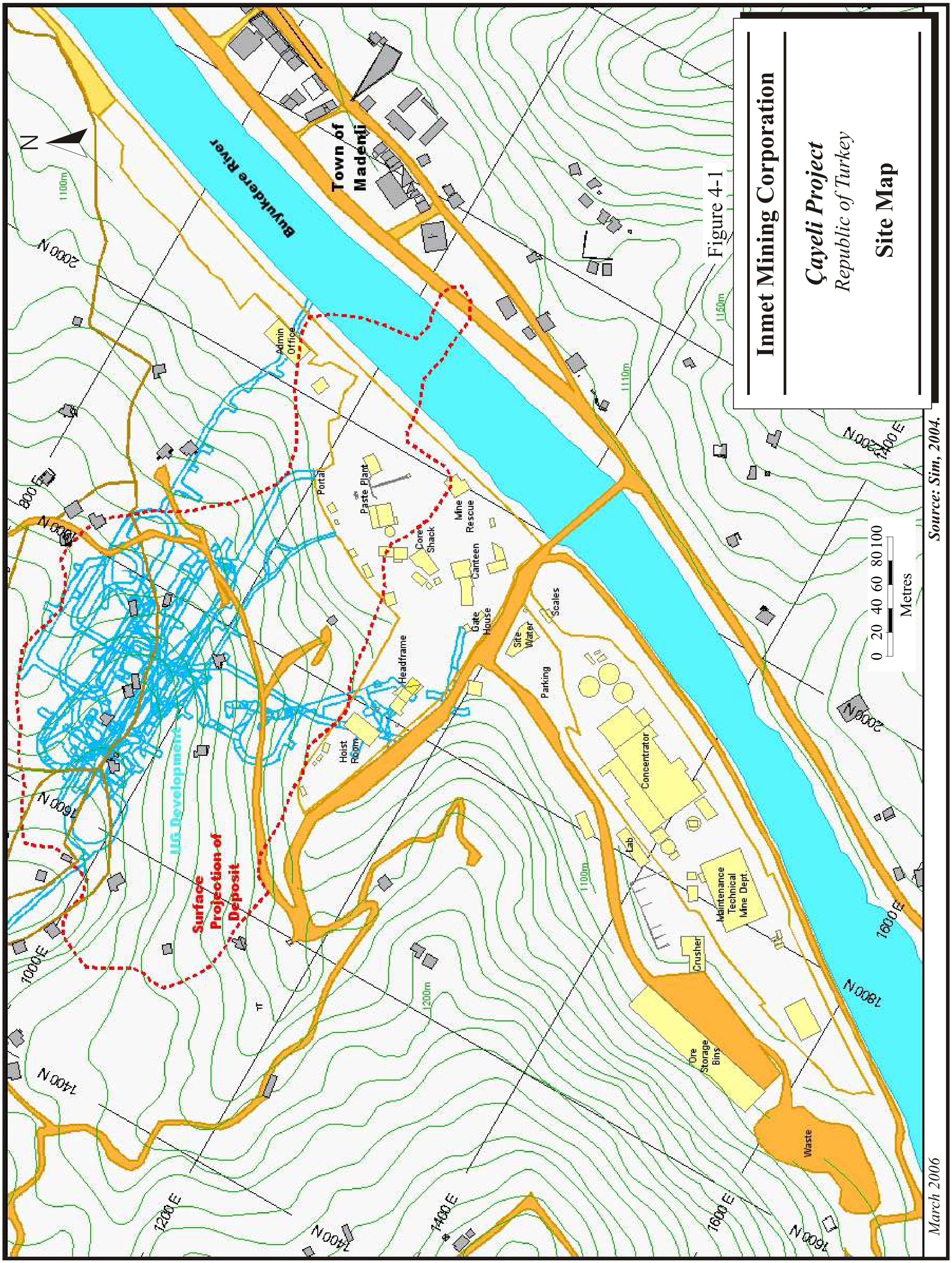


Figure 4-1

Inmet Mining Corporation

Çayeli Project

Republic of Turkey

Site Map

5 HISTORY

Mining activities along the Black Sea coast and at Çayeli date back at least a thousand years. At the turn of this century, minor exploration by Russians was reported, and, between 1930 and 1955, various shafts and adits were driven and some minor production took place.

The work which led to the present mine was started in 1967 by the Turkish Mineral Research and Exploration Institute (MTA). MTA carried out a geophysical survey and drilling program, and drove an adit into massive sulphide ore located just south of the current deposit.

In 1981, CBI was formed as a company, with Etibank (now Eti Holding AŞ), Phelps Dodge, and Gama Endüstri AŞ as shareholders, to develop the orebody. Phelps Dodge sold its 45% share to Metall Mining (now Inmet Mining) in 1988. Further underground work and metallurgical testing were done in the period from 1988 to 1991. Positive results were obtained and a production decision was made. Site construction started in 1992, basic mine infrastructure commenced in 1993, and the first concentrate was produced in August 1994. The total capital cost of the operation was approximately $200M.

In 2004, Inmet purchased Eti Holding AŞ's 45% interest and became the owner (100%) of the total share.

The historical reserve and production figures are listed for comparison purposes in Table 5-1. The classification parameters for proven and probable reserves (as listed in the table) are considered to be consistent with National Instrument 43-101.

TABLE 5-1 HISTORICAL RESERVES AND PRODUCTION

Inmet Mining Corporation – Çayeli Mine

	2005	2004	2003	2002	2001	2000	1999	1998	1997	1996	1995	1994	1993*
Proven + Probable Reserves													
Tonnes (million)	11.6	14.3	15.9	16.0	16.9	14.6	11.8	11.0	11.4	10.7	11.3	12.7	10.6
Cu %	3.7	3.4	3.6	3.6	3.8	4.1	4.2	4.8	4.3	4.4	4.5	4.4	4.7
Zn %	5.9	5.3	5.6	5.7	5.8	6.1	5.6	5.5	5.2	5.1	6.3	6.2	7.3
Additional Resources													
Tonnes (million)	4.6	3.7	3.2	3.3	2.9	1.3	4.3	5.9	6.5	6.5	6.5	4.1	N/a
Cu %	3.1	4.4	3.8	5.8	4.9	4.9	3.9	3.5	4.0	3.6	3.6	4.1	N/a
Zn %	2.8	4.8	5.9	8.7	6.9	8.6	8.0	8.3	8.0	7.8	7.8	8.4	N/a
Production													
Cut-off (EqCu%)	3.4	3.4	3.4	3.4	3.4	5.0	5.0	5.0	5.0	3.0	3.0	3.0	
Tonnes (million)	0.83	0.76	0.93	0.90	0.82	0.86	0.90	0.71	0.76	0.66	0.49	0.14*	
Cu %	3.8	3.9	4.2	4.2	4.6	4.9	5.1	4.6	4.7	3.6	3.3	4.4	
Zn %	6.7	5.8	5.1	5.1	4.5	4.5	5.3	6.6	7.0	8.5	7.2	8.1	

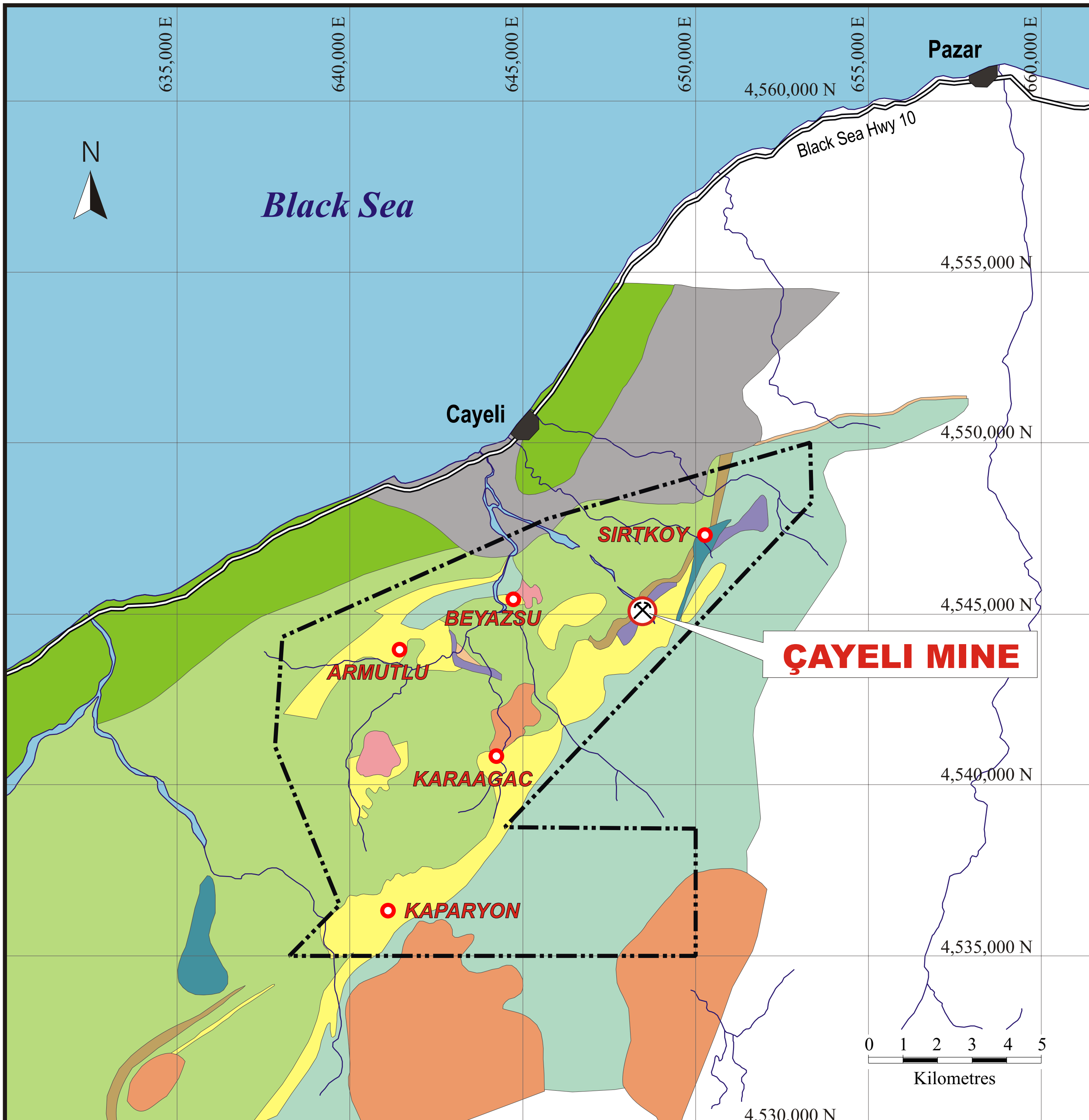
Notes: Reserves and resources are reported at a $US 46 NSR/tonne cut-off for 2005 and at a 2.5% EqCu for other years.
(*) Includes 67,000 tonnes used to commission the mill extracted during 1993.

6 GEOLOGICAL SETTING

REGIONAL GEOLOGY

The Eastern Black Sea volcanic province (Pontid Belt) is composed of three major geologic units: (i) a Precambrian-Palaeozoic crystalline basement complex, (ii) a Jurassic-Pliocene volcanic-sedimentary series, and (iii) a granitic-granodioritic complex of Cretaceous - Oligocene age that has intruded the crystalline basement and part of the volcanic-sedimentary series (Cağatay & Boyle 1980) (Figure 6-1).

The Pontid volcanic belt, which hosts the Çayeli deposit, is thought to be part of a large island arc system that developed during the late Jurassic- Eocene. It is generally accepted that two complete basalt-andesite-dacite-rhyodacite volcanic cycles evolved during the formation of the Pontid arc. The orebody is located at the contact between the Lower Acidic and Upper Basic Series.



Legend:

- *Volcanic Conglomerate*
- *Basalt - Pyroxene Porphyritic*
- *Upper Basic Series*
- *Volcano Sedimentary Series*
- *Purple Hematized Dacite*
- *Dacite Series*
- *Lower Basic Series*
- *Granite*
- *Felsic Intrusion*
- *Mafic Intrusion*
- *Mineralized Occurrence*
- *CAYELI-RIZE Property Boundary (approx.)*

Figure 6-1

Inmet Mining Corporation

Çayeli Mine

Republic of Turkey

Regional Geology

March 2006

Source: Sim, 2004.

PROPERTY GEOLOGY

The Çayeli orebody is located between hangingwall pyroclastites and flows (Upper Basic rock series) and footwall rhyolite (Lower Acidic Mine Rhyolite). The hangingwall rock sequence is a series of intercalated acid to intermediate pyroclastic and basaltic layers with some minor carbonate beds (Figure 6-2). The footwall rock sequence consists of rhyolite and felsic pyroclastic rocks. Hydrothermal alteration related to the formation of the deposit is restricted to the footwall stratigraphy in the form of clay (argillite) and chlorite. The footwall also hosts an extensive stockwork zone consisting of sulphide veins (chalcopyrite and pyrite) and silicification of varying degrees.

Mineralization, which is typical of a volcanogenic massive sulphide environment, is known over a strike length of 920 m. The measured resource has a strike length of approximately 600 m, a vertical depth of over 600 m and varies in thickness from a few metres to 80 m, with a mean of approximately 20 m. The average dip is 65° NNW in the upper part of the deposit and approximately 50° at depth.

A prominent fault with a strike subparallel to the deposit but dipping 50° ESE separates the upper "Main" zone of the deposit from the lower "Deep Ore" zone (Figures 6-3, 6-4). This structure is referred to as the "Scissor" Fault due to its variable displacement, with essentially no displacement in the southern parts of the deposit and up to 80 m of reverse displacement at the northern limits of the orebody. There is evidence that this is a synvolcanic structure that acted as an escarpment during the formation of the deposit and remained mildly active for a period after the development of the orebody. The Scissor Fault does not present any gauge along its plane.

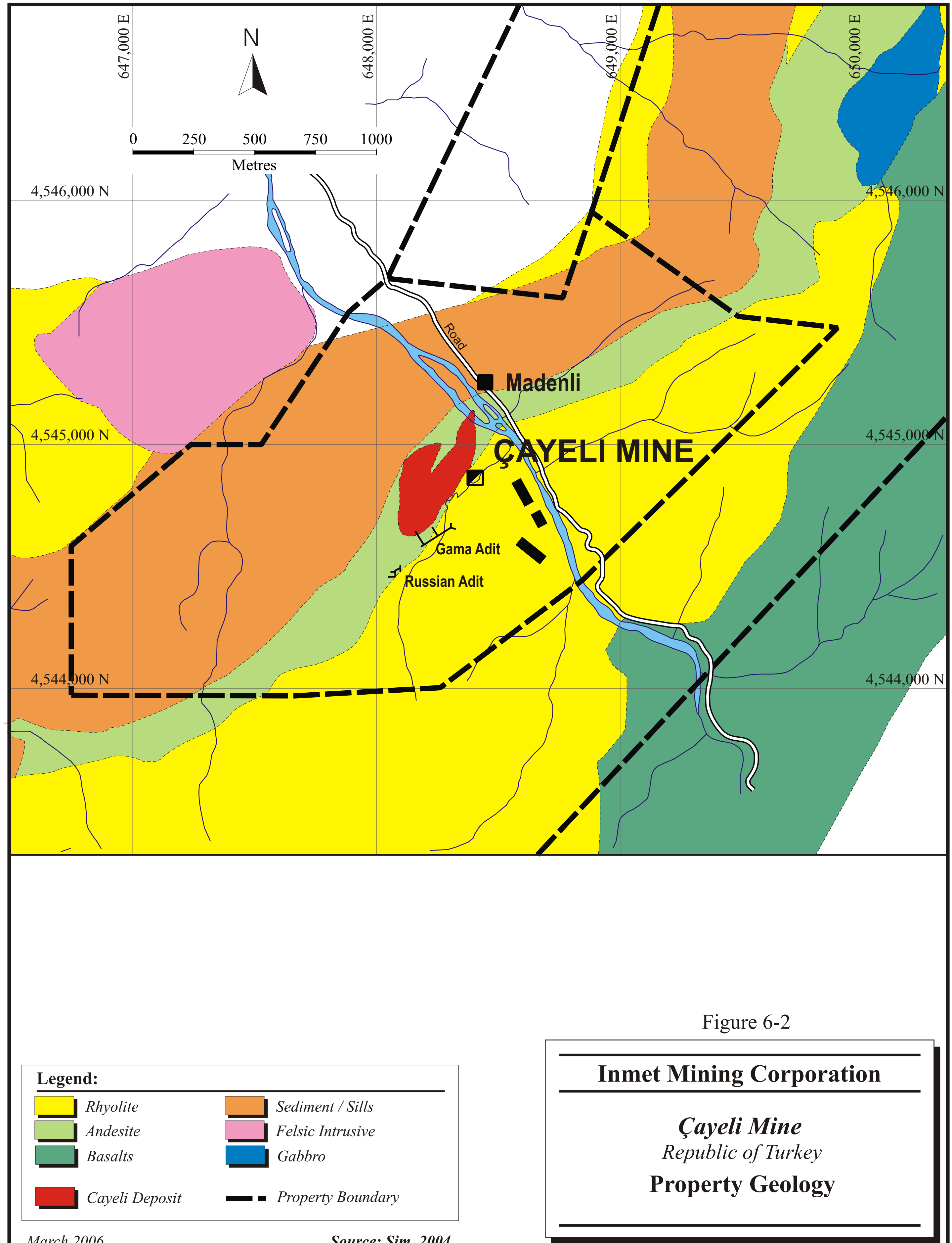


Figure 6-2

Inmet Mining Corporation

Çayeli Mine
Republic of Turkey
Property Geology

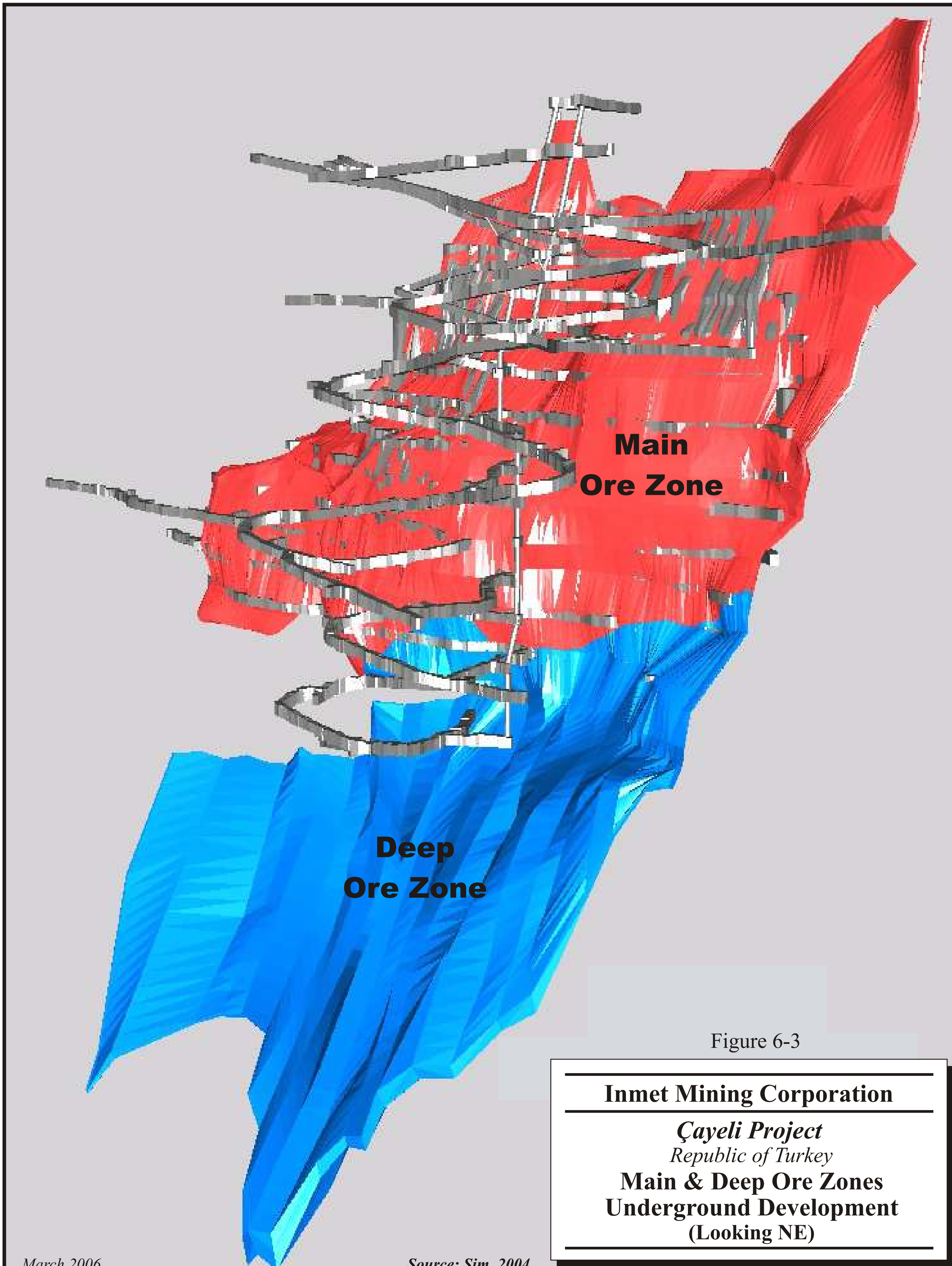


Figure 6-3

Inmet Mining Corporation

Çayeli Project

Republic of Turkey

Main & Deep Ore Zones Underground Development (Looking NE)

March 2006

Source: Sim, 2004.

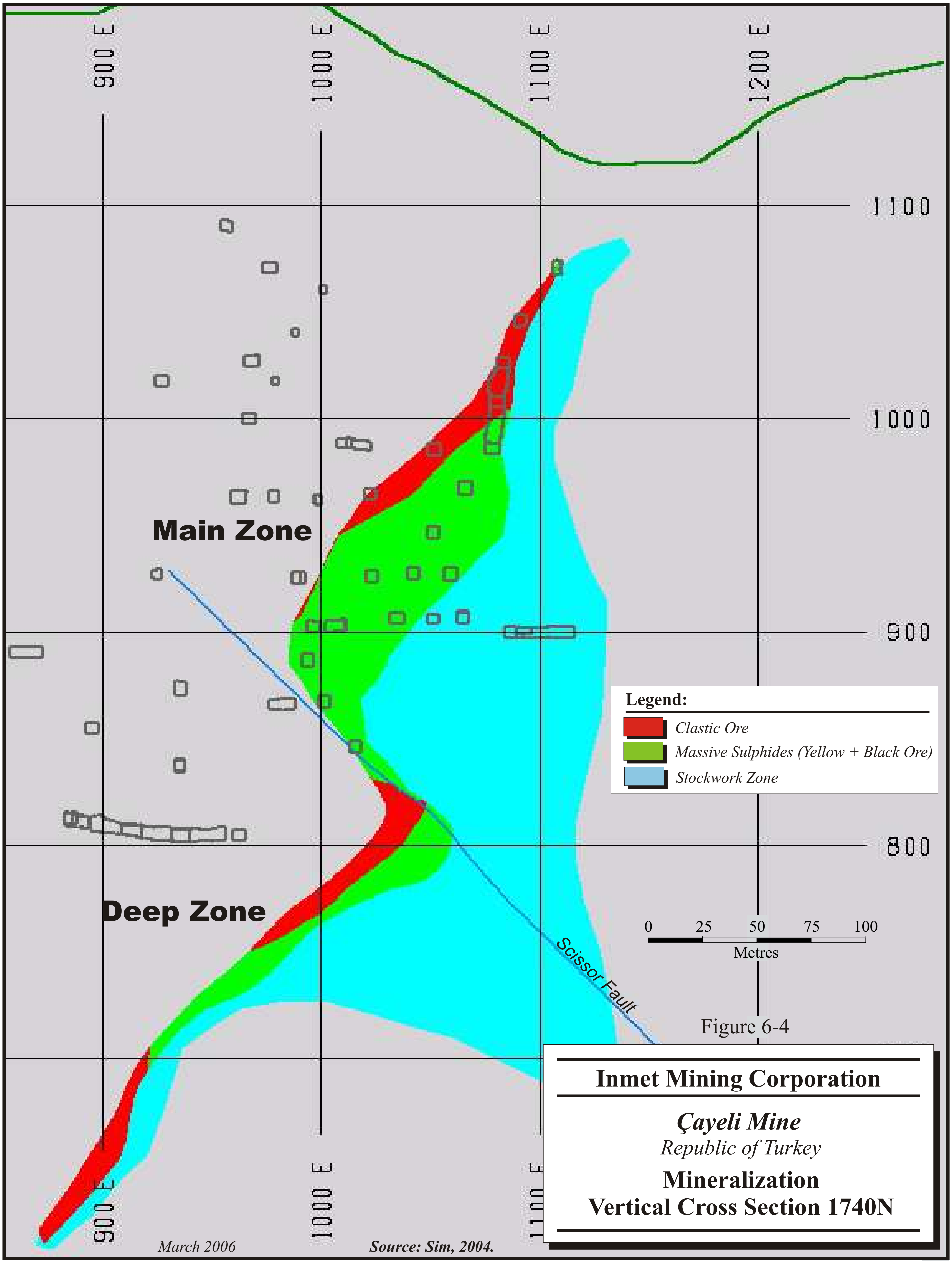


Figure 6-4

Inmet Mining Corporation

Çayeli Mine

Republic of Turkey

Mineralization
Vertical Cross Section 1740N

7 DEPOSIT TYPES

The Çayeli deposit is a volcanogenic massive sulphide deposit with many similarities to bodies of the Kuroko type.

8 MINERALIZATION

The sulphide composition and grade distribution can be highly variable throughout the deposit indicating an active depositional environment with multiple sources (vent points) for hydrothermal activity. Massive sulphides form the majority of the ore in the deposit. The ore comprises varying amounts of pyrite, chalcopyrite, and sphalerite with minor dolomite and barite. The stratigraphic top of the deposit (the hangingwall side) often shows fragmental textures indicating sloughing of the primary sulphides, an ore type that is referred to as "Clastic".

Below the massive sulphides, there are varying widths of stringer mineralization which can often exceed the cut-off grade due to chalcopyrite veining. This is predominantly copper mineralization represented by a series of veins and referred to as "Stockwork". The upper portion commonly shows pyrite and clay with veins of chalcopyrite which grade into more typical veins of chalcopyrite and pyrite in siliceous rhyolite.

Based on the requirements of the concentrator, the mine carries out segregation of three ore types. Clastic ore is material which, based on visual estimates, contains more than 10% sphalerite fragments in the massive sulphides. These fragments are generally fine grained and have intergrowths of chalcopyrite in the sphalerite which can affect the copper recoveries. The relative amount of Clastic ore is higher in the Deep ore, 40% compared to 11% in the Main ore zone, with an overall average of 19%.

The other two ore types, called Black and Yellow, are based on the contained zinc grade and are mined separately in order to allow for optimal grade blending from the stockpile bins on surface. These two ore types are referred as Spec ore. Black ore comprises 33% of the resource and is defined as material with more than 4.5% Zn and a Cu/Zn ratio of less than one. Approximately 48% of the deposit is Yellow ore which is comprised of approximately one half massive sulphides and one half stockwork material (stockwork mineralization generally contains very little sphalerite). Yellow ore consists of copper-rich, zinc-poor sulphides.

9 EXPLORATION

Çayeli is a relatively "mature" mine having been in operation for 11 years. However, timing and accessibility constraints for exploration drilling have contributed to the fact that it remains potentially "open" at depth and towards the south. Exploration drilling for extensions to the deposit is ongoing.

Considering the history of this deposit, exploration activity in the surrounding area is very limited. Efforts were initiated in 2002 to begin exploring for satellite sulphide deposits proximal to the Çayeli deposit. CBI acquired a large property holding in 2003 which covers approximately 15 km along the strike of the stratigraphic horizon that hosts the deposit. A surface exploration work was conducted during 2004 where approximately 1800 m core drilling and DEEPEM ground geophysics were carried out. Geologic mapping, geochemistry, geophysics, and diamond drilling will be continued over the next years. Exploration history is presented in Table 9-1.

In 2006 the exploration plan is to drill eight holes (3,000 m) and complete four DEEPEM loops. Drilling will test Çayeli extensions (deep, down dip, and plunge), Kaynarca and Çayeli SW (Incearap). The loops are planned for Beyazsu, Armutlu East, Orta, and Sarisu (Figures 9-1, 9-2). The location of prospects relative to the mine site is listed below.

- Russian adit: 150 m south of the orebody
- Kaynarca: one km NE of the mine
- Tuysuzler: 2.5 km NE of the mine
- Sarisu 1.5 km NW of the mine
- Sirtkoy 3.5 km NE of the mine
- Incearap 1.5 km SW of the mine
- Armutlu 6 km W of the mine
- Oksemeli 10 km SW of the mine
- Kaparyon 8-10 km SW of the mine
- Beyazsu 3.5 km NW of the mine
- Orta four km SW of the mine

In RPA's opinion, the Çayeli property has very good exploration potential, both on surface and underground. CBI retains large land holdings surrounding the mine, which host almost 15 km of the stratigraphic strike equivalent to the Çayeli ore horizon. Most of this ground remains essentially unexplored at this time.

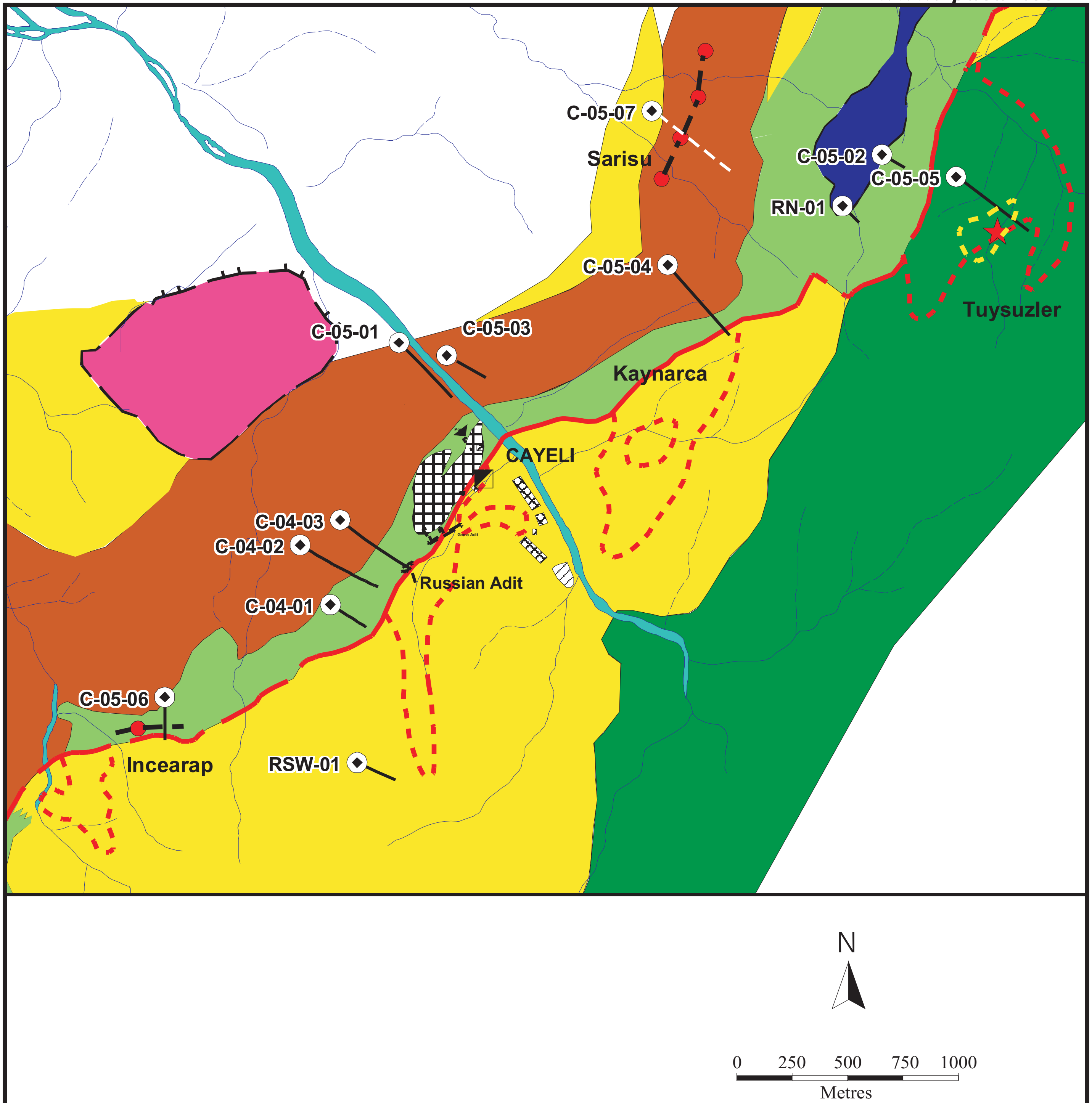
TABLE 9-1 SURFACE EXPLORATION SUMMARY - Cayeli-Rize Properties

YEAR	COMPANY	PROSPECTS	LOCATION	WORK	DESCRIPTION	RESULTS	COSTS	COMMENTS
1989	Cominco	Regional	property wide	Silt sampling	1,939 samples			
1990	Cominco	Cayeli	10 km strike of footwall stratigraphy	Soil Sampling	3,807 samples	Several anomalous areas identified		
1991	Cominco	Beyazsu		Drilling	1 DDH 206 m	0.22% Zn / 201m		
1992	CBI	Cayeli	Russian Adit	Drilling	1 DDH 340 m			also 8 u/g holes 114m
1993	Cominco	Incearap		UTEM	1 Loop - 12.4 line km			
1993	Cominco	Sirtkoy, Armutlu		Drilling	2 RC holes - 126 m	No significant results		
1993	Cominco	Kaparyon Tepe	10 km SW of Cayeli	UTEM	3 Loops - 32.3 line km			
1996	CBI	Cayeli, Kaynarca	Mine property	Rock Geochem	92 samples			Vital Pearson Phd
2000	Noranda	Tuysuzler, Armutlu, Sarisu		TEM	4 loops - 23.9 line km	Deep conductor at Sarisu	US$ 3,786/line km	
2000	Noranda	Tuysuzler, Cayeli		Rock Geochem	45 samples			
2001	Noranda	Armutlu, Cayeli SW		Drilling	4 DDHs - 1,125 m	7.2%Cu/0.45m (Armutlu)	$US 163/m	Copper stringers
2001	Noranda	Armutlu, Cayeli SW		BHPEM	4 holes		$10,859/hole	
2003	Inmet	Kaynarca, Tuysuzler	5 km strike of Mine footwall	Rock Geochem	172 samples	3 alteration pipes defined		
2004	Inmet	Cayeli SW, Kaynarca		DEEPEM	2 Loops - 14.1 line km		$US 2,674/line km	
2004	Inmet	Cayeli SW	Russian adit downdip	Drilling	3 DDHs - 1,775 m	3.3%Cu/4.5m	$US 92/m	Copper stringers
2004	Inmet	Cayeli SW	Russian adit downdip	BHPEM	1 hole			
2005	Inmet	Oksemeli, Beyazsu, Orta		Rock Geochem	86 samples			
2005	Inmet	Tuysuzler, Sirtkoy,Oksemeli		DEEPEM	4 Loops - 25.7 km		$US 8,500/km (est.)	
2005	Inmet	Cayeli Deep, Kaynarca, Tuysuzler		Drilling	6 DDHs - 2,865 m	No significant results	$US 165/m (est.)	
2005	Inmet	Kaynarca, Tuysuzler		BHPEM	5 holes			

1989-2005 TOTALS:		
	DDHs	**15 holes - 6,311m**
	RC holes	**2 holes - 126m**
	EM Loops	**14 Loops - 108.4kms**
	BHPEM	**6 holes**
	Rock Geochem	**395 samples**
	Soil Geochem	**3,807 samples**
	Stream seds	**1,939 samples**

* drilling costs are contractor payment only and do not include salaries, geochem, reclamation and drill site rental payment

*EM surveys are 200m line separation



Legend:

- *Alteration Pipe*
- *Surface DDH*
- *EM Conductor*

Figure 9-1

Inmet Mining Corporation

Çayeli Mine

Republic of Turkey

Exploration Drilling Plan

March 2006

Source: Sim, 2004.

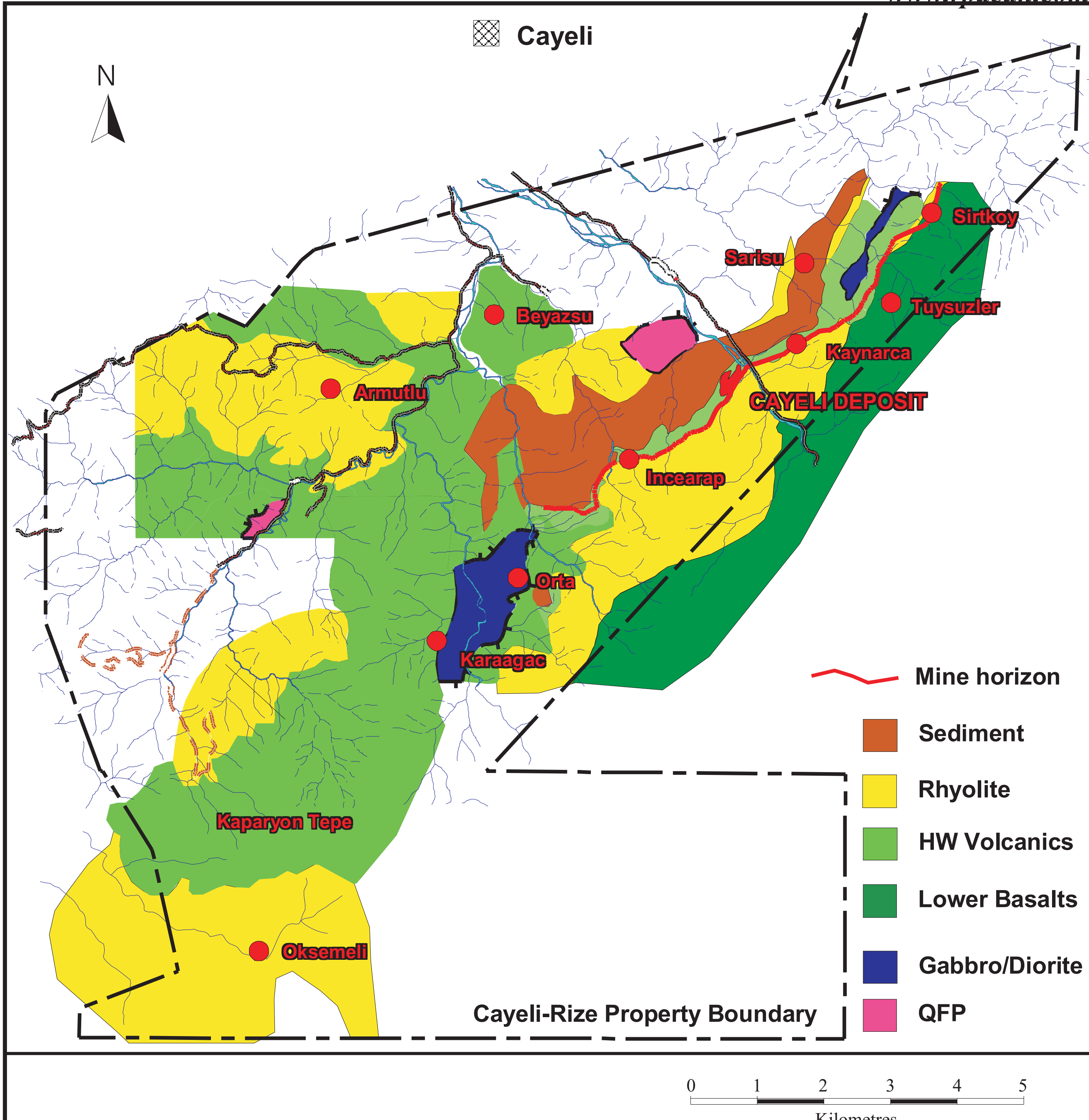


Figure 9-2

Inmet Mining Corporation

Çayeli Mine

Republic of Turkey

Location of Exploration Prospects

March 2006

Source: Sim, 2004.

10 DRILLING

Diamond drill holes are planned (azimuth, dip, length) by geologists on vertical cross sections, level plans, and vertical longitudinal sections. These planned holes are checked on screen in 3D. Drill lines are marked underground (front sight and back sight) by the mine surveyors.

Hole deviations (azimuth and dip) are measured with Electromagnetic survey instruments approximately every nine to twelve metres. In addition, dip angles are measured at intervals varying from nine to twelve metres using the same instruments. Electromagnetic survey instruments provide accuracy better than +/- one degree. Once a hole is completed, collars are surveyed by mine surveyors.

The core is placed in sequential order in core boxes labelled with hole number. Each run is identified by a wood block on which the depth of the hole is marked. Missing (not recovered) core is identified by a wood stick indicating the length of the missing section. At the end of each shift, core boxes are transported to surface by drillers and placed on tables at the core shack.

Upon receipt of core boxes, core is washed and verified for accuracy. Once done, RQD, core recovery measurements, and meter marking are carried out for the entire length of the holes. Boreholes are photographed digitally by geology technicians and information is stored in database. Then a geologist describes the core geology and enters geological and structural data into a digital logging package. Core logging is carried out in English.

RPA considers the drilling protocol at Çayeli to be consistent with industry standards.

DELINEATION AND STOPE DEFINITION DIAMOND DRILLING

There are two levels of detail when defining the deposit through diamond drilling. Initial Delineation drilling involves holes collared from the hangingwall development and designed to intersect the deposit at a nominal spacing between 20 m and 40 m. Pierce angles of these holes with the ore horizon range from 90° to 45°. A second stage of drilling, called Stope Definition drilling, was initiated during 2001 and involves drilling horizontal holes down the centre of each planned sill drift. This results in holes spaced on approximately seven metre centres on each 20 m vertical sublevel.

DRILL CORE RECOVERY VERSUS GRADES

The overall drill hole recoveries range from zero to 100% with an average of 80%. Poor recoveries occur in some of the hangingwall rocks where there is intense chlorite developed. In the ore zone (i.e., stratigraphically below the hangingwall contact), low recoveries tend to occur in areas of friable sulphides. It is standard practice during drilling in the ore zone to collect the cuttings in a bucket. If core is not recovered for a specific run, the cuttings are placed in the core box for that interval.

A total of 10% of the diamond drill hole database in the ore zone is comprised of samples with zero core recoveries or samples derived through the collection of cuttings during drilling. Analysis carried out by Çayeli geology staff indicates that, overall, samples derived from drill cuttings are 20% to 25% lower in average copper and zinc grades than core samples. Geology staff concluded that this apparent bias may be due to such factors as segregation inside the hole during drilling or sampling problems that might have occurred during the collection of the cuttings. Geology staff concluded it may also be due to the fact that the lower grade sulphides are less competent.

In order to quantify the potential effects of these lower grade drill hole cutting samples on the block model, simple resource estimation comparisons were conducted by geology staff using the core and core+cuttings data sets. It was found that the model estimate including the drill hole cutting samples was only 2% to 3% lower in copper and zinc grades when compared to the drill core-only estimation. Geology staff concluded that there is no spatial bias in the data set and that the overall effects of the cuttings data on the model are relatively minor.

Statistical evaluation of the drill hole database carried out by geology staff indicates that the core samples do not show any apparent relationship or bias between core recovery and copper grades. However, higher zinc grades tend to show higher core recoveries, a feature of the more competent Clastic ore zone.

The drill hole cutting samples have been retained in the database for estimation purposes, and no modifications were made to the values prior to interpolation.

RPA agrees on the above methodology of assigning grades to low recovery core. RPA found results to be on the conservative side when used for resource estimation.

RQD DATABASE

Prior to any logging, the rock quality is measured and calculated. The RQD (Rock Quality Designation) values are kept as paper records. They are not currently in a database. RPA recommends entering RQD values into a database. Such a database, in combination with rock types, ore types alteration zones, and micro-seismic data, may be very useful for development heading planning, stope design, and ground support.

11 SAMPLING METHOD AND APPROACH

The resource model has been developed using data from three sources - diamond drill holes, underground channel samples, and "sludge" samples (Figure 11-1), which are collected from the cuttings produced while drilling off a development heading using a jumbo. The details concerning these three types of sampling are outlined below.

Sample selection is done by geologists in the case of diamond drill holes and by beat technicians in the case of channel, sludge, and broken ore sampling.

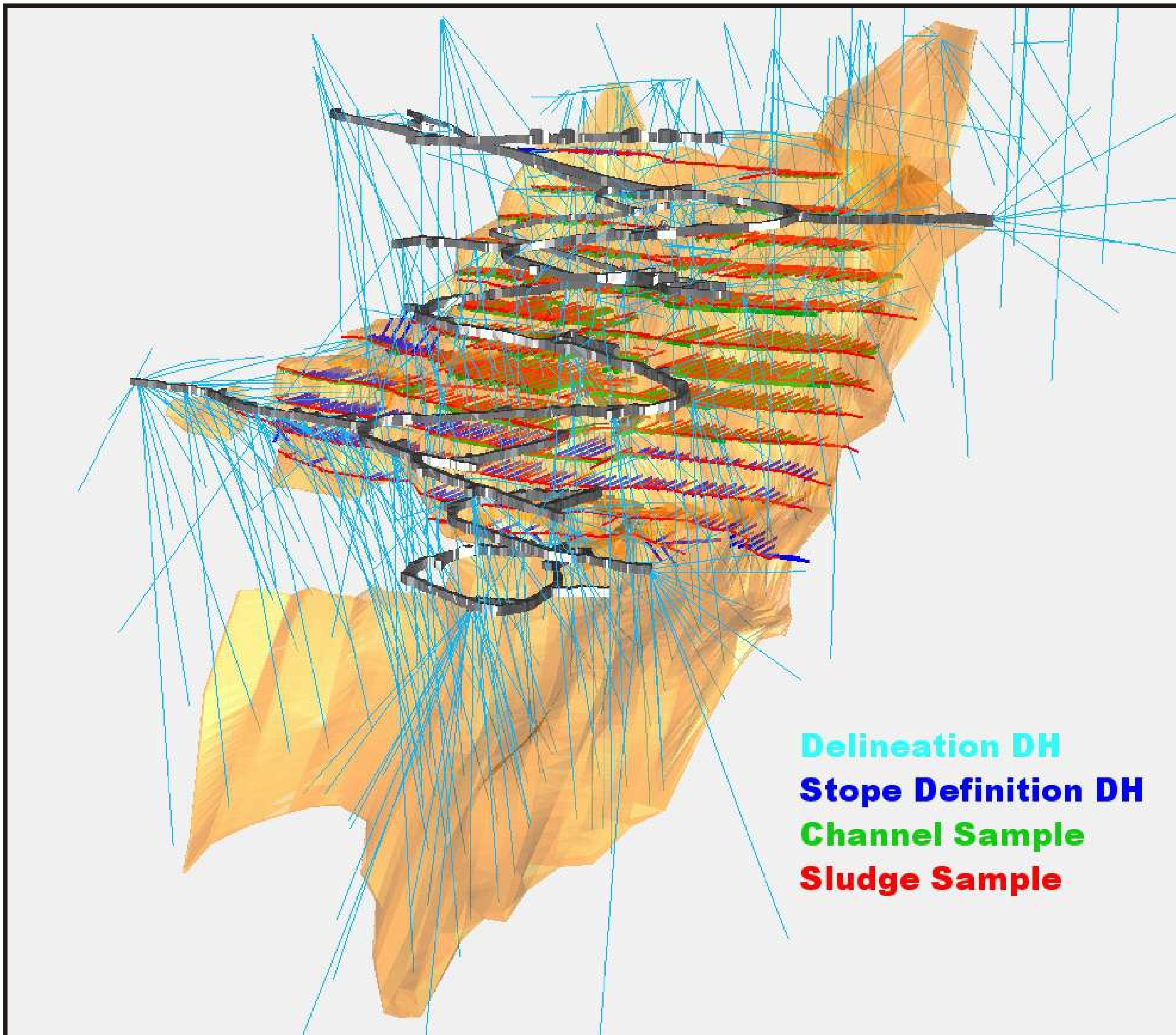


Figure 11-1

Inmet Mining Corporation

Çayeli Mine

Republic of Turkey

Distribution of Sampling Type

March 2006

Source: Sim, 2004.

DIAMOND DRILL SAMPLES

Almost all delineation drill holes are NQ size and the intersections have been sawed in half, with one half sent for analysis and the other retained for future uses. Sample intervals are variable, based on the geology and range from 0.1 m to a maximum of four metres (35% of samples are less than one metre in length and 61% are one metre to two metres in length).

The stope definition holes are BQ size and, following geological logging, the complete core is sent to the lab for analysis. The standard sample interval is two metres but can locally vary based on the geology of the hole. Geology staff carried out grade comparison between NQ and BQ core sizes and found that there are no indications of grade bias related to sample size.

UNMINERALIZED ZONES

Intervals which intersect unmineralized zones (such as dykes) are generally not sent for analysis. These are identified by the rock type codes in the database and are assigned default grades of zero.

MISSING SAMPLES

Intervals for which no core was recovered and no drill cuttings were collected are marked as “missing” in the database and have no effect on block grade estimates during interpolation.

UNDERGROUND CHANNEL SAMPLES

Underground channel samples were taken extensively in the mine up until 2001. They were then replaced with the stope definition drill hole since the latter improved the timing of the results for mining purposes and reduced sampling costs. The channel samples are retained in the database as they still contribute valuable information on active areas in the upper portions of the deposit.

Underground channel samples are obtained following the completion of primary sill drifts. A channel is cut using a diamond saw on both the north and south walls of the drift, and the material is removed with a chisel and placed in sample bags. The wall is logged and the sample intervals (from and to a known point) are recorded in a similar fashion to diamond drill core processing. In the database, each wall which is channel sampled is treated like an individual drill hole.

Geology staff carried out a statistical comparison in 1999 between diamond drilling and channel sample data for a given volume of data including a relatively equal amount of sampling of each type. There were no indications of significant bias between the two data sets.

DEVELOPMENT SLUDGE SAMPLES

Development sludge samples are obtained using a small shovel and scooping a total of five to seven kilograms of drill cuttings that accumulate along the base of a face during the jumbo drilling of each round in the ore zone. Each sample represents approximately four metres of advance in the drift. In the database, each individual drift is presented as a drill hole located about the centerline of the drift. A statistical analysis carried out by geology staff indicates that the sludge sample data produce slightly lower grades than channel or drill hole samples over a given area. It has been concluded that this trend is due to the difference in selectivity in the sample sizes and the related dilution effects at a cut-off limit.

Sludge samples have been collected since the beginning of production, but, historically, this information has only been utilized for visually aiding the daily grade control activities. In 2001, this data was converted to digital format and testing began for ways to better utilize this information. It was found that it could improve the short-term forecasting of stope production. During interpolation, the effect of the sludge data is limited to primarily higher-level (measured) resources within a distance of 15 m from the sample location. The sludge sample has no effect on the longer-term resources/reserves in the model.

SAMPLE DISTRIBUTION

The distribution of samples by type, as of the end of 2004, is listed in Table 11-1. Core samples account for 57% of total samples in the database.

TABLE 11-1 SAMPLE DATABASE STATISTICS
Inmet Mining Corporation – Çayeli Mine

Type	Number of Holes	Metres	Number of Samples	% Samples
Delineation Drill Holes	660	83,283	28,174	51.4
Stope Definition Drill Holes	247	7,130	3,256	5.9
Channel Samples	644	14,527	15,582	28.4
Sludge Samples	846	28,975	7,834	14.3
Total	**2,397**	**133,915**	**54,846**	**100**

RPA considers the quantity of samples in the database that are used for grade estimation appropriate and sufficient.

BROKEN ORE SAMPLES

Broken ore from stopes and from stockpiles in surface ore bins are sampled on a continuous basis, one 10 kg sample for every 250 tonnes of ore. Stope samples are taken from every tenth 25-tonne truck. Stope samples are used for grade reconciliation.

Stockpile samples are used to report daily production of mill feed. Stockpile grades are not used for grade reconciliation.

Broken ore samples are taken by geology technicians or geologists. They are all collected by hand shovel.

There are no broken ore samples taken in development headings.

RPA recommends conducting a more detailed study on broken ore sampling in order to see if sampling quantity is appropriate for grade reconciliation, or whether additional sampling is necessary. Maybe one sample per 250 tonnes is not enough.

DEVELOPMENT GRADE CONTROL

Grade control in development headings is based on:

- Visual estimates. Development faces are split in quadrants and the proportion of ore type (clastic, spec, stockwork) is estimated in each quadrant. Surface ore bin stockpiles become necessary when a bigger proportion of ore type is identified in the face.

- Sludge sample grades. Sludge samples are analyzed by X-ray Fluorescence (XRF) to provide rapid discrimination between ore and waste. Visually estimated grades and sludge sample grades are entered into the database to be compared by each of the geology technicians. Technicians are instructed about significant discrepancies between the visual estimates and sludge grades.

12 SAMPLE PREPARATION, ANALYSES AND SECURITY

SAMPLE PREPARATION – CORE SHACK

Following logging, individual drill core samples are collected by geology technicians in the core shack and placed in plastic sample bags. An Excel spreadsheet is initiated for each hole and contains hole number, sample number, sample intervals, geology codes, and density results. This spreadsheet is located on the computer network and can only be accessed by geology technicians (the core shack), the laboratory and geology staff. The final lab results are entered into this spreadsheet and the data is eventually verified and transferred into the main geology database for use in the resource estimates.

SAMPLE PREPARATION – MINE LABORATORY

CBI has operated an on-site assay laboratory since the prefeasibility work conducted in 1984. The assay laboratory continues to operate and process samples from the geology, mill, marketing, and environmental departments. It presently has no official certification and is not accredited. The laboratory staff consists of 11 people (chemists and technicians) working on day and night shifts. An average of 150 samples are prepared and analyzed on a daily basis. These samples consist of:

- 125 mill samples
- 25 geology samples (20 broken ore samples and five core samples)

The geology technicians deliver batches of samples to the mine laboratory on a regular basis. These are accompanied with a hard copy of the data sheet containing basic information such as the sample type and number sequence. Lab personnel initially crosschecks the sample bags with the submission sheet, prepares a work flow control sheet, and assigns a job number to each batch of samples.

Samples are dried for eight hours at 105ºC in a heavy-duty drying oven (note that there is a dedicated drying oven for rock samples). Samples are then run through a jaw crusher to minus one centimetre size and Cone Crusher to minus four millimetres. Depending on the size of the original sample, it may undergo some degree of splitting by coning and quartering and/or reduction through a riffle splitter. Rejects at this stage are stored in sealed plastic bags in the lab. Finally, a ring mill pulverizer (100 cc or 250 cc cups) is used to reduce the sample to 100% passing -100 mesh and 80% passing -200 mesh. The sample at this stage ranges between 250 g and 400 g in size.

SAMPLE ANALYSIS

A 10 g portion of the sample is collected, by squaring and quartering, to prepare a pressed tablet for XRF analysis. A second 100 g sample is collected, by squaring and quartering, for Atomic Absorption Spectrometry (AAS). The remaining material is retained for internal or external checks and for archive purposes. Note that XRF analysis is used as an initial check on samples, such as on-site samples, requiring quick turnaround results (i.e., surface stockpile samples). Only AAS results are utilized in resource estimations. XRF analysis turnaround is about two hours, while AAS turnaround is one day.

The XRF analyzer is a 1994 model produced by Outokumpu Instruments (X-Met 880 with Heps 2431 probe). The primary AAS analyzer is a Varian Spectr AA-250 (double beam with graphite furnace, 1994). A second Varian AA275 (1980) single beam model acts as a backup. Routine assays carried out by the mine laboratory include Cu, Zn, Pb, Fe, Ag, Cd, and Au.

GOLD ASSAYING VS CONTRIBUTION TO NSR VALUE

Gold is a relatively minor contributor to the NSR of the operation and, as a result, only the delineation drill hole samples are analyzed for Au. Therefore, the overall sample density for this element is much lower.

DENSITY

Prior to splitting the drill core for analysis, all delineation drill hole samples are measured for density. This is done using the immersion method whereby the sample is weighed in air and in water. Pieces of core measuring approximately 20 cm are weighed in air and while submerged in water. Density is then calculated as the Weight (air)/(Weight (air)-Weight (water).

RPA considers the method of density determination at Çayeli to be consistent with the industry standards.

Density determinations are not conducted for stope definition drill holes, channel samples, and any delineation drill holes in which sludge was recovered or in development sludge data. In these cases, average values were assigned based on the rock code designation (rock codes are defined during drill core logging or underground mapping). These values are listed in Table 12-1.

TABLE 12-1 DENSITY DETERMINATIONS BY ROCK TYPE
Inmet Mining Corporation – Çayeli Mine

Rock Type	% Samples with actual measurements	Mean Bulk Density (t/m3)
Black Ore	51%	3.98
Clastic Ore	66%	4.04
Yellow Ore	37%	4.07
Vein Ore	62%	3.91
MS/SMS Pyrite	71%	3.78
FW Rhyolites	33%	3.21
Other (dykes, volcanics)	0%	2.70

In the case of stope definition drill holes, channel samples, and delineation drill holes in which sludge is received, RPA recommends that density be determined by an interpolation technique like kriging rather than giving a density value based on rock codes. Density would then be based on more local values and not on average or regional values.

QA/QC

The mine laboratory conducts a series of internal and external checks to ensure the accuracy of their product. Those checks are, in general, requested by the laboratory manager.

All geology department samples undergo both XRF and AAS analyses. The results are compared and any significant discrepancies are investigated. Internally, 10% of all lab samples, from both coarse and pulverized fractions, are randomly replicated and the results are constantly monitored. Some samples from the previous day are reread. If doubt about assay results occurs, the laboratory manager asks for reassay. Results of these check assay programs are distributed to department heads on a regular basis.

Each batch of 25 samples contains:

- Five duplicates,
- Four in-house standards,
- One blank.

For external checks, composite samples (one kg) from broken ore samples and from mill are sent to Inspectorate Laboratories in England one or two times a year. Comparison between original assays and checks is very acceptable and well within a 5% difference, generally within a 1% difference. Concentrate assaying is, in general, lower by one percent at an outside laboratory. Approximately one percent of concentrate samples goes to an umpire laboratory for final concentrate grade settlement.

At least one in ten standard control samples are introduced into each series of samples submitted to the lab. Anomalous results, which are extremely rare, are investigated and sample batches are rerun as required. Overall, CBI’s results agree with all consulting lab results within a 0.3% (absolute) error.

In order to ensure that the standard samples have relative matrix components, the lab has been preparing its own standards since 1986. Çayeli has five different in-house standards. These standards are verified using as many as 10 different reputable consulting laboratories. The following laboratories have been used on a consulting basis by CBI:

- In the prefeasibility study period (1984 – 1987):
  - BERGSTROM & BAKER, South Africa
  - McLAHLAN LAZAR (now Inspectorate), South Africa
  - SKYLINE , USA
  - CU STATE , USA
  - MINTEK, South Africa
  - MTA, Turkey
- In the feasibility study period (1988 – 1992)
  - Sachtleben (Metallgesellschaft) Germany
  - XRAL Canada
  - MTA, Turkey
- In the production period (1994 – present)
  - XRAL, Canada certified, accredited, SGS
  - MTA, Turkey
  - SGS, England , certified, accredited
  - ALEX STEWART ASSAYERS, England, accredited, certified Lloyds, ISO 9002
  - ALFRED H. KNIGHT, England
  - INSPECTORATE GRIFFITHS, England, British Sta. Inst. Accredited, Certrified UKAS, ISO9002
  - WALKER and WHITE, USA
  - BACHELET, Belgium

Equipment calibration is carried out using only certified calibration standards.

CBI has been in operation for many years, and during this time there have not been any significant problems with respect to feed grades, metallurgical balance, or reconciliation. This is an indication that the underlying data (provided by the lab) is correct.

RPA has reviewed several check assay sheets and have found no major discrepancies between original and checked values. CBI reports that the lowest of the original and checked values is used for grade estimation and that there is no field for check assays in the database. RPA recommends that the average of the original and checked values be used for grade estimation and a field for check assays be provided for in the database.

RPA considers the sampling method and analysis at Çayeli to be consistent of industry standards. RPA has visited the mine laboratory and found the lab clean and well organized. All borehole certificates since May 1984 are in file cabinets. All of them are in a database. Splits of all samples that have been processed by the laboratory are in the sample archive and are accessible for inspection. RPA found the mine laboratory well managed and reliable.

RPA recommends that check assay programs carried out on geology samples should be more closely followed by geologists. Check assays should also be requested by geologists, not only by the laboratory manager. The geology department should set up a database for check assays that will contain sample type, original values, checked values, and dates. The geology department should be in a position to confirm that their data for resource estimates have been checked and are reliable. Check assays should be included in the Mineral Resource and Mineral Reserve Estimates Technical Report. RPA also recommends that the results be presented on XY graphs.

13 DATA VERIFICATION

SIM GEOLOGICAL VERIFICATION

In January 2005, a series of randomly selected holes representing 10% of the database was verified by Mr. Rob Sim from Sim Geological. Mr. Sim was Chief-Geologist at Çayeli from 2002 to 2004. The verification included database versus original source comparisons between assay results, survey data, and geology information. No significant problems were identified. RPA has not verified the results of this work.

CORE SHACK VISIT

RPA examined the core shack during the site visit and found it to be efficient and well organized. Samples were individually wrapped in closed plastic bags, with assay tickets inside. Samples were placed in order for transport to the mine laboratory. There were no significant delays in core logging.

CROSS SECTIONS, LONGITUDINAL SECTIONS, PLAN VIEWS, CORE LOGS, AND DATABASE

RPA reviewed cross sections, longitudinal sections, and plan views, and found the interpretation of the mineralization to be well done. RPA carried out some spot checks in the database and did not find any significant errors.

14 ADJACENT PROPERTIES

CBI is currently waiting for the legal permissions from the government to start development of the Cerattepe massive sulphide deposit located near Artvin, approximately 150 km east of Çayeli. This deposit is being considered as a potential source for additional high-grade mill feed at CBI.

RPA has been unable to verify this information, and the information reported is not necessarily indicative of the mineralization on the property that is the subject of the technical report.

15 MINERAL PROCESSING AND METALLURGICAL TESTING

Details on mineral processing and metallurgical testing can be found in Item 17 Other Relevant Data and Information of this report.

16 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

SUMMARY

Mineral resource and mineral reserve estimates as of December 31, 2005, for the Çayeli Mine are summarized in Tables 16-1 and 16-2.

TABLE 16-1 MINERAL RESOURCES
Inmet Mining Corporation – Çayeli Mine

Category	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	$NSR/t
Measured	1.91	3.08	3.09	22	0.50	0.18	57
Indicated	2.65	3.05	2.55	20	0.44	0.15	54
Meas.+ Ind.	**4.56**	**3.06**	**2.78**	**21**	**0.46**	**0.16**	**55**
Inferred							
Çayeli	0.62	3.38	2.85	18	0.26	0.12	58
Russian Adit	0.45	3.29	11.08	N/A	N/A	N/A	N/A
Total Inferred	**1.07**	**3.34**	**6.29**	**N/A**	**N/A**	**N/A**	**N/A**

Notes:
1. CIM definitions were followed for mineral resources.
2. Mineral resources are estimated at a cut-off grade of US$35 NSR/tonne of ore.
3. Mineral resources are estimated using prices of US$1.10 per pound Cu, US$0.55 per pound Zn, US$5.60 per ounce Ag, $US450 per ounce Au.
4. Measured and indicated mineral resources are exclusive of mineral reserves.
5. Russian Adit inferred resource: Au and Ag have never been assayed.

TABLE 16-2 MINERAL RESERVES
Inmet Mining Corporation – Çayeli Mine

Category	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	$NSR/t
Proven	4.70	3.77	5.85	44	0.59	0.30	73
Probable	6.90	3.57	5.88	52	0.53	0.36	69
Total	**11.60**	**3.65**	**5.87**	**49**	**0.56**	**0.34**	**70**

Notes:
1. CIM definitions were followed for mineral reserves.
2. Mineral resources are estimated at a cut off grade of US$35 NSR/tonne of ore.
3. Mineral resources are estimated using prices of US$1.10 per pound Cu, US$0.55 per pound Zn, US$5.60 per ounce Ag, $US450 per ounce Au.

MINERAL RESOURCES

Mineral resource and mineral reserve estimates are based on all available drilling, sampling, and mapping data as of December 31, 2005. Statistics, variography, and capping factors described below are derived from available data as of December 31, 2004. It is RPA's opinion that the data from 6,070 m of diamond drilling added in 2005 will not change significantly the results and interpretation. The resource and reserve estimations were completed using the MineSight software system developed by Mintec Inc. of Tucson, Arizona.

GEOLOGIC MODEL

The geologic model is constantly being modified as new information is gained through drilling and development. The deposit was divided into three distinct geologic domains based primarily on the style of mineralization. Each of these domains is further divided by the Scissor fault which separates the Main and Deep Ore zones (Appendix 1, Figures 23-1 to 23-10). The domains are:

- Clastic ore domain.
- "Spec" ore domain (which includes all massive sulphides – Yellow and Black Ore - excluding the Clastic ore portion).

- Stockwork ore zone. Areas in the footwall comprised of massive pyrite and veins of chalcopyrite have been included in the Stockwork domain due to the vein nature of the copper mineralization.

The geological interpretation of all material above and including the 820 m level has been conducted on level plans on 20 m intervals. Information is derived from underground geologic mapping and diamond drilling/channel sampling interpretation.

The geologic interpretation and model of the majority of the Deep Ore zone, below the 820 m level, was based on a series of vertical E-W cross sections spaced at 20 m intervals, and used primarily the diamond drilling information.

Three-dimensional wireframes were developed for all domains and tested for any gaps or overlaps. Wireframes were developed using raw data.

Contact profiles, which help evaluate the changes in grade across domain boundaries, are shown for copper and zinc distributions in Figure 16-1.

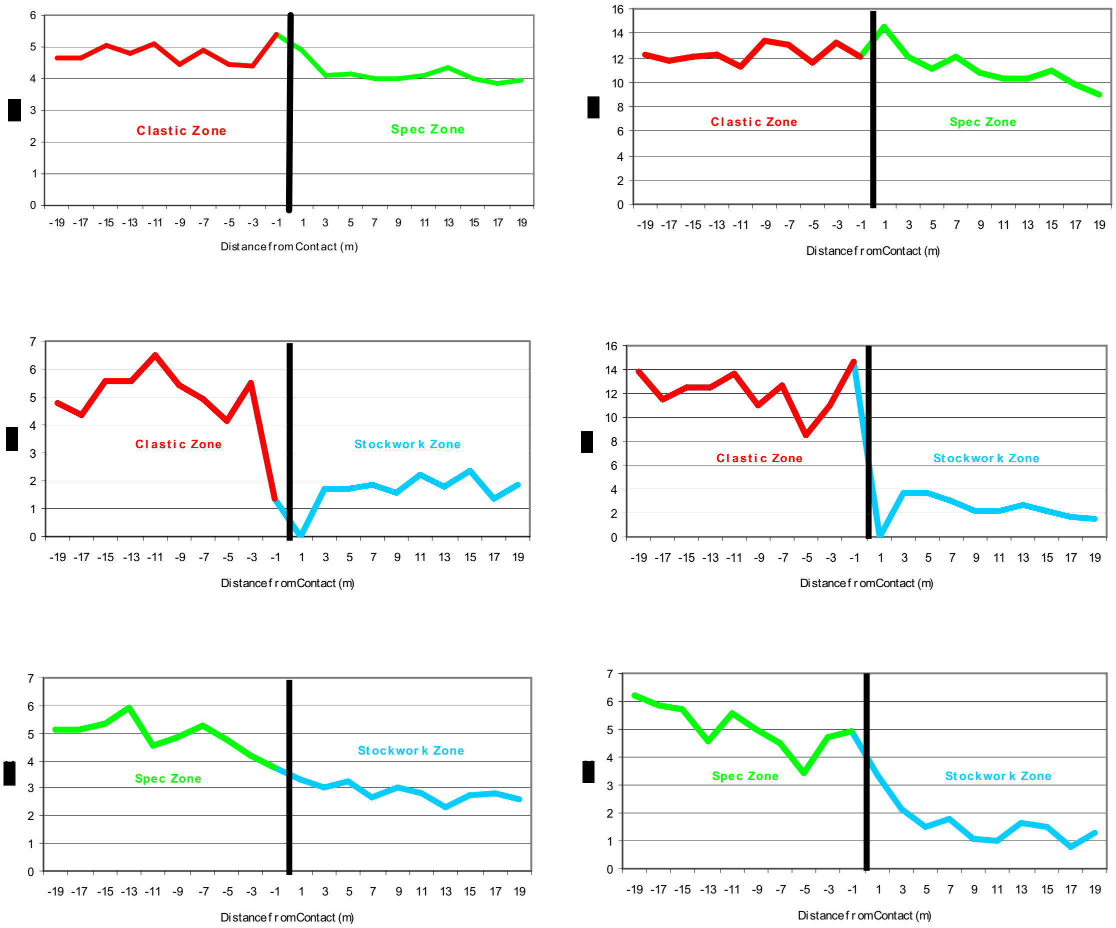
Although the two units are geologically quite distinct, there is little difference in grade between the Clastic and Spec zones evident in the contact profiles. Conversely, the contrast between the high-grade Clastic and generally low-grade Stockwork zones is very evident in the contact profiles. Finally, the Spec–Stockwork contact is somewhat gradational for copper and rather abrupt for zinc grades.

The final geological model contains the following six domains:

- Zone 1: Main Zone Clastic Ore (MOCO)
- Zone 2: Main Zone Spec Ore (MOSpec)
- Zone 3: Main Zone Stockwork Ore (MOStwk)
- Zone 4: Deep Ore Zone Clastic Ore (DOCO)
- Zone 5: Deep Ore Zone Spec Ore (DPSpec)
- Zone 6: Deep Ore Zone Stockwork Ore (DOStwk)

FIGURE 16-1 CONTACT PROFILES ACROSS DOMAINS FOR CU AND ZN

(Modified from Sim 2004)



ASSAY STATISTICS BY DOMAIN

The drill hole, channel, and sludge samples in the database were "tagged" with an individual wireframe domain that ensured that all sample intervals within each zone were identified as belonging to that specific zone. The basic statistics of the original assay data by zone are listed in Table 16-3.

TABLE 16-3 ORIGINAL ASSAYS - STATISTICS BY ZONE
Inmet Mining Corporation – Çayeli Mine

Zone	Metres	Cu%	Zn%	Ag g/t	Au g/t	Pb%	EqCu%
1	6,061	5.24	12.99	136	1.83	0.83	10.05
2	34,909	4.82	6.17	41	0.84	0.27	7.10
3	5,509	4.62	1.86	14	0.37	0.09	5.33
4	1,041	5.20	9.60	127	1.02	0.74	8.75
5	1,917	4.09	9.45	71	0.67	0.52	7.59
6	544	4.28	2.93	19	0.28	0.12	5.36
Total	**49,980**	**4.82**	**6.68**	**52**	**0.90**	**0.34**	**7.30**

Notes
1. Drilling as of December 31, 2004
2. Statistics at 2.5% EqCu Cut-Off
3. (Zones: 1=MOCO, 2=MOSpec, 3=MOStwk, 4=DOCO, 5=DOSpec, 6=DOStwk)

COMPOSITING THE DRILL HOLE DATABASE

Prior to block model interpolation, the underlying sample database was converted to samples of constant length. All data was composited, "down-the-hole", into two-metre samples. This calculation honoured the Zone domain boundaries of the geologic model, and the resulting grades were weighted by both the length and bulk density of the original sample. The basic statistics of the composited database are listed in Table 16-4.

TABLE 16-4 COMPOSITES - STATISTICS BY ZONE
Inmet Mining Corporation – Çayeli Mine

Zone	Metres	Cu%	Zn%	Ag g/t	Au g/t	Pb%	EqCu%
1	6,355	5.09	12.67	129	1.81	0.82	9.78
2	35,767	4.78	6.11	41	0.84	0.27	7.04
3	5,910	4.38	1.80	12	0.35	0.09	5.05
4	1172.7	5.34	9.33	122	1.01	0.70	8.79
5	2,003	4.01	9.12	69	0.67	0.50	7.38
6	600.8	3.95	2.61	17	0.28	0.13	4.91
Total	**51,808**	**4.74**	**6.57**	**51**	**0.89**	**0.33**	**7.18**

Notes:
1. Drilling as of December 31, 2004
2. Statistics at 2.5% EqCu Cut-Off
3. (Zones: 1=MOCO, 2=MOSpec, 3=MOStwk, 4=DOCO, 5=DOSpec, 6=DOStwk)

There is a dilution effect when one composites samples of varying grades into larger intervals and compares the results at a cut-off grade. This effect is due to the reduction in selectivity in longer composited intervals compared to original drill hole assay data. Table 16-4 shows an increase in the total metres above the 2.5% EqCu cut-off and a corresponding reduction in grade in comparison with Table 16-3. This represents approximately 3% dilution as a result of compositing the database.

RPA considers the compositing approach by honouring zones to be appropriate.

VARIOGRAPHY

Multidirectional variograms (correlograms) were produced for copper and zinc in each of the six domain zones in the geological model. The search ellipsoids that have been used for selection of composites for grade interpolation are defined from two-structure correlograms. A maximum distance of 150 m is used for grade interpolation of Cu and Zn. In this case, only interpolation of Cu in Zone 2 is limited by the 150 m distance parameter.

The variogram parameters and search ellipsoid orientations are listed in Tables 16-5 and 16-6, and graphs are presented in Appendix 2 Figures 24-1 through 24-12.

TABLE 16-5 VARIOGRAM PARAMETERS BY ZONE - COPPER
Inmet Mining Corporation – Çayeli Mine

Zone	Nugget	Sill	Long Axis			Intermediate Axis			Short Axis	Type
			R (m)	Az	Dip	R (m)	Az	Dip	R (m)	
1	0.2	0.45	40	0	0	20	-270	-60	20	Sph
		0.35	100	0	0	40	270	-60	40	Sph
2	0.25	0.55	70	0	0	45	270	-60	30	Sph
		0.2	200	0	0	80	270	-60	60	Sph
3	0.35	0.55	50	0	0	25	270	-60	25	Sph
		0.1	70	0	0	35	270	-60	30	Sph
4	0.25	0.45	15	300	-60	10	210	-80	7	Exp
		0.3	65	300	-60	60	210	-80	35	Sph
5	0.2	0.5	20	35	0	15	305	-45	10	Exp
		0.3	100	35	0	70	305	-45	40	Sph
6	0.3	0.3	30	0	0	30	270	-55	15	Sph
		0.4	70	0	0	70	270	-55	50	Exp

(Zones: 1=MOCO, 2=MOSpec, 3=MOStwk, 4=DOCO, 5=DOSpec, 6=DOStwk)

TABLE 16-6 VARIOGRAM PARAMETERS BY ZONE - ZINC
Inmet Mining Corporation – Çayeli Mine

Zone	Nugget	Sill	Long Axis			Intermediate Axis			Short Axis	Type
			R (m)	Az	Dip	R (m)	Az	Dip	R (m)	
1	0.25	0.55	15	180	-30	10	270	-60	10	Sph
		0.2	70	180	-30	60	270	-60	25	Sph
2	0.2	0.4	40	330	0	25	240	-60	25	Sph
		0.4	75	330	0	60	240	-60	50	Sph
3	0.3	0.55	30	270	-30	40	180	0	15	Sph
		0.15	100	270	-30	100	180	0	50	Sph
4	0.2	0.7	15	30	0	15	300	-50	10	Sph
		0.1	50	30	0	50	300	-50	30	Sph
5	0.25	0.55	30	30	0	30	300	-50	15	Sph
		0.2	70	30	0	70	300	-50	30	Exp
6	0.25	0.55	30	30	-30	30	300	-45	15	Sph
		0.2	100	30	-30	100	300	-45	30	Sph

RPA considers the variogaphy approach at Çayeli to be appropriate. RPA recommends that the variography exercise also be done for silver and gold.

CAPPING OF HIGH GRADE SAMPLES

CBI reports that the database was tested, by zone, for the presence of anomalous high-grade values by means of a Decile Analysis. This is a statistical method which sorts the data by grade and evaluates the relationship between the contained metal and the number of actual samples that define this material. Several very high-grade samples can represent a high proportion of the overall contained metal in the database and, in some circumstances, this can result in overestimation of resources in the grade model.

The results of the contained metal in the highest bin of the decile analysis are shown in Table 16-7. If there is greater than 40% of the contained metal from an individual zone in one decile bin, this is an indication that some form of top-cutting may be required. The Clastic and Spec zones generally show lower total metal in the top decile due to the more consistent, high-grade, nature of the mineralization in these domains.

The stockwork zone, however, is much more variable in grade, and this is reflected in the decile analysis.

TABLE 16-7 CONTAINED METAL IN TOP DECILE - BY ZONE
Inmet Mining Corporation – Çayeli Mine

Zone	Cu	Zn	Pb	Ag	Au
1	27	22	30	24	26
2	23	35	49	37	30
3	39	69	65	77	40
4	24	22	28	20	22
5	27	27	41	29	28
6	57	75	57	76	29

Following a detailed inspection of the grade distribution in the stockwork zones (zone 3 and 6), it is apparent that most of the high-grade samples occur very close to the contact with the overlying massive sulphides. The general nature of the Stockwork zone shows the highest concentration of veining (and overall grades) at the top, decreasing with depth into the footwall. This feature is also apparent in the sloping grade distributions for the stockwork zones in the contact profiles (Figure 16-1). This type of grade distribution in a footwall stockwork zone of a VMS deposit is quite common.

Although these higher-grade samples are not considered geologically anomalous, CBI reports that their influence during block grade interpolation should be locally restricted. CBI reports that this has been accomplished through the application of an "outlier limitation". The outlier limitation confines the effect of high-grade samples to a specified distance during block grade interpolation. It should be noted that operational experience has greatly influenced the selection of the grade/distance parameters in the outlier limitations. In the Main Stockwork domain (zone 3), zinc grades above 20% were limited to an effective distance of 10 m during interpolation. There were no limitations placed on copper samples in zone 3. In the Deep Stockwork domain (zone 6), copper samples grading over 9% were limited to 20 m and zinc samples grading over 15% were also limited to 20 m of influence.

The overall effect of the outlier limitation is related to sample density during interpolation. The sample spacing in the Deep ore is generally much wider than in the Main ore zone. As a result, the outlier limitations stated above have a lesser effect on the upper portion of the deposit. Overall, the contained copper and zinc in the mineral resource is reduced by 1.6% and 0.8%, respectively, as a result of the application of these limitations.

In general, RPA agrees with the outlier limitation, but prefers to cap values. RPA has reviewed the Cu-Zn probability plots that have been produced for each zone (Appendix 3, Figures 25-1 to 25-12). Only these plots were available to RPA. Probability plots are derived from composites rather than raw assays. RPA's approach is to cap assays prior to compositing. By doing so, the high-grade assays are prevented from being smeared over two composites. Table 16-8 presents the capping factors that could be appropriate to be applied to Cu and Zn. Although capping can be applied, its effect on the overall grade will probably be relatively limited. Only very few samples have to be capped as shown by the percentage of the population that is subject to capping. Grade reconciliation will also show whether or not capping is needed.

TABLE 16-8 CAPPING FACTORS BY ZONE – CU & ZN
Inmet Mining Corporation – Çayeli Mine

Zone	Element	Capping Factor	% of Pop. Capped	Element	Capping Factor	% of Pop. Capped
1	Cu	28 %	0.1 %	Zn	35 %	0.4 %
2	Cu	None	None	Zn	None	None
3	Cu	None	None	Zn	23%	0.3 %
4	Cu	18 %	1 %	Zn	None	None
5	Cu	15 %	1 %	Zn	None	None
6	Cu	None	None	Zn	14 %	0.8 %

(Zones: 1=MOCO, 2=MOSpec, 3=MOStwk, 4=DOCO, 5=DOSpec, 6=DOStwk)

NET SMELTER RETURN (NSR) AND EQUIVALENT COPPER CALCULATIONS

For the purposes of the resource and reserve estimation, and since copper is the most important metal in terms of its contribution to the NSR, CBI had been using, until December 2005, an equivalent copper grade (EqCu%) as a common parameter from which the cut-off was derived. In December 2004, resources and reserves were reported at a 2.5% EqCu and the EqCu formula was:

$$EqCu\% = Cu\ \% + 0.37*Zn\%$$

RPA recommends using NSR values, rather than EqCu values, for resource and reserve reporting, cut-off determination, and stope design. The NSR value for each block of the model is calculated on the basis of its grades (Cu, Zn, Ag, Au) and metal recoveries. Metal recoveries are obtained from the recovery curves based upon head grades.

In February 2006, CBI determined that:

- The resources are estimated using a lower cut-off of $US 35 per tonne in order to inventory material with potential to be mined at some future date, should higher metal prices prevail.
- The reserves are reported at a US$ 46 per tonne of ore cut-off.

RPA recommends reporting the resources at the same, $US 46 per tonne of ore, cut-off as the reserves. This approach will give a better idea of the potential for resources to become reserves.

RPA agrees with the US$46 per tonne cut-off for reserve reporting, given that the current 2006 Five Year Plan operating costs are forecast at $52 per tonne.

BLOCK MODEL LIMITS

The following section outlines the parameters and methodology used in the development of the block model.

A block model was developed based on the parameters in Table 16-9.

TABLE 16-9 BLOCK MODEL PARAMETERS
Inmet Mining Corporation – Çayeli Mine

	Minimum	Maximum	Size	Number of Blocks
X (East)	800	1200	5 m	80
Y (North)	1400	2150	5 m	150
Z (Elevation)	500	1200	5 m	140

Blocks in the model have been coded using domain zone designation. Blocks which straddle an internal boundary between two zones are designated on a majority basis. The percentage of each block which occurs inside the geology domain solids is also stored in order to accurately determine the contained resources.

BLOCK MODEL INTERPOLATION OF GRADE AND DENSITY

DOMAINS AND HARD BOUNDARIES

Copper and zinc grades in the block model are estimated using the Ordinary Kriging interpolation method. Silver, gold, lead, and density are estimated using the Inverse Distance weighting method. Appendix 4 (Figures 26-1 to 26-10) presents results of interpolation (EqCu cross sections).

Each domain zone is treated with “hard” boundary limitations, which means that block grades could only be estimated using samples located inside that specific zone (for example, the Main Zone Clastic Ore (zone 1) block grades are estimated using only samples located inside zone 1).

RPA generally agrees with the ‘hard’ boundary approach which is based on ore type. However, RPA recommends being very careful in this approach in the southern part of the deposit where there is no displacement of the mineralized zone by the Scissor Fault (Appendix 1, Figures 23-1 to 23-10). RPA recommends outlining a ‘box’ in the southern part of the deposit in the vicinity of the Scissor Fault where the ‘hard’ boundary is not applicable. For example, RPA has observed significant differences in grades for two adjacent blocks which belong to the same ore type (Clastic Ore) but different domains (one block being in the Main Zone and the other block in the Deep Zone) and where the Scissor Fault shows no apparent displacement. In this particular case, the block on the hanging wall of the fault has an estimated grade of 12.62% Zn, while the adjacent block on the footwall side of the fault has an estimated grade of 4.10% Zn. Grades are overestimated on one side of the fault and underestimated on the other.

GRADE ESTIMATION

Grade estimations were done in two passes as follows. The first pass utilizes the search ellipse parameters listed previously and is limited to diamond drill hole and channel sample data (i.e., sludge data is excluded from the first pass interpolation). Grades in each block are estimated using a maximum of 12 of the closest composite samples, with a maximum of four of two-metre composite samples used from a single drill hole or channel sample. Finally, octant search criteria were put in place for the grade search to occur in all directions.

The second pass in the interpolation process is limited to recalculating only those blocks that are within 15 m of an underground Sludge sample. In this case, all diamond drill hole, channel, and sludge sample data are utilized. Block grades in Clastic and massive sulphide domains (zones 1, 2, 4, and 5) use the closest 16 composite samples, and blocks in the Stockwork domains (zone 3 and 6) use the closest 12 samples. Similar to the first pass, a maximum of four of two-metre composites are used from a single drill hole, channel, or sludge sample.

This second pass of estimation was added to the resource estimates in 2002 because it was found to improve the short-term grade estimates. The search limits have been defined at a maximum of 15 m from a Sludge sample location in order to have some influence in providing more accurate estimations for all blocks which may occur between two developed sublevels. As a result, the Sludge data affects only higher-level resources (measured and, to a lesser extent, indicated resources) and this information does not influence any longer term resources.

RPA generally agrees with the grade interpolation methodology and considers it to be consistent with industry standards.

DENSITY ESTIMATION

Density was estimated for each block using the inverse distance interpolation method. A 60 m x 60 m x 20 m search ellipse was oriented at 0W–65W for the Main zone and 30W–50W for the Deep Ore zone. "Hard" domain boundaries were applied to the data and a maximum total of 12 composite samples, four per hole, were used for each block estimate.

RUSSIAN ADIT ZONE

The Russian Adit zone is a relatively small near surface sulphide occurrence located approximately 400 m south of the Çayeli headframe. The overall potential of the area is not well known at this time, but there are numerous drill holes and several small adits that give information on the zone.

The inferred resource in the Russian Adit Zone was reviewed in 1999, and no further work has been done in this area since that time. This estimate was conducted using a sectional polygonal method, resulting in a total undiluted inferred resource of 500,000 tonnes at an average grade of 4.8% copper and 10.5% zinc. There are no silver, gold, or lead assays available from this area. At present, no reserves have been estimated for this area.

CBI reports that the Russian Adit resource estimates excludes several very good drill intersections that are too widely spaced to correlate with other drilling information. Surface outcrops throughout this area show large zones of intense hydrothermal alteration, indicating that potential ore forming fluids were active in the past. This area remains an excellent target for future exploration.

RPA has reviewed the available documentation on resource estimates of the Russian Adit zone consisting of one report published in 2000 and entitled 'Review of the Russian

Adit Resource - Çayeli Deposit, Çayeli, Turkey'. RPA considers the polygonal approach to be consistent with industry standards.

At the time of the site visit, RPA tried, with geology staff, to find the backup of the Russian Adit resource estimates. The attempt to find it was unsuccessful. It has been agreed between RPA and CBI staff that the Russian Adit zone will be reinterpreted and its resource re-estimated in the event the backup interpretation of the polygonal estimate is not found.

In February 2006, CBI reports an inferred resource of 448,000 tonnes at an average grade of 3.3% Cu and 11.1% Zn. In comparison with the 1999 estimates, one hole has been removed and put into the main orebody. This resulted in lower tonnage and grade changes for the Russian Adit resources. The resources were calculated using the polygonal method.

MINERAL RESOURCE AND MINERAL RESERVE CLASSIFICATION

The in-situ mineral resource has been classified as measured, indicated, and inferred based on a combination of the number of samples used in the interpolation, the distance from the block, and, finally, the type of samples used. Proven and probable mineral reserves are derived from measured and indicated resources, respectively. Sound engineering has shown that they can be mined and processed at a profit. These definitions result from experience gained through past mining and show confidence levels that are consistent with the definitions in the CIM Standards on Mineral Resources and Reserves (Aug 20, 2000). The definitions are derived using all drill hole, channel, and sludge sample data.

During the estimation process, the distance to the nearest sample (D) and the number of composites (N) are stored in the model. After the estimation process, these stored values are used to define the ore classes according to the definitions above. The process of classification is the following:

- All blocks are tagged as inferred initially.

- All blocks that
    - are between 10 m and 28 m of ‘openings’ (openings here are referred as sludge samples or channel samples or stope definition drill holes)
    - have more than three samples that have been used for grade estimation,

  are considered in the Indicated category.

- All blocks that:
    - are within 10 m of ‘openings’ and
    - have more than three samples that have been used for their grade estimation,

  are considered in the Measured category.

The classification parameters are described as follows:

- **Measured Resource (Proven Reserve)** – Blocks in the model in which the grade has been estimated using a minimum of three drill holes. All blocks in this category are located within a maximum distance of 10 m from either an existing drift opening underground (Channel or Sludge sample) or a Stope Definition drill hole.

- **Indicated Resource (Probable Reserve)** – Blocks in the model that do not meet the criteria for measured resources, but have had grades estimated using a minimum of three drill holes within the search ellipse, the closest of which is a maximum distance of 28 m from the centre of the block.

- **Inferred Resource** – Blocks in the model which do not meet the criteria for measured or indicated resources but have had grades interpolated by at least one drill hole within the search ellipse.

Based on the these classification parameters, resources are not considered to belong to the highest (Measured) class until they have been either exposed with an underground drift or have been diamond drilled on seven metre centers with stope definition holes.

In general, RPA agrees with the resource classification and considers it to be in accordance with the National Instrument 43-101 standards. The resource classification is illustrated in Figure 16-2 and on vertical cross sections in Appendix 5 (Figures 27-1 to 27-10).

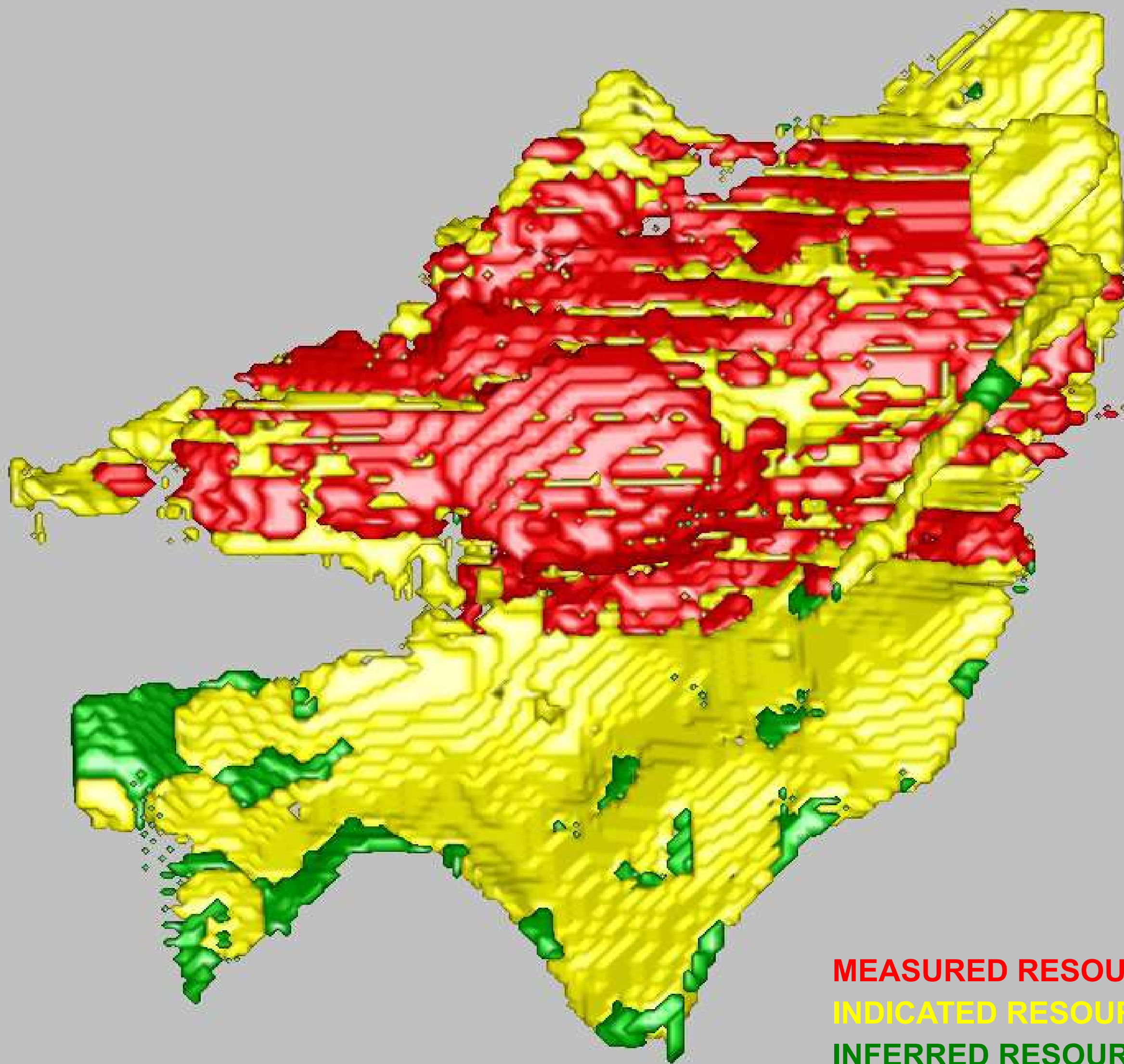


Figure 16-2

Inmet Mining Corporation

Çayeli Project

Republic of Turkey

Distribution of Resource Classification (Looking NE)

March 2006

Source: Sim, 2004.

MINERAL RESOURCE – TONNES AND GRADES

The undiluted mineral resources, as of December 31, 2005, for the Çayeli deposit are listed in Tables 16-10 and 16-11.

TABLE 16-10 MINERAL RESOURCES (INCLUSIVE OF MINERAL RESERVES)

Inmet Mining Corporation – Çayeli Mine

Category	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	$NSR/T
Measured	6.41	4.10	5.84	45	0.66	0.31	78
Indicated	9.61	3.88	5.69	52	0.59	0.36	73
Meas.+ Ind.	16.02	3.97	5.75	49	0.62	0.34	75
Inferred							
Çayeli	0.71	3.55	3.15	28	0.52	0.20	61
Russian Adit	0.45	3.29	11.08	N/A	N/A	N/A	N/A
Total Inferred	**1.16**	**3.45**	**6.23**	**N/A**	**N/A**	**N/A**	**N/A**

Notes:
1. CIM definitions were followed for mineral resources.
2. Mineral resources are estimated at a cut-off grade of US$35 NSR/tonne of ore.
3. Mineral resources are estimated using prices of US$1.10 per pound Cu, US$0.55 per pound Zn, US$5.60 per ounce Ag, $US450 per ounce Au.
4. Measured and indicated mineral resources are inclusive of mineral reserves.
5. Russian Adit inferred resources: Au and Ag have never been assayed.

TABLE 16-11 MINERAL RESOURCES (EXCLUSIVE OF MINERAL RESERVES)

Inmet Mining Corporation – Çayeli Mine

Category	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	$NSR/T
Measured	1.91	3.08	3.09	22	0.50	0.18	57
Indicated	2.65	3.05	2.55	20	0.44	0.15	54
Meas.+ Ind.	**4.56**	**3.06**	**2.78**	**21**	**0.46**	**0.16**	**55**
Inferred							
Çayeli	0.62	3.38	2.85	18	0.26	0.12	58
Russian Adit	0.45	3.29	11.08	N/A	N/A	N/A	N/A
Total Inferred	**1.07**	**3.34**	**6.29**	**N/A**	**N/A**	**N/A**	**N/A**

Notes:
1. CIM definitions were followed for mineral resources.
2. Mineral resources are estimated at a cut-off grade of US$35 NSR/tonne of ore.
3. Mineral resources are estimated using prices of US$1.10 per pound Cu, US$0.55 per pound Zn, US$5.60 per ounce Ag, $US450 per ounce Au.
4. Measured and indicated mineral resources are exclusive of mineral reserves.
5. Russian Adit inferred resources: Au and Ag have never been assayed.

CBI reports the mineral resources at a cut-off of US$ 35 per tonne, which is lower than the cut-off used to report mineral reserves (US$ 46 per tonne) by 24%. CBI considers to inventory material with potential to be mined at some future date, should higher metal prices, and/or an operating cost benefit associated with the shaft deepening, an/or a weaker Turkish lira, prevail. RPA recommends reporting mineral resources at the same cut-off as the mineral reserves (US$46 NSR per tonne) for the following reasons:

- Mining and processing ore at a $35 per tonne cut-off has not been the case for the last three years.
- The benefits of shaft deepening and other improvements have to be tested in the second half of 2006 in order to validate any assumptions in regard to the potential lowering of the operating cost per tonne.
- Operating at a $35 per tonne cut-off would require a reduction of operating cost by 30%.

MINERAL RESOURCE – REPORTING BLOCKS ABOVE CUT-OFF

In December 2004, mineral resources were reported at a 2.5% EqCu cut-off. All individual blocks with EqCu grade above the cut-off are summed for tonnage and Cu, Zn, Ag, Au, Pb, and SG. At the time of the site visit, RPA recommended for the December 2005 Mineral Resource Report that isolated blocks that are not adjacent to each other and which do not represent clusters with a minimum tonnage for mining (e.g., three blocks high by three blocks wide by three blocks deep) should be removed as they do not represent reasonable prospects for mineral extraction. This recommendation has been taken into account in this report.

A decision whether or not to include blocks in mineral resources should be made in conjunction with engineering staff.

DISTRIBUTION OF RESOURCES BY ORE TYPE

The distribution of the total mineral resources by ore type, as of December 31, 2005, indicates that 20% of the total resource is Clastic ore, 28% is Black ore, and 52% is Yellow ore.

TABLE 16-12 MINERAL RESOURCES BY ORE TYPE
Inmet Mining Corporation – Çayeli Mine

Ore Type	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	NSR/T
Clastic	3.36	5.45	9.81	115	1.14	0.73	101
Black	4.73	3.33	9.77	65	0.68	0.50	78
Yellow	8.63	3.70	1.76	14	0.36	0.09	62
Total	16.73	3.95	5.64	48	0.61	0.34	74

Notes: Ore types for total mineral resources (Measured+Indicated+Inferred) at a US$ 35 NSR per tonne of ore cut-off.

COMPARISON WITH PREVIOUS RESOURCE ESTIMATES

The original (pre-mining) undiluted resource models from 2004 and 2005 are compared in order to see what relative changes may have taken place as a result of such factors as new data, new geological interpretation, new methodology, etc. The result of the comparison is shown in Table 16-13.

TABLE 16-13 2005 VS. 2004 MINERAL RESOURCES
Inmet Mining Corporation – Çayeli Mine

	Dec. 31, 2005			Dec. 31, 2004			Difference %		
Category	MT	Cu %	Zn %	MT	Cu %	Zn %	T	Cu %	Zn %
Measured	6.41	4.10	5.84	7,67	4.26	5.90	-16.4	-3.6	-1.0
Indicated	9.61	3.88	5.69	8,16	3.65	5.91	17.8	6.2	-3.7
Meas.+ Ind.	**16.02**	**3.97**	**5.75**	**15.82**	**3.94**	**5.91**	**1.3**	**0.6**	**-2.6**
Inferred									
Çayeli	0.71	3.55	3.15	2.83	4.43	6.08	-75.1	-19.9	-48.2
Russian Adit	0.45	3.29	11.08	0.50	4.77	10.54	-10.4	-31.0	5.1
Total Inferred	**1.15**	**3.45**	**6.23**	**3.33**	**4.48**	**6.75**	**-65.4**	**-23.1**	**-7.7**

Note: Variance expressed as percentage change from December 31, 2004 estimates

CBI reports that changes in mineral resources between 2004 and 2005 are due to:

- An update of the interpretation of the mineralized envelope based on diamond drilling carried out this year,
- The removal of isolated blocks,
- The mining of some 833,600 tonnes of ore,
- The shifting from an EqCu to a NSR formula.

COMPARISON OF INTERPOLATION METHODS

CBI conducted a comparison between Ordinary Kriging, Inverse Distance, and Polygonal interpolation methods for the total original (pre-mining) resource. RPA reviewed the comparison results. As the results are quite close, RPA considers Ordinary Kriging to be appropriate. Ordinary Kriging is the most conservative of the three methods. This similarity between the interpolation methods is due to the high average sample density and the consistently high-grade nature of the deposit. Table 16-14 presents the comparison results.

TABLE 16-14 COMPARISON OF INTERPOLATION METHODS (TOTAL RESOURCE – 2.5% EqCU CUT-OFF)
Inmet Mining Corporation – Çayeli Mine

Method	M tonnes	Cu%	Zn%	EqCu%	Contained Metal (M tonnes EqCu)
Krige	26.0	4.15	6.00	6.37	1.66
IDW	25.9	4.22	6.03	6.45	1.67
Polygonal	24.2	4.73	6.53	7.15	1.73

Note: Comparison was carried out on Total Resource as of December 2004 at a 2.5% EqCu cut-off.

MINERAL RESERVES

The mineable reserves are listed in Table 16-15. Mineable reserves are derived from measured and indicated resources.

TABLE 16-15 MINERAL RESERVES
Inmet Mining Corporation – Çayeli Mine

Category	MTonnes	Cu %	Zn %	Ag g/t	Au g/t	Pb %	$NSR/T
Proven	4.70	3.77	5.85	44	0.59	0.30	73
Probable	6.90	3.57	5.88	52	0.53	0.36	69
Total	**11.60**	**3.65**	**5.87**	**49**	**0.56**	**0.34**	**70**

Notes:
1. CIM definitions were followed for mineral resources.
2. Mineral reserves are estimated at cut-off grade of US$46/tonne of ore.
3. Mineral reserves are estimated using prices of US$1.10 per pound Cu, US$0.55 per pound Zn, US$5.60 per ounce Ag, US$450 per ounce Au.

MINE DESIGN

Detailed mine designs have been developed for all economic material between the 1060 and 580 levels. The majority of the resource outside of these limits remains in the Inferred class and, therefore, requires upgrading through future diamond drilling.

Mineral reserves have been developed using an equivalent cut-off limit of US$46 NSR/tonne cut-off.

Basically, two mining methods are employed to extract ore in the Main zone and Deep Ore zone, which are more extensively described in Item 25. They are Transverse and Longitudinal Stoping, which are carried out on 20 m vertical sublevels.

A large proportion (+90%) of the Main zone is extracted using Transverse Stoping because the thickness of the ore is typically very high in this zone. An initial Strike Access drift is driven along the hangingwall contact on each level. Primary sill drifts are then driven, seven metres wide and on 14 m intervals, to the footwall contact. Ore is blasted by rows in order to retreat towards the mucking drift. If necessary, ore is also blasted by ore type (clastic, spec, stockwork), and mucked out entirely before blasting another type. Stopes are mined without being cut into panels. Following the extraction of the primary Transverse stopes, the openings are backfilled with either cemented rock fill or paste backfill. The secondary sill drifts and subsequent stopes are then taken out and filled with waste rock. The final step is to retreat back towards the centre of the deposit, extracting the remaining material between the initial Strike drifts using Longitudinal Stoping. This material is generally taken out in 20 m long intervals in order to allow for backfilling and to reduce the amount of exposed hanging wall.

In the Deep Ore zone, a similar mining method as described above is used in all areas where the ore is greater than 15 m in thickness. Narrower ore zones are extracted using the Longitudinal Stoping method. In areas where the dip becomes relatively flat (approximately 50 degrees), the complete ore zone from the hanging wall (HW) to footwall (FW) will be extracted using transverse stopes.

The remaining reserve will be extracted in the following proportion: approximately 24% of the Main zone will be extracted through Longitudinal Stoping and the other 76% through Transverse Stoping. The Deep Ore will have 14% Longitudinal and 86% Transverse stopes.

CREATING STRINGS AND WIREFRAMES FOR THE MINING BLOCKS

The equivalent copper grades in the block model were used to control the extent of development during the mine design process. Three-dimensional wireframes have been developed which represent all planned sill drift and stope designs. There are several criteria for this procedure:

- The angle of stopes will not be shallower than 50°.
- Ore will be taken in Longitudinal stopes where ore thickness is less than 15 m horizontally.
- In cases where isolated patches of resources exist, an economic analysis is conducted in order to ensure the ore mined supports the additional development costs.

APPLICATION OF DILUTION FACTORS

There are two types of dilution identified, planned and unplanned. These come from the following sources:

PLANNED DILUTION

- Internal dilution from barren dykes and waste inclusions (for example, dykes are often narrow and somewhat erratic and, as a result, cannot be effectively separated as waste during mining. Therefore, they are included in the resource estimate as part of the ore zone).
- “Design” dilution where waste material is mined in order to improve the geometry of a stope.
- Hangingwall dilution in both longitudinal stopes and some Deep Ore Transverse stopes.
- Footwall dilution (which is often low-grade material).

The planned dilution (tonnes and grades) is part of the undiluted mineral reserves.

UNPLANNED DILUTION

- Backfill dilution from the sidewalls and floor during the extraction of secondary and tertiary stopes and from the back of blind primary and secondary stopes.
- Rock from hanging wall and footwall.

Dilution has been applied to individual workplaces based on the following criteria/assumptions:

- Calculations depend on the actual stope productions.
- All dilution factors are calculated by tonnage (as opposed to volume).
- Calculations are based on the Cavity Monitoring System (CMS) data of stopes.
- All actual dilution is calculated for secondary and tertiary stopes.
- For primary stopes, assuming 0.5 m overbreak during stope blasting, dilution from one end (HW or FW) will be 2.0% (the average stope length is approximately 25 m).
- Secondary and tertiary/longitudinal stopes have an average dilution of approximately 5% according to actual CMS survey data.
- Similar dilution factors are assumed for the deep ore zone (ore below the 820 level).
- All dilution included is at zero grade.

The average dilution factors are summarized in Table 16-16:

TABLE 16-16 DILUTION FACTORS
Inmet Mining Corporation – Çayeli Mine

Mining Method	Dilution Factor
Primary Transverse Stopes (dilution from one end, FW or HW)	2.0%
Secondary Transverse Stopes (dilution from one end & sidewalls)	5.0%
Tertiary / Longitudinal Stopes (dilution from one end & sidewalls)	5.0%
Overall	**4.5%**

Based on reviewing the historical data and CMS monitoring records, RPA estimates the dilution factors to be appropriate.

APPLICATION OF ORE EXTRACTION FACTORS

Planned and unplanned ore losses come from the following sources:

- Ground problems during advancing overcut/undercut drifts, which may cause stopping the advance. The ore left behind may be sterilized and thus may not be possible to recover.

- Improper drift advances and blasting of stopes, which may cause underbreaks from sidewalls and from HW and FW faces.
- An average of one metre pillar that is left at the back in blind stopes for protecting from unwanted overbreaks from stope backs.
- Losses in long stopes that are split into two and panelled out.

Overall, average extraction factors are summarized in Table 16-17.

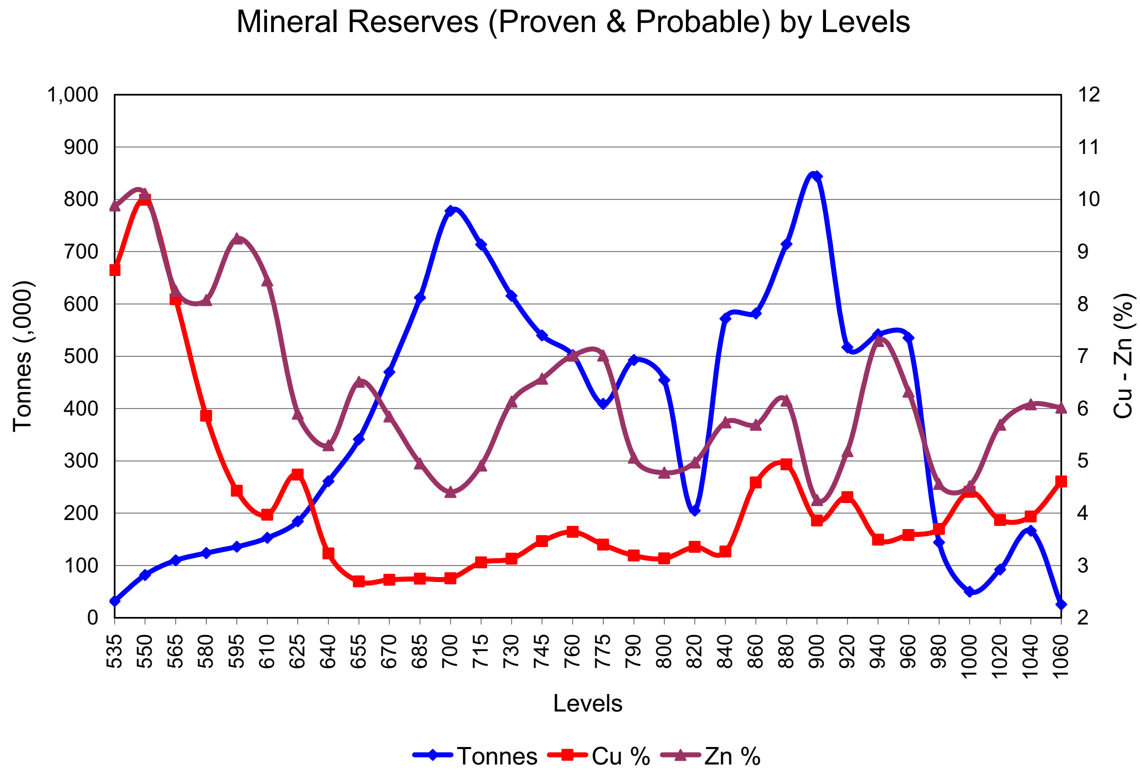
TABLE 16-17 EXTRACTION FACTORS
Inmet Mining Corporation – Çayeli Mine

Mining Method	Extraction Factor
Normal stopes (primary and secondary)	95%
Blind stopes (primary and secondary)	90%
Stope in central pillar (blind, primary and secondary)	90%
Overall	**92.5%**

DISTRIBUTION OF MINERAL RESERVES BY LEVELS

Figure 16-3 presents the distribution of proven and probable reserves by levels. The tonnage is essentially concentrated between the 670 and 960 levels, with two peaks on the 700 and 900 levels. Copper and zinc grades decrease from the 535 to 640 level. From that point to the 840 level, copper grade is fairly stable, being at approximately 3%, where it increases to approximately 4% thereafter. Zinc grade fluctuates between 4.5% and 7% from the 640 to 1060 level.

FIGURE 16-3 DISTRIBUTION OF RESERVES BY LEVELS



USE OF CAVITY MONITORING SYSTEM (CMS) SURVEYS FOR DILUTION AND ORE EXTRACTION CALCULATIONS

CBI currently maintains a very thorough stope-by-stope database, detailing the planned stope tonnage based on stope layouts, the actual mined ore based on CMS surveys, as well as ore loss and backfill and host rock dilution. In 2005, a total of 85 stopes, both conventional and blind, in all categories, primary, secondary, and tertiary, were mined and analyzed.

In each case, the stope solid is intersected with the CMS solid to calculate dilution and recovery. If the intersection includes volumes of paste fill, waste fill, or host rock, this tonnage is totaled and divided by the CMS tonnage to arrive at a dilution value for that stope.

Calculating recoveries requires more discretion. For example, if a primary stope underbreaks, the intersection volume does not necessarily represent ore loss as this volume can be taken with the adjacent secondary or tertiary blasts. If a primary stope overbreaks, this simply means more tonnes are drawn, not a 'negative' ore loss offsetting true ore loss within the same stope.

In 2005, the dilution and recovery figures, calculated in the above manner, were 4.5% and 92.5%, respectively. When applying this dilution to the CMS as-mined volumes for 2005, CBI reports an excellent correlation with the mill figures, indicating a validation of the block model and methodology for calculating dilution.

In the December 2004 report, an average of 7.5% for dilution and 89% for recovery were used. These were partly based on historical data and partly on estimates, and are now considered conservative, after having completed a substantial number of tertiary stopes (as opposed to essentially none in 2004) and having begun mining in the Deep Ore block 4, below the 820 level. The 2005 dilution and recovery figures were used to calculate the December, 2005 reserves.

RECONCILIATION WITH HISTORICAL PRODUCTION

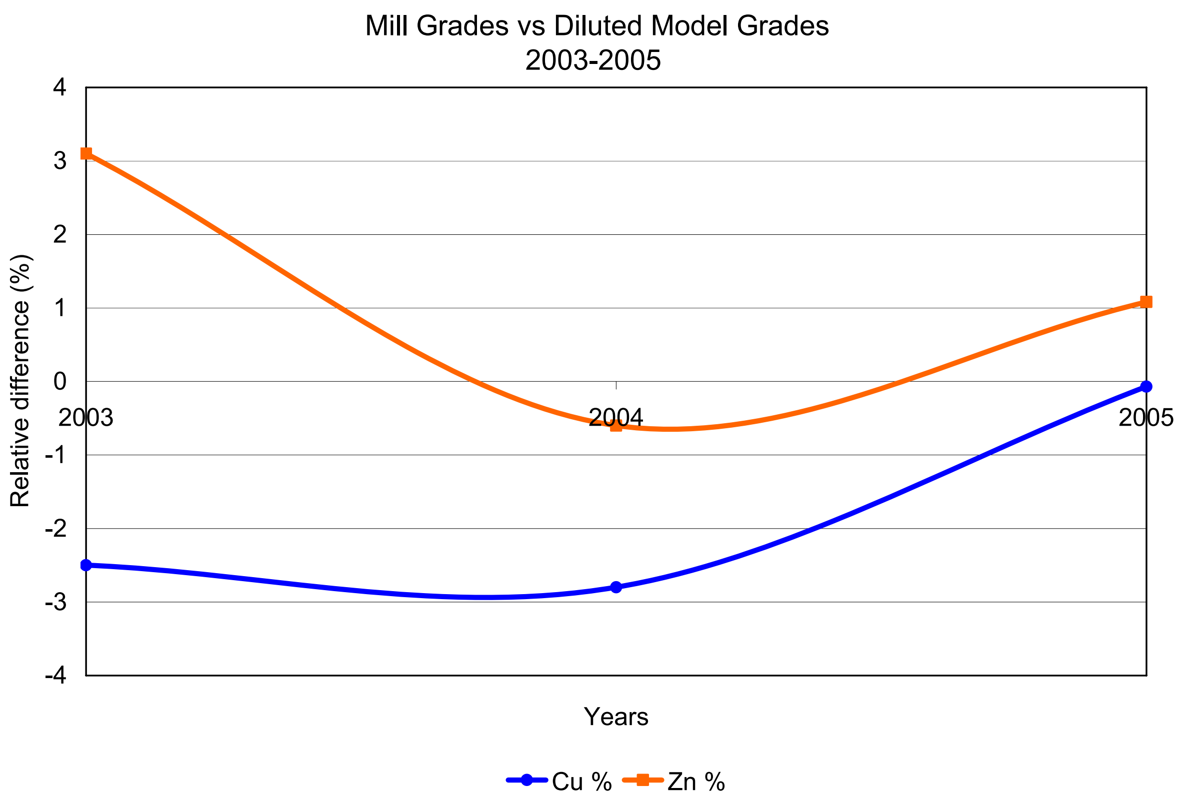
Reconciliation comparisons between the diluted reserve model grades and actual grades were carried out on an individual workplace location basis throughout the 2005 production year. In order to carry out the reconciliation, as-mined volumes are accurately surveyed with a cavity monitoring system and intersected with the geological (undiluted) model to calculate actual tonnes and grades within the surveyed volumes. Dilution is then applied to the geological (undiluted) grades. The tonnes inherently include any resulting dilution. These figures are then compared to the actual milled tonnes and grades to obtain a relative difference.

The 2005 reserve model slightly underestimated Cu and slightly overestimated Zn. Table 16-18 presents reconciliation between mill grades (actual) and diluted model grades for 2005, while Figure 16-4 shows the relative differences for copper and zinc grades for years 2003 to 2005. From this exercise, CBI concludes that model reconciliation is very good. RPA concurs with this conclusion and is of the opinion that the yearly grade difference between actual grades (mill) and diluted model grades (mine) is typical of a massive sulphide deposit.

TABLE 16-18 TONNES & GRADES RECONCILIATION - 2005
Inmet Mining Corporation – Çayeli Mine

Metal	Actual Grade (%)	Diluted Model Grade (%)	Diff. (%)
Copper	3.840	3.837	-0.07%
Zinc	6.740	6.813	1.08%

FIGURE 16-4 COPPER & ZINC RECONCILIATION FOR 2003 TO 2005 PRODUCTION



On a short-term basis (daily, weekly, monthly) CBI reports that grade reconciliation between the geology model and mill grades is essentially based on the broken ore stope sample grades, adjusted to mill grades. Stope grades for a specific month are adjusted according to the ratio of broken ore sample grade to mill grade. The adjusted grades become the 'real' grades and are compared to a 'best guess' diluted block model for which dilution is visually estimated by a geology technician when visiting stopes. The estimated dilution factor does not correspond necessarily to the unplanned dilution factor (see Application of Dilution Factors) and is not applied in every case. RPA has found that the estimated factor is generally lower than the unplanned dilution factor. The difference between the two is sometimes quite significant. Adjustment to block model tonnage is based on the truck tonnage/mill tonnage ratio.

Grade reconciliation is not easy when ore is stockpiled in different piles on surface before being sent to the mill, however, RPA recommends the following for grade reconciliation:

- In order to be coherent in the process of grade reconciliation, the forecasted diluted grade of stopes, which considers a dilution factor (unplanned), should be used rather than a dilution factor that is estimated at the time of underground stope visits. Reconciliation is carried out between stopes (that are represented as a block model on which dilution factors have been added) and mill. Since stopes are the basis of short-term and long-term forecasts and budgets, the forecasted stope diluted grade should be compared to the mill.

- If possible, grade reconciliation should be attempted for portions of stopes that have been blasted. Stopes are generally subdivided into several blasts and sometimes grades of these blasts vary significantly from one to another.

- Reconciliation should be carried out between mill and broken ore samples for diluted block model, stopes that have been CMS surveyed, and undiluted block model.

- The broken ore stockpile mine/mill grade ratio should be used to back-calculate stope grades.

- A more detailed study on broken ore sampling should be conducted in order to see if sampling quantity is appropriate, or whether additional sampling is necessary. Also, broken ore sample grades are generally higher than mill grades. Such results suggest that broken ore sampling should be revisited.

Long-term grade reconciliation should be carried out between the mill heads and the following:

- Broken ore samples
- Stopes that have been CMS surveyed
- Diluted block model
- Undiluted block model

COMPARISON WITH PREVIOUS ESTIMATES

Table 16-19 compares the current mineral reserves with the previous estimates.

TABLE 16-19 2005 VS. 2004 MINERAL RESERVES
Inmet Mining Corporation – Çayeli Mine

	Dec. 31, 2005			Dec. 31, 2004			Difference %		
Category	**MT**	**Cu %**	**Zn %**	**MT**	**Cu %**	**Zn %**	**T**	**Cu %**	**Zn %**
Proven	4.70	3.77	5.85	5.44	4.01	5.69	-13.6	-5.9	2.9
Probable	6.90	3.57	5.88	8.85	3.06	5.06	-22.1	16.6	16.2
Total	**11.60**	**3.65**	**5.87**	**14.29**	**3.42**	**5.30**	**-18.9**	**6.7**	**10.7**

Note: Variance expressed as percentage change from December 31, 2004 estimates

The overall changes to the reserve during 2005 are presented in Table 16-20.

TABLE 16-20 GAIN (LOSS) TO 2005 MINERAL RESERVES
Inmet Mining Corporation – Çayeli Mine

Metal	Mtonnes	Cu%	Zn%
Reserve Dec 31, 2005	11.60	3.65	5.87
Reserve Dec. 31, 2004	14.29	3.42	5.30
Reserve loss in 2005	2.70	2.43	2.85
Ore Processed in 2005	0.83	3.84	6.74
Total Reserve Loss in 2005	**1.86**	**1.80**	**1.11**

Stope definition drilling resulted in the transfer of most of the probable reserves into proven reserves.

CBI reports that the total reserve loss, which represents 1.86 Mt, is considered sterilized and has been removed from the reserves. This includes approximately 1.2 Mt of ‘bits and pieces’ scattered throughout, or surrounding, the as-mined wireframes and no longer considered accessible. In addition, approximately 0.25 Mt have been removed from the reserves on the north sides of the 1020, 1040, and 1060 levels. These tonnes are either sterilized due to falls of ground or their extraction would negatively impact on the critical surface infrastructure such as the production hoist. Approximately 0.41 Mt of inferred resources have also been removed from the mineral reserves.

17 OTHER RELEVANT DATA AND INFORMATION

MINING OPERATIONS

MINE DESIGN

The mine design is based on underground bulk mining methods with the use of delayed backfill to extract the ore in a sequential manner. A single production shaft located on the footwall side of the orebody and a service ramp provide access to the mine (Figure 17-1).

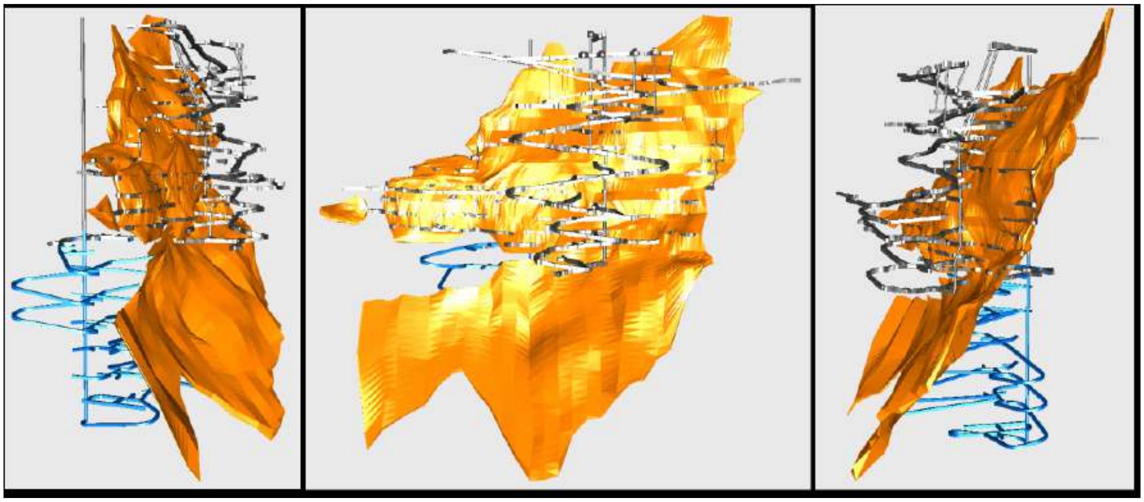
MINE DEVELOPMENT

The main levels are developed off the service ramp along the strike of the orebody at 45 m to 100 m vertical intervals. From the top of the mine down to the level 800, levels are located on the hangingwall side, and, from the 800 level down to the bottom of the mine, they are located on the footwall side of the orebody. The main levels divide the orebody into mining blocks.

Sublevels are developed inside the orebody along the contact with the hanging wall or in the centre of the orebody in the upper parts of the mine. In the lower parts of the mine, sublevels are developed along the contact with the footwall. The sublevels are part of the stopes. The ore within sublevel drift configurations is recovered after the primary and secondary stopes in a block are mined out and backfilled. Extraction of the ore from the sublevel drifts is called the tertiary stoping and is done in a retreat scenario.

The sublevel vertical distance is dictated by the stope height. In the upper parts of the mine, it is 20 m, allowing development of a 15 m high by seven metres wide stope bench for production drilling. In the lower part of the mine, the sublevels will be developed 15 m apart, allowing development of a 10 m high by 10 m wide bench for production drilling.

FIGURE 17-1 ACTUAL AND PLANNED UNDERGROUND DEVELOPMENT
(Modified from Sim 2004. Planned development in blue)



MINING METHOD

The primary mining method selected for mining the Çayeli orebody is retreat transverse and longitudinal long hole stoping with paste fill and loose or consolidated waste rock backfill application. The stopes are mined in primary, secondary, and tertiary sequencing. The primary and secondary stopes are mined as transverse and the tertiary as longitudinal stopes.

In the upper parts of the mine, stope sill drifts are seven metres wide (stope width) by five metres high and are driven on a seven metres centre. In the lower parts of the mine, the sill drifts will be of the same cross section, but driven on a 10 m centre (10 m wide stopes). The length of the sill drifts depends on the thickness of the orebody. The sill drift length varies, and can be from 10 m to 50 m long. The average stope size in the upper part of the mine is 7,000 t to 8,000 t. In the lower parts of the mine, the average stope size will be in a 5,000 t to 6,000 t range.

Stope production comprises extraction of a 15 m or 10 m high bench created between two sill drifts (Figure 17-2A). Stope blast holes in the bench are drilled with Tamrock H695 Solomatic top hammer drills (Figure 17-2B). Blast holes, of a diameter of 64 mm to 89 mm, are drilled on a variety of patterns depending on the ore and stope types. The blast holes can be drilled as up or down holes.

A slot raise is excavated between the sublevels at the end of the sill drifts with the use of a Cubex drill equipped with a V-30 Machine Roger Hammer (Figure 17-2B). The Cubex drills a 203 mm pilot hole from the upper to the lower level or vice versa, and then enlarges it with a 762 mm diameter Machine Roger Hammer head. The rest of the slot raise is drilled with a Tamrcok Solomatic drill and blasted out in steps to the full width of the stope to create a free breaking face.

The remainder of the bench is drilled and blasted in steps retreating from the open slot (Figure 17-2C). The main blasting agent is ANFO with NONEL initiation system. Paus trucks are used for transporting the ANFO. The blast holes are pneumatically loaded. Stope blasting is carried out sequentially with blasting one to two rows at a time and mucking, until the stope excavation is complete (Figure 17-2C). The mine has a central blasting system, and major stope blasts are executed at the end of shifts.

The ore is mucked from the lower sill drift intermittently with drilling and blasting one or two fans at a time (Figure 17-2C). All load-haul-dump machines (LHD) used for stope mucking are equipped with remote controlled features. After the stope has been mined out completely, it is filled with either paste or cemented rock fill or waste rocks up to the floor level of the upper sill drift (Figure 17-2D). The top of the backfilled stope becomes the mucking floor for the next lift.

Once the backfill in the primary stopes is completely cured, the pillars between the primary stopes can be mined as secondary stopes using the same stope preparation process as described above. The secondary stopes are backfilled with consolidated or loose waste rock fill depending on stope sequencing.

The tertiary stopes are longitudinal and are mined along the hanging wall or footwall stope access drifts. They are mined on a retreat pattern as soon as the primary and secondary stopes are mined out in a given mining block.

In the past, the overall stope sequencing in the upper parts of the mine began from the flanks (extremities) and progressed to the centre of the orebody and up the dip from the main levels. This sequencing created high stress concentration around the central parts in the upper levels of the mine. As a consequence of this high stress environment, parts of the developed blocks in the upper central zone collapsed. The ore within the collapsed areas is treated as sterilized and has been removed from the reserves.

Below the 880 level, the mining sequence has been changed and ore extraction starts in the centre of the block and progresses outwards to the extremities of the orebody.

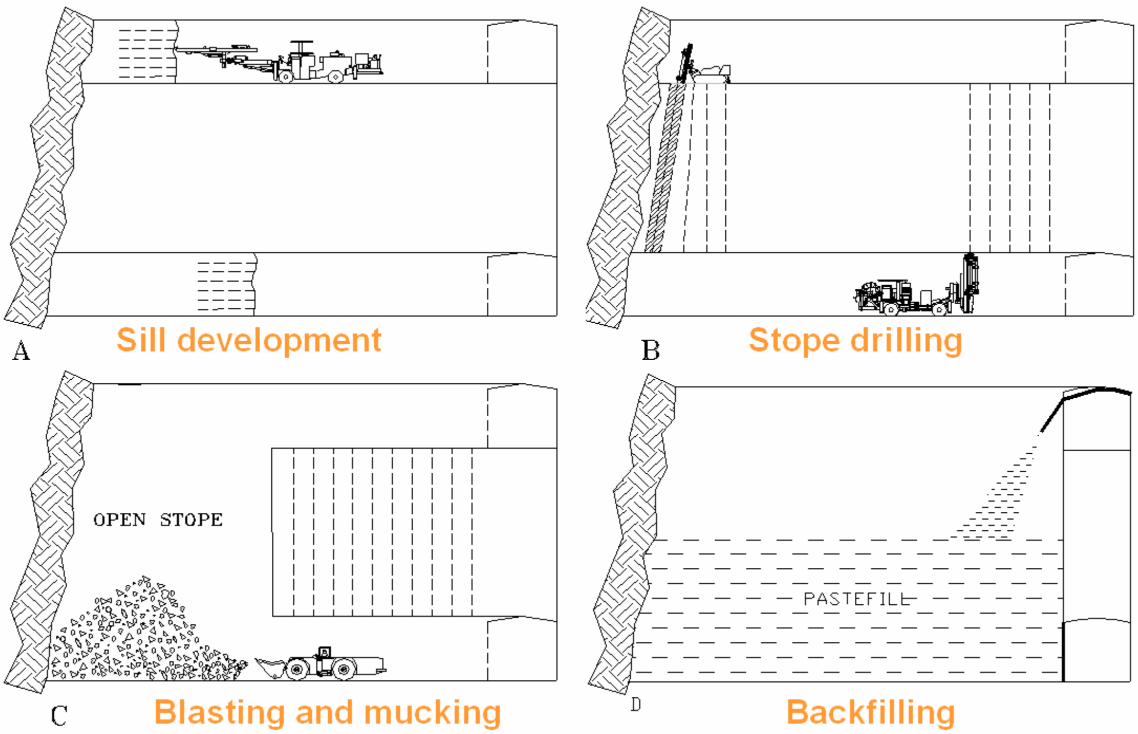
The mining blocks are not separated by traditional ore sill pillars, but by a row of primary and secondary stopes that have been mined out and filled with consolidated backfill. These primary and secondary stopes form sill pillar(s), allowing effective separation of the mining blocks. In such case, mining activities can be conducted in several blocks simultaneously without interference with each other.

The stope preparation process is based on “in-time” drifting, production drilling, blasting, and mucking philosophy. The reason for this is that, due to the relatively poor ground conditions, the excavations often deteriorate quickly. As a consequence, they require rehabilitation, which slows down extraction and increases production costs.

RPA recognizes the need to respect the potential for rapid deterioration of prematurely excavated openings. The in-time preparation philosophy is an appropriate measure to deal with this problem, however, it slows stoping productivity and ties mine resources to a given extraction area in a very unproductive way.

RPA recommends using the current stope delineation drilling for additional testing of RQDs and assessing, on a stope by stope basis, if a given stope qualifies for an exemption from the “in time” development and can be prepared and extracted without interruption. This would eliminate the need for moving in and out of the stope several times for drilling, blasting, and mucking.

FIGURE 17-2 MINING METHOD AND SEQUENCE
(Modified from Sim 2004)



GEOMECHANICS AND GROUND SUPPORT

In general, the orebody rock mass and host rocks, from the point of view of ground conditions, can be classified as poor to fair, with frequently encountered weak areas attributed to extreme schistosity and foliation. These conditions require special attention and very intensive ground support measures. Initially, the mine used conventional ground support measures, such as rock bolting and screening with occasional shotcreting, which proved to be inadequate to stop quick deterioration of the openings.

The ground support standards have been upgraded recently to deal with the generally poor ground conditions encountered in the mine and particularly on the hanging wall side of the orebody. The primary development at depth has been transferred from the hanging wall into the more competent footwall of the deposit.

The new standard ground support measures in primary development consist of steel fibre reinforced shotcrete applied on the entire walls and backs of the openings, grouted rebar or split sets or swellex bolts, and cable bolting.

The following is a list of standard ground support measures applied in primary development:

- Steel Fiber Reinforced Shotcreting (SFRS) or Mesh Reinforced Shotcreting (MRS) applied on the entire walls and backs of the openings.
- Standard bolting with mesh and split sets (2.4 m), swellex (2.4m), super swellex bolts (3.3 m to 4.0 m), grouted rebar (2.4 m).
- Grouted cable bolting (9.0 m "bird cage" design).

A 10 cm thick shotcrete layer is applied, from invert to invert, as soon as the blasted round is mucked out. SFRS is applied in one pass, while MRS is applied in two passes with a mesh layer in between. Following shotcrete application, rock bolting (2.4 m long) is applied on a regular 1.5 m x 1.5 m pattern. At some practical distance behind advancing drifts, cable bolts are installed as the last ground support measure. Each cable ring consists of seven cable bolts, nine metres long, and rings are spaced two metres apart. Drifting cannot advance more than 30 m from the last ring of installed cable bolts.

In ore, along the hanging wall or footwall and central parts of the orebody, development stope access drifts are supported with steel fibre reinforced shotctrete and grouted rebar bolts. In addition, nine metres long fully grouted cable bolts are regularly installed on a four bolts per row pattern, spaced at 2.5 m.

Sill drifts in ore are supported by standard screen mesh and bolts (approximately 90% split sets) on a nine bolts per row pattern, spaced at 1.25 m. Shotcrete is applied as required.

Occasionally, considerable bolting and shotcreting is required to rehabilitate older development and sill drifts in some parts of the mine.

Stope scheduling is based on in-time development and preparation philosophy, due to the frequently encountered poor ground conditions sometimes leading to premature deterioration of finished excavations.

Ground support equipment consists of Normet mobile shotcrete applicators, Normet 5 m³ and Paus 2.5 m³ concrete mixers, which deliver wet shotcrete from a batch plant on surface to underground. Memco MacLean platform bolters are used for drilling and rock bolt installation. New Atlas Copco bolters will replace the MacLean units in 2006. Tamrock Solomatic units are used for drilling the cable boltholes.

In recent years, the mine has introduced a program of extensive stress monitoring of workplaces and integrated computerized underground seismic monitoring system. A three-dimensional stress modeling program assists in the sequencing of mining extraction.

RPA agrees that the ground support measures adopted at the Çayeli mine are working well and, in general, are appropriate.

BACKFILLING

Backfilling is an essential part of mining at Çayeli (Figure 17-2D). The use of backfill allows relatively high extraction of the deposit and at the same time maintaining stability of the mine. Particularly, the regional stability of the weak hanging wall depends on timely backfill placement.

Several types of fill are used for backfilling: cemented rockfill/cemented waste fill (CRF/CWF), cemented paste fill (PF), and uncemented waste fill (WF).

The mining sequencing requires that the primary stopes be filled with consolidated fill (PF and/or CRF/CWF) to allow safe and maximum extraction of the secondary and tertiary stopes. The primary stopes are filled with CRF/CWF or PF with 5% to 7% cement content on average. The backfill cement content depends on stope requirements.

The majority of the secondary and tertiary stopes are backfilled with loose waste rock, and occasionally these stopes require consolidated fill in order to maintain access for mining the remaining ore or creation of sills for mining underneath the remnant ores.

The need for the partial use of consolidated backfill in the secondary and tertiary stopes results from the stope accesses being driven inside the ore.

Secondary stopes, except for the brow portion, which will be exposed during tertiary extraction, are filled with uncemented development waste material or paste fill having 2% cement content. The brow portions that will be exposed during tertiary extraction are filled with CRF/CWF with 5% cement or PF with 7% cement in order to minimize dilution.

Tertiary and longitudinal stopes are backfilled with cemented rock fill and/or paste fill at 5% cement.

Generally, the cement content of the CRF and PF is 5% to 7%, however, the stope geometry and the prevailing mining conditions dictate the final cement content in backfill.

The uppermost sill drifts on main levels require tight backfilling against the back to minimize spans and to improve regional stability. Application of paste fill, due to the excellent floatability of the paste mass, allows successful accomplishment of this task.

The use of WF for the most part facilitates disposal of waste from development mining, eliminating the need to truck or skip the material to surface waste stockpiles.

INFRASTRUCTURE

SHAFT

A 5.5 m diameter vertical concrete lined shaft, located on the footwall side of the orebody, services the mine. The shaft was recently deepened by 295 m and is 570 m deep. The shaft sinking contractor is currently working on the ore handling system and shaft bottom infrastructure. Completion of this work is expected to take place in the middle of 2006.

The shaft collar is at the 1110 m level and the bottom is at the 540 m level. Current underground access to the shaft is at the 900 m level. After the service ramp connection with the lower part of the mine (mid-2006) is established, additional shaft access will be provided on the 570 m and 540 m levels. The shaft is equipped with two skips, a main cage, and a small cage for hoisting man.

The hoistroom is equipped with two winders. One winder is for hoisting two 5.67 tonne capacity skips and the second one for hoisting main cage. In addition to the skips and man cage, an auxiliary small cage is installed in the shaft for transporting men.

The rated hoisting capacity of the shaft is 260 t of ore/waste per hour. The skips discharge ore and waste into two storage bins. A 200 t bin is assigned to ore and a 50 t bin to waste. Both bins can be used to handle ore.

From the shaft, the ore is transported by two Volvo A35 trucks to the ore stockpile shed located approximately 400 m away. The shed has eight compartments each rated at a 3,500 t capacity.

The headframe is a sheet metal clad steel frame structure erected on a shaft concrete collar. The headframe support legs rest on steel reinforced concrete footings. At present, the top of the headframe is off vertical by approximately 250 mm in relation to the collar. As a consequence, the hoist rope sheave wheels are off the centre line. The hoisting is apparently unaffected by the headframe deformation. Allowing this situation to continue for longer time might cause premature wear of the cage shoes and guides. A monitoring system has been set up at the headframe for measuring and recording any further deformation.

The northwest corner of the hoistroom floor has subsided approximately 500 mm in the last several years. Reportedly, this was caused by neighboring hillside movement or existence of a cavity under the floor. Remedial work recommended by Golder's Associates of the UK, consisting of erecting a rock filled buffer against the hillside, did not slow the subsidence process. Another attempt using extensive grouting under the subsided floor appears to have virtually eliminated the subsidence process. The hoist foundations themselves have been unaffected by this subsidence.

Experts have been engaged and the situation is resolved and subject to monitoring.

RAMPS

The main access to the mine for personnel, equipment, and materials delivery is provided by a 15% inclined service ramp, which was initially driven on the hangingwall side of the orebody. The ramp has a 25 m^2 cross section area and the portal is at the 1096 m level. The ramp also provides access to the top and bottom of the aggregate backfill raises and serves as an exhaust way for ventilation. The upper portion of the ramp is located on the hangingwall side of the orebody down to the 800 m level (Figure 17-1). On the 800 m level, the ramp location was switched to the footwall side from where it continues down to the lowest level in the mine. The service ramp is currently being deepened and will be connected with the new shaft bottom in the middle of the next year.

Additional access to the mine is also provided via an exploration ramp with a nine square metres cross sectional area on a 17% gradient down to the 1000 level. This ramp is used as a return airway and accommodates paste fill delivery pipelines.

EXPLOSIVE MAGAZINE

The explosives magazine is located on the 1040 level. It comprises a 120 m long drift with a 40 tonnes of explosive storage capacity.

REFUGE STATIONS

Refuge stations are provided throughout the mine. The communication system comprises telephones on each level. Crews are also equipped with wireless communication units through a leaky feeder system. There is also a leaky feeder video transmission that is used for the control of filling operations when backfilling stopes with pastefill.

MATERIAL HANDLING

ORE AND WASTE HANDLING

A fleet of six and eight cubic yard LHDs and 25 t trucks handles ore mucking from stopes and development headings. The LHDs work in combination with Wagner MT 400 series mine trucks due to the long haulage distances from most of the development faces and some of the stopes to the ore passes. The LHDs are equipped with ejection buckets and remote control features. Remote stope mucking is mandatory.

Until the middle of 1998 (before commissioning of the shaft) all ore was hauled via the service ramp directly to the ore storage bins on surface. The trucks now dump the ore at the ore pass loading stations on the 960 m, 940 m, and 920 m levels. Haulage distances on the 900 m to 960 m levels do not exceed 200 metres. Ore from the upper levels is hauled to the 960 m level and from the lower levels to the 920 level.

Two ore passes, each with a cross section of 10.2 m², are located in the central pillar of the main ore zone (section N1760) and are part of the mine ore handling system. The ore passes are steel-concrete lined. The two ore pass system allows separate handling of three different ore types. The orebody consists of three types of ore - Yellow, Black, and Clastic ore. Clastic ore currently accounts for about 30% of the total ore mined and is mucked and hauled in campaigns as a separate material.

The ore passes have 400 t capacity each between sublevels and must be pulled constantly. Ore pass hang-ups rarely occur due to the excellent fragmentation of the generally friable ore.

On the 900 level, the ore is transferred by an ST 6C LHD from the ore passes to a feeder hopper with a grizzly, which limits the ore block size to 300 mm x 300 mm. The haulage distance from the two ore passes to the feeder hopper is 25 m and 50 m, respectively. Separation and handling of the different ore types is strictly observed. When the ore type changes, the feeder hopper is emptied completely before another ore type can enter the system.

Waste rock is hauled from the development faces by trucks either directly to stopes, as backfill material, or to surface for stockpiling and future use for backfilling.

AGGREGATE DELIVERY

Two backfill raises for aggregate storage are located in the hanging wall. The raises are 3.1 m in diameter and inclined at 85 degrees. The loading stations for both raises are at the 1080 m level. One raise discharges on the 1040 m level and the second on the 1020 m level.

SHOTCRETE DELIVERY

The mine uses a wet shotcrete, which is prepared on surface at a batch plant located close to the ramp portal. The batched shotcrete is transported in mixer trucks to the underground for application.

MATERIALS

Consumable materials and supplies are transported from surface to underground work places by Paus trucks.

FUEL

There are two diesel fuel stations, one located near the ramp portal on surface and the other at the 800 level next to the underground workshop. All underground mobile fleet, including surface fleet, refuel at this station. The service crew is currently installing a diesel fuel line and an underground fuelling setup in the lower portion of the mine. When commissioned, this system should increase operating availability of the mobile equipment.

COMPRESSED AIR

Four compressors located on surface provide compressed air for the mine. Each unit is powered by a 160 kW motor and delivers 380 l/sec of compressed air at 10 bars pressure. Three pipelines, 100 mm to 200 mm in diameter, distribute compressed air to the different parts of the mine. One line is installed in the shaft, the second in the ventilation raise, and the third in the service ramp.

WATER SUPPLY

Fresh water is supplied to the mine from seven wells on the bank of the Büyük Dere River. The wells have a supply capacity of 450 m^3/h. Although potable water is supplied from Madenli municipality, water for consumption is delivered in special plastic tubs or small bottles. Storage facilities for process water and potable water have a capacity of 1000 m^3 and 60 m^3, respectively. The total water requirement of the mine and the mill is estimated at over 350 m^3/h.

Water is supplied to the underground via three 50 mm and 100 mm diameter pipelines installed in the shaft, ventilation raise, and service ramp.

VENTILATION

Fresh air is delivered to the mine through two downcast raises and the shaft at a combined maximum flow rate of 245 m^3/s. The ventilation raises are equipped with a 220 kW fan each with a maximum capacity of 110 m^3/s. The two ventilation raises are located on the hangingwall side. Each raise has a diameter of 3.6 m and is inclined at 70 degrees. A horizontal drift accesses both of these raises from surface.

Ventilation raise No. 1, located at section N1730 of the mine grid, is serving the mining area between the 900 and 980 levels. Ventilation raise No. 2 (section N1710) bottoms at the 1000 level and ventilates mining areas above the 1000 level only. The shaft is equipped with a 30 kW fan, which provides 25 m^3/s of fresh air through a 900 mm ventilation duct to the bottom of the shaft.

Several raises within the orebody, with diameters between 0.8 m and 3.0 m, serve as exhaust raises. Exhaust air is returned through the top levels of mining areas to the service and exploration ramps to surface.

The ventilation requirements of the mine are calculated based on the total kW of underground diesel equipment in use in accordance with Ontario, Canada regulations (0.06 m^3/s per kW). At the Çayeli mine, Cogema fans are used due to their low noise generation. The ventilation network of the mine is checked on regular basis. Any adjustments and ventilation calculations are performed with the use of VNETPC software.

DEWATERING

The Çayeli mine can be considered a dry mine. Almost half of the underground water is generated from the used process water.

The current main dewatering pump is installed adjacent to the shaft on the 900 level. The recently deepened shaft and shaft bottom development activities are handled through temporary pumping arrangements. After completion of these activities a permanent setup will be installed at the deepest part of the mine to handle water from the lower levels.

All mine water is decanted through drainage holes and ditches to the 900 level. At this level, water is collected in a dirty water sump and is pumped to surface. The dewatering pipeline is located in the shaft. The main dewatering pump is a 132 kW Geho model ZPM 700. The pump can handle 65 m^3/h of dirty water at an operating pressure of 53 bars.

Additionally, a standby pumping arrangement on the 1000 level pumps clear water to surface through a pipeline installed in the exploration ramp.

The total mine water pumped to surface is 25 m^3/h on average. Process water accounts for approximately 40% to 45% of the total mine water pumped to surface.

MAINTENANCE

There is a small underground shop for servicing equipment and handling miscellaneous repairs. Most of the major repairs are done on surface in a large maintenance shop. This shop is properly equipped to handle bigger jobs and can accommodate several pieces of mobile equipment at the same time. Dry facilities and mine offices are located in the same building.

ELECTRIC ENERGY

The mine's electrical main substation is connected by a single 31.5 kV-30 MVA rated overhead power line with the TEK substation north of the town of Madenli. The substation at Madenli is equipped with one 25 MVA transformer and one 10 MVA back up unit.

Electric energy is delivered to the underground by 6.3 kV lines through the shaft and service ramp to the mine 300 or 500 kVA substations located on different levels and sublevels. The feed from the substations to the electrical equipment is reduced to 380 Volt.

MINE EQUIPMENT

The underground mobile equipment fleet is listed in Table 17-1.

TABLE 17-1 MOBILE EQUIPMENT LIST
Inmet Mining Corporation - Çayeli Mine

Type	No of pieces	Description
Jumbos	4	4 Atlas Copco 282 twin-boom
Production Drills	6	2 Tamrock H629 Solomatic 1 Cubex Megamatic ITH 1 Gemsa 2 V-30 Machines Roger Hammers
Bolters	3	2 Memco MacLean 1 Atlas Copco Two Boom & Basket Bolter
LHDs	9	2 Wagner ST8B 3 Wagner ST6C 3 Toro 1400 1 JS 200
Trucks	8	4 Wagner MT 433 (30 t) 4 Wagner MT 4336B (33 t)
Shotcrete Equipment	6	2 Normet Spraymec 6050 WPC 2 Normet Unimixers (5 m^3) 1 Normet Unimixers (7 m^3) 1 Paus mixer
Utility Vehicles	12	3 Fargo scissors-lift trucks 5 Paus platforms 2 Paus ANFO trucks 1 Personnel Carrier 1 Paus Grouting Basket

ORE HOISTING AND TRANSPORT TO SURFACE STOCKPILE

The underground skip loading arrangement consists of two belt conveyors (100 m and 15 m long) with a feed rate of 350 tph each. The conveyors transport the ore from the ore pass feed hopper and discharge it to a 50 m³ surge bin located at the skip loading station. The surge bin feeds two skip measuring bins each having a 5.67 t capacity. The measuring bins load the skips at an average rate of 5.35 t/skip. The rated capacity of the shaft, based on a hoisting speed of 7.7 mps, is 260 tph.

At the shaft headframe, the skipped ore is discharged into storage bins. There are two storage bins. One, having a 200 t storage capacity, is assigned to ore and the other, with a 50 t storage capacity, is allocated to the waste rock. Both bins can be used for ore storage.

From the shaft the ore is transported by Volvo A35 trucks with a payload of 32 t to the ore stockpile. The transport distance is approximately 400 m.

MINERAL PROCESSING

COPPER ZINC PROCESS PLANT

During mining, ore is separated into three different types and hoisted to surface, where it is transferred into one of the eight storage compartments. Each compartment has a 3,500 t storage capacity. Mill feed is prepared by blending various types of ores according to their metallurgical composition and copper and zinc grades.

In 2005, the average feed grades were 3.84% copper and 5.74% zinc. Concentrate grade averaged 22.81% for copper at a 82.64% recovery and zinc concentrate grade averaged 50.39% at a 75.24% recovery. In the last five years, the average recovery of copper has been 86.05% and of zinc 72.12%.

The ore processing facility consists of conventional crushing, grinding, differential flotation, and pressure filtration. The facility is equipped with an online Yokogawa process control system (Figure 17-3).

TABLE 17-2 MILL PRODUCTION – 2001-2005

Inmet Mining Corporation - Çayeli Mine

Year	Milling	Copper Concentrate				Zinc Concentrate		
	Tonnes	Tonnes	% Cu	g/t Au	g/t Ag	Tonnes	% Zn	g/t Ag
2001	816,480	135,957	24.25	0.70	66.36	50,129	50.52	121.63
2002	895,423	132,133	24.65	0.80	78.62	64,778	51.06	126.42
2003	927,892	135,923	24.64	0.77	73.99	65.853	51.03	116.40
2004	765,329	112,915	23.71	0.85	85.35	65,713	50.77	121.44
2005	833,638	116,054	22.81	1.19	96.36	83,903	50.39	126.20
Total	**4,238,763**	**632,982**	**24.06**	**0.85**	**79.45**	**330,376**	**50.74**	**122.65**
Aver age	847,753	126,596	24.01	0.86	80.14	66,075	50.75	122.42

TABLE 17-3 MILL RECOVERIES – 2001-2005

Inmet Mining Corporation - Çayeli Mine

Year	Copper recovery %	Zinc recovery %
2001	84.99	65.88
2002	86.68	71.96
2003	86.54	71.71
2004	89.41	75.82
2005	82.64	75.24
Average	86.055	72.12

Based on the requirements of the concentrator, the mine carries out segregation of three ore types, Clastic, Black, and Yellow. Clastic ore is material that, based on visual estimates, contains more than 10% sphalerite fragments in massive sulphides. These fragments are generally fine-grained and form intergrowths with chalcopyrite, which may affect the copper recoveries.

The Black and Yellow ore types are based on the contained zinc grade and are mined separately to allow for optimal grade blending from the stockpile bins on surface. These two ore types are referred to as “Spec ore”. Black ore is defined as material with more than 4.5% Zn and a Cu/Zn ratio of less than one. Yellow ore consists of copper-rich, zinc-poor sulphides. Yellow ore is comprised of approximately one half massive sulphides and one half stockwork material (stockwork mineralization generally contains very little sphalerite).

Two copper concentrates are produced:

- Spec concentrate. In general, a tonne of Yellow Ore combined with approximately 0.42 t of Black Ore yields Spec concentrate.

- Non-Spec concentrate. The concentrate is comprised of Clastic Ore and the balance of Black ore. Approximately 10% of copper concentrate is generated from this type of ore. The concentrate is high in precious metals and also contains higher zinc. The number of customers for this type of product is more limited.

In the 2006 Five Year Milling Plan, 40% of ore mined will yield Spec concentrate and 60% will yield Non-Spec concentrate. The average grades of Spec ore feed are 4.3% Cu and 5.1% Zn. The average grades of Non-Spec ore feed are 3.4% Cu and 7.2% Zn. The average head grades for both types of feed are 3.7% Cu and 6.4% Zn. The NSR per tonne of ore for Spec ore feed is $80, while the NSR per tonne of ore for Non-Spec ore feed is $67. Using the above ratio, the average NSR per tonne of ore is $72.

Copper and zinc concentrates are loaded onto trucks for transportation to the Rize port. Each truck is weighed at the plant gate prior to transporting the concentrate.

PROCESS DESCRIPTION

CRUSHING

Crushing is done in three stages. A jaw crusher located close to the ore storage bins does the first stage crushing. The crushed ore is transported from the crusher to a double deck screen by a conveyor. The top screen oversize is sent to a secondary cone crusher. The bottom screen oversize (plus 12 mm) feeds a tertiary cone crusher. The cone crushers operate in closed circuit. Discharges from the cone crushers plus material from the jaw crusher are combined and returned to the double deck screen by a common belt conveyor. Undersize from the screen is conveyed to a 2,500 t capacity fine ore bin. A wet scrubber controls the dust generation in the facility.

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GRINDING

The fine ore is delivered to a 560 kW 3,200 mm diameter by 4,300 mm long ball mill by a belt conveyor. Primary and secondary mill discharges are combined and enter a cyclone unit. The cyclone underflow feeds a 2,100 kW (4,400 mm diameter by 7,200 mm long) secondary ball mill. Both mills are rubber lined. The overflow from the cyclone goes to the copper rougher circuit as a flotation feed. The grinding circuit produces a flotation feed of 70% passing -36 µm. A collector and lime are added to the primary ball mill. Power consumption is 19 KWh/t of ore and steel consumption is 1.9 kg/t of ore.

COPPER FLOTATION

The copper flotation consists of conventional copper rougher and scavenger circuits. The copper rougher circuit comprises three16 m^3 Outokumpu free flow cells followed by four 16 m^3 Outokumpu scavenger cells. Frother is added in various points to flotation cells.

COPPER ROUGHER AND ROUGHER CLEANER CIRCUIT

The copper rougher concentrate from the first three rougher cells is pumped to the first-stage copper rougher cleaning cells, consisting of three 8 m^3 and two 16 m^3 Outokumpu free flow cells. The copper rougher cleaner concentrate is pumped to the second stage consisting of three 8 m^3 free flow Outokumpu cells. The first and second stage rougher cleaning cells are in closed circuit.

The copper rougher cleaner concentrate from the second stage, which is floated in the three 8 m^3 Outokumpu cells, is fed to six 3 m^3 Outokumpu cells, while its tail is circulated back to the first-stage rougher cleaner. The concentrate from the six 3 m^3 cells is the final copper concentrate, while its tails are pumped to the regrinding circuit.

COPPER ROUGHER SCAVENGER AND COPPER SCAVENGER CLEANER CIRCUIT

The copper rougher scavenger concentrate is combined with copper cleaner and tails and goes to the copper regrinding circuit. The copper rougher scavenger tail is fed to a zinc rougher circuit.

The copper regrinding cyclone cluster consists of ten 150 mm diameter cyclones. The cyclone underflow is reground in the mill, and cyclone overflow gravitates to copper scavenger cleaner conditioner to which collector is added. Overflow from the conditioner gravitates to eight Outokumpu copper scavenger cleaner cells. The concentrate from these cells is pumped to six free flow cells for second stage cleaning. The copper scavenger cleaner tails are combined with copper rougher scavenger tails and are fed to the zinc rougher circuit. The concentrate from the second stage scavenger cleaner cells is transferred to two cells for final stage cleaning, while its tails are circulated to the copper regrinding circuit.

The concentrates from the copper scavenger cleaner circuit and copper rougher cleaner circuit are combined as final copper concentrate and pumped to the copper dewatering circuit.

ZINC FLOTATION

The copper rougher scavenger and copper scavenger cleaner first tails are combined and pumped to two agitating zinc flotation conditioners. Lime is added to the tail box of copper rougher scavenger cells. Copper sulphate is added to the first conditioner and collector to the second. Overflow from the second conditioner gravitates to three 38 m^3 Outokumpu flotation cells for zinc rougher flotation. The zinc rougher tail goes to tail pump box. The zinc rougher concentrate is transferred to the cleaner conditioner. After conditioning, zinc rougher concentrate is sent to zinc cleaner cells. Tails from conditioning cells are sent to column cleaner scavenger cells.

COPPER AND ZINC CONCENTRATE DEWATERING

The copper and zinc concentrate slurries are thickened in two separate identical circuits. Each circuit consists of a 16.0 m diameter conventional thickener. The thickener underflows are transferred to two surge tanks. Pressure filters are used for filtration of the thickened copper and zinc concentrates. There are four pressure filters. Two of them are used for copper, one for zinc, and the fourth is a standby unit for either copper or zinc filtration. The final concentrates contain 8% moisture. They are well within the flow moisture limits of 11.11% and 11.45% for copper and zinc concentrates, respectively.

Filtered copper and zinc concentrates drop directly down to a concrete reinforced 1,000 t capacity loadout area. The concentrates are transported to the Rize port by trucks.

The port is located approximately 28 km from the mine site. The concentrate trucking is under a contract with a local contractor.

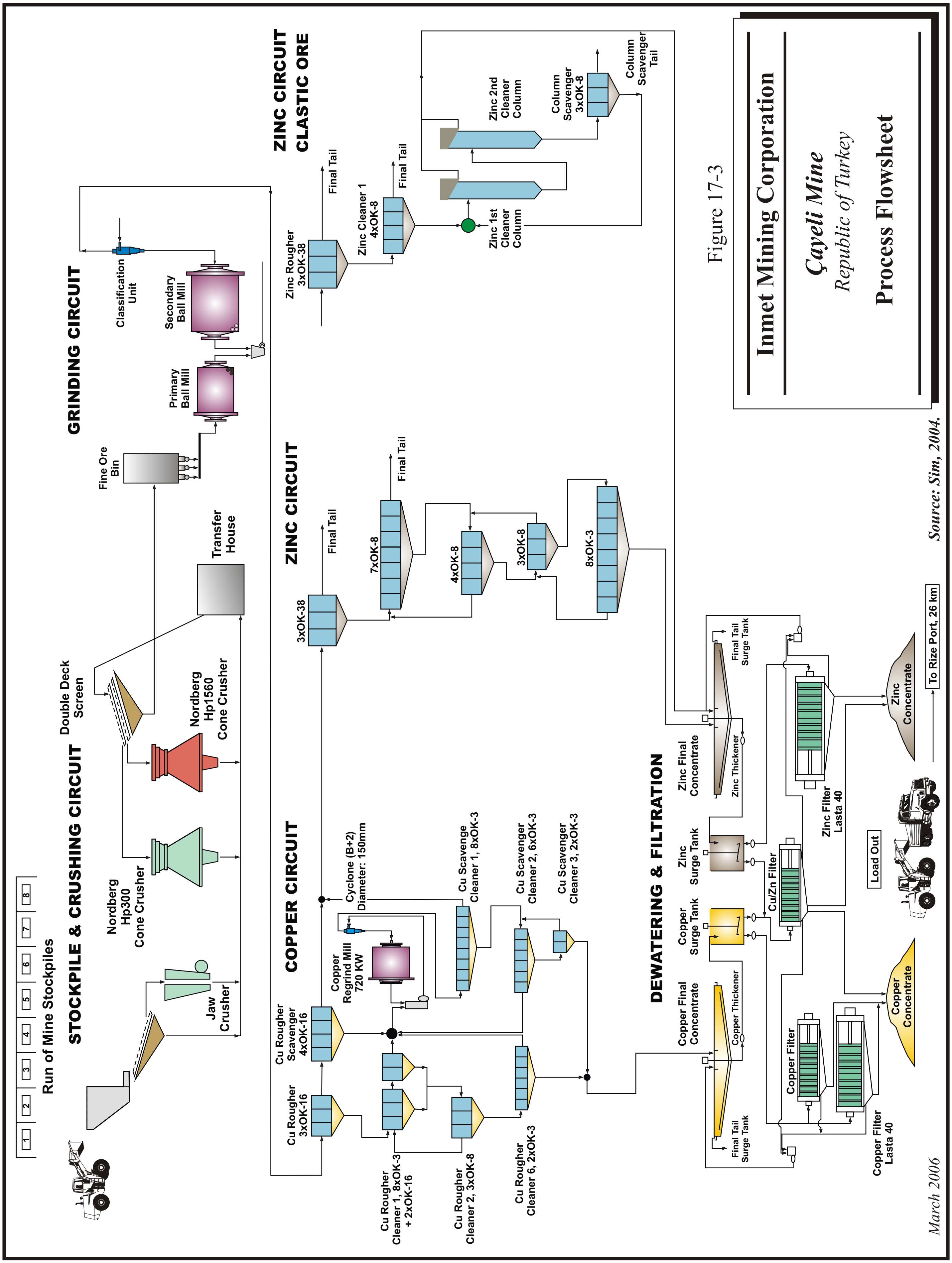


Figure 17-3

Inmet Mining Corporation

Çayeli Mine
Republic of Turkey

Process Flowsheet

PASTE FILL PLANT

The plant is designed to deliver paste mass to the underground at a rate of 45 m^3/h. The plant building is positioned close to the service ramp portal and approximately 300 m away from the concentrator building.

Tailings from the concentrator are pumped to the paste fill plant thickener at 20% solids by weight. The density of tailings in the thickener is increased from 20% to 65-75% solids by weight. The underflow from the thickener is pumped to an agitating tank. Depending on a required paste fill rate, the agitating storage tank underflow is pumped to one or both of the two vacuum disc filters.

The disc filters reduce the moisture content of the tailings to approximately 85% to 86% solid by weight. The discharged tailings cake from the disc filters drops on a reversible belt conveyor, which delivers them to a surge hopper. The filtered cake is agitated and mixed with water or slurry in a conditioner tank until it obtains a 7” to 8” slump. After that, cement is added to the paste mass. The cement content is governed by the specific requirements of each stope and varies from 3% to 9%. Average cement content is 5%.

The final paste mass is pumped by two piston type Putzmeister pumps through a pipeline for placement in the underground. The underground paste fill system is equipped with two main pipelines. A single operator controls the plant from a control room equipped with a computerized monitoring system. Each hour, a sample is taken for the paste slump control. TV cameras are used for monitoring paste placement in stopes.

ENVIRONMENTAL CONSIDERATIONS

The site has an environmental plan in place to control dust and effluent emissions. The plan is implemented in accordance with the Turkish Mining and Environmental regulations and requires the operation to monitor on a regular basis the effluents at the mine site. The results of effluent monitoring are reported to authorities on a quarterly basis.

During the site visit, RPA checked the operation's environmental records for the last two years of operating. RPA found that the operation maintained proper records in regard to controlling and monitoring the effluents and was in compliance with the regulations. There were no reported violations of the regulations with regard to dust control, quality of water discharged from the mine, maintaining acid generating rock storage area, noise control at the site, and tailings discharge in the Black Sea.

At the present time, the relevant Turkish Mining and Environmental Legislation does not require CBI to prepare a closure plan for the mine, however, CBI has a policy to incorporate effective environmental management practices into all of its business functions and, as part of its corporate Safety, Health and Environmental Policy.

In 1999, CBI commissioned SRK UK Ltd. to generate a closure plan for the mine. The final report, Decommissioning and Closure Plan for the CBI Copper/Zinc Mining Operation in Rize, Turkey, was received in November 1999. The closure activities have been broken down into five main component areas:

- Underground mine
- Process plant and general site infrastructure
- Waste rock dumps
- Port facilities
- Tailings disposal system

For each component area, the main decommissioning and closure issues are highlighted along with their respective performance criteria, closure investigations and test work, available closure options, closure actions, provisions for the demonstration of closure performance, and the costs of the closure activities.

The decommissioning and closure costs are summarised in the Table 17-4.

TABLE 17-4 SUMMARY OF DECOMMISSIONING AND CLOSURE COSTS
Inmet Mining Corporation - Çayeli Mine

Closure Phase	Area	Closure Cost ('000 US$)
Decommissioning	Administration	575
	Underground mine	779
	Process Plant and general infrastructure	1,903
	Hydrological issues	343
	Environmental issues	141
	Rollover	897
	Power	177
	Contingency	980
Post-Closure (0-5years)	Site rehabilitation	189
	Monitoring	120
	Long-term management	544
	Contingency	213
TOTAL		**6,862**

The plan anticipates a decommissioning and closure period of some 12 months for all site infrastructure and the underground mine, and five years for site rehabilitation and environmental compliance monitoring. The base-case scenario that has been evaluated in terms of costs assumes that all infrastructure will be decommissioned, demolished, and removed.

FIVE YEAR MINE PLAN

The Çayeli operation does not have a life of mine plan. The operation has developed a preliminary five-year mining plan, aimed at mining one million tonnes per year of ore for the next five years. RPA recommends developing a life of mine plan for the operation based on mineable reserves. When complete, such plan should be a subject to reviews and updating on a regular basis.

Tables 17-5 and 17-6 summarize yearly production and development schedules to support the planned production from 2006 to 2010.

TABLE 17-5 FIVE YEAR PRODUCTION PLAN (PRELIMINARY)
Inmet Mining Corporation – Çayeli Mine

Years	2006	2007	2008	2009	2010	TOTAL
Tonnes	980,000	1,000,000	1,000,000	1,000,000	1,000,000	4,980,000
% CuEq	5.95	6.44	5.85	5.91	6.19	6.07
Zn/Cu ratio	1.65	1.61	1.57	1.84	1.89	1.71
% Cu	3.69	4.03	3.70	3.52	3.64	3.72
% Zn	6.11	6.51	5.81	6.47	6.88	6.36
% Pb	0.33	0.33	0.26	0.35	0.42	0.34
g/t Ag	47	48	39	45	56	47

TABLE 17-6 FIVE YEAR DEVELOPMENT PLAN (PRELIMINARY)
Inmet Mining Corporation – Çayeli Mine

Development Type	2006	2007	2008	2009	2010	TOTAL
Ore development (m)	924	980	1,200	688	402	4,194
Sill drift development (m)	1,810	1,024	1,169	1,119	1,574	6,697
Waste development (m)	1,383	1,369	21	0	0	2,773
TOTAL	**4,117**	**3,373**	**2,390**	**1,808**	**1,976**	**13,665**

RPA reviewed the 2006-2010 preliminary mining plan and, in general, agrees with the proposed production and development schedules. The schedules are based on average development and stoping productivities achieved in the last two years. The increased production targets also take into account the expected higher ore handling capabilities to be realized after completion and commissioning of the new ore handling and hoisting infrastructure in the middle of 2006.

In the last four years, part of the production tonnage increase came from below the hoisting arrangements of the shaft. This situation required trucking some ore from stopes and development works on the lower levels to the upper part of the mine for further hoisting to surface. In spite of this constraint, the mine was able to produce, on average, approximately 800,000 tonnes of ore per year.

In the middle of 2006, the ore hoisting and handling system is expected to be ready for handling production and development of ore from the entire mine, including its lowest parts. This will substantially improve the capability of the mine to produce and handle higher tonnages.

The physical setup of the mine and the orebody configuration, including reserves, are capable of proposed yield for the next five years. Additionally, the mine has a very capable and highly trained workforce and the appropriate equipment fleet to deliver the planned production quotas.

CAPITAL AND OPERATING COST ESTIMATES

The original capital cost of the project in 1994 was $200M. The operation typically runs an annual sustaining capital budget between $6M to $8M, for items as equipment replacement and incremental project improvements. In 2003 a number of projects were initiated to improve recovery at the processing plant. In 2004, a major underground expansion program at a total estimated cost of $17M was undertaken. This program included shaft deepening, excavation of the shaft new infrastructure, ore handling system, and service ramp extension.

The shaft deepening has been completed. The work associated with the development and installation of the shaft underground infrastructure, ore handling system, and service ramp extension were advancing on schedule at the time of RPA's site visit. Completion of this expansion program is expected to take place some time in the middle of 2006.

CAPITAL EXPENDITURES IN 2001-2005

Table 17-7 summarizes the mine capital expenditures between 2001 and 2005.

TABLE 17-7 CAPITAL EXPENDITURES - 2001-2005
Inmet Mining Corporation – Çayeli Mine

Years	Capital Expenditures (US$ M)
2001	4.697
2002	7.965
2003	5.910
2004	16.634
2005	18.722

CAPITAL EXPENDITURES BUDGET FOR 2006

The 2006 capital expenditures budget breakdown by department is summarized in Table 17-8.

TABLE 17-8 CAPITAL EXPENDITURES BUGDET IN 2006
Inmet Mining Corporation – Çayeli Mine

Department	Capital Expenditures (US$ M)
Mine	$2.4
Shaft Project	$10.8
Mill	$2.2
General and Administration	$0.2
Deferred development	$4.8
Total	**$20.6**

The deferred development capital covers primary underground development such as ramps, main levels, and underground infrastructure development.

OPERATING COSTS

The operating costs from 2001 to 2005 are summarized in Table 17-9.

TABLE 17-9 OPERATING COSTS PER TONNE MILLED - 2001-2005
Inmet Mining Corporation – Çayeli Mine

	2001	2002	2003	2004	2005
Tonnes Milled	816,480	895,423	927,892	765,329	833,638
Mine	$12.37	$13.60	$17.20	$23.97	$26.48
Mill	$8.51	$9.16	$10.59	$12.98	$12.86
Administration	$8.05	$8.70	$12.54	$16.65	$17.27
Total Cost	**$28.93**	**$31.46**	**$40.33**	**$53.60**	**$56.61**

The operating costs include concentrate delivered to the port at Rize. In the last three years, operating costs have significantly increased at the operation. This is mostly attributed to a Turkish Lira appreciation against the US dollar and major wage and salary increases that took place at the same time. Additionally, an increase in energy, material, and part costs contributed to the operating costs increase.

CUT-OFF GRADE

NSR TERMS

The Çayeli operation produces zinc concentrate and two types of copper concentrates, regular "Spec" copper concentrate and a high precious metal copper concentrate. This second copper concentrate, which is produced during Clastic ore processing campaigns, has higher zinc content, the penalties of which are generally offset by elevated silver and gold contents.

The treatment and refining charges (TC and RC, respectively) for the copper concentrates are summarized as follows:

Base TC = $90 per tonne

RC = $0.09 per lb Cu

Deductions = 1% copper concentrate grade
30 g silver
1 g gold

The treatment and refining charges for the zinc concentrate is summarized as follows:

Base TC = \$175 per tonne

Deductions = 8% zinc concentrate grade

93 g silver

Treatment charges for both copper and zinc are subject to escalation and de-escalation based on actual metal prices. The average freight cost from Rize to European customs is \$37 per tonne for copper and \$27 per tonne for zinc.

NSR values and copper equivalent are shown in Table 17-10.

TABLE 17-10 NSR VALUE/TONNE – 60% NON-SPEC & 40% SPEC ORE
Inmet Mining Corporation – Çayeli Mine

	Five Year Milling Plan – 2006-2010		
	Non-Spec Ore 60%	**Spec Ore 40%**	**Average**
Head Grade Cu%	3.35	4.29	3.72
Head Grade Zn%	7.20	5.07	6.36
Zn/Cu ratio	2.15	1.18	1.71
NSR/tonne of ore	\$67	\$80	\$72
% EqCu	6.11	5.72	5.52
NSR/1% EqCu	\$10.97	\$14.03	\$12.19
Metal Prices (US \$)	Cu = 1.10/lb, Zn = 0.55/lb, Ag = 5.60/oz, Au = 450/oz		

For the purposes of resource and reserve estimation, and since copper is the most important metal in terms of its contribution to the NSR, CBI had been using, until December 2005, an equivalent copper grade (EqCu%) as a common parameter from which the cut-off is derived. In December 2004, the EqCu formula was:

$$EqCu\% = Cu\ \% + 0.37*Zn\%$$

The EqCu formula is derived by setting a head grade (generally, the reserve grade) and a recovery factor for each metal. By doing so, blocks in the model will have lower head grades than that used to determine the EqCu formula and a higher NSR than the actual NSR because they are based on a recovery factor that is not appropriate. Conversely, blocks with higher head grades than that used to determine the EqCu formula will use recovery factors that are lower than actual. In this case the end result is a lower NSR than actual.

At the time of the site visit, RPA recommended using NSR values, rather than EqCu values, for resource and reserve reporting, cut-off determination, and stope design. The NSR value for each block of the model is calculated on the basis of its grades (Cu, Zn, Ag, Au) and metal recoveries. Metal recoveries are obtained from the recovery curves based upon head grades. The mill can supply such curves.

In January, CBI has converted its EqCu formula into a NSR formula without considering variable metal recovery, which is based on variable head grades. By using variable metal recoveries, the determination of NSR values for each block of the model implies a more elaborate formula in comparison to a simplistic NSR or EqCu that is based on fixed metal recoveries. By using an average recovery, lower grade material will present a higher NSR than actual. At the time of stope design, marginal ore will have a higher NSR value than actual and, thus, may be part of the design. As the Çayeli massive sulphides have quite a high NSR value, this is a concern on the footwall side only, where the grades are gradational from massive sulphides into stockwork ores. RPA recommends verifying this issue in this particular case.

RESOURCE AND RESERVE REPORTING VERSUS NSR

CBI has determined that the resources, outside reserves, should be estimated using a lower cut-off of $US 35 per tonne, in order to inventory material with potential to be mined at some future date, should higher metal prices prevail.

For reserve reporting over a long term, CBI uses a US$ 46 per tonne of ore cut-off to determine the mining limits. RPA agrees with this assumption, given that the current 2006 Five Year Plan operating costs is forecast at $52 per tonne. The new Deep Ore infrastructure (shaft extension, ore passes, ventilation raise, ramp), although not presently quantified, is expected to reduce operating costs and increase production. Material between US$ 46 and US$ 52 could be considered as incremental ore.

RPA recommends that the 2006 Resources and Reserves be reported at the same US$ 46 per tonne of ore cut-off.

RPA also recommends reporting mineral resources exclusive of mineral reserves as per CIM recommendations and Inmet standards.

MANPOWER

The Çayeli mine employs 467 people. Most of the workers are unionized. Table 17-11 shows the breakdown of manpower by department.

TABLE 17-11 MANPOWER
Inmet Mining Corporation – Çayeli Mine

Department	Number of People
Mine	196
Mill	94
Maintenance	49
Administration	87
Technical (includes diamond drillers)	41
Total	**467**

MARKETS/CONTRACTS

The Çayeli mine produces between 115,000 and 135,000 tonnes of copper concentrate and 65,000 and 80,000 tonnes of zinc concentrate each year. Approximately 10% of the copper concentrate is generated from the Clastic mill feed producing concentrate high in precious metals and also containing higher zinc. The number of customers for this type of product is limited.

CBI markets its products to numerous consumers in Europe, North America, as well as Central and Eastern Asia. On average, 75% of sales are realised through long-term contracts and the remaining 25% is sold on the open markets.

RPA considers the smelter terms at Çayeli to be consistent with industry standards.

ROYALTY AND TAXES

CBI pays an annual royalty to Eti Holding of a minimum of $667,000/yr or 7% of net after tax profit. The royalty is separate from the stock ownership (which Inmet acquired from ETI) and relates to the underlying ownership of the mining lease. The "net after tax profit" is based on the Turkish Lira statements, not the US dollar statements.

Municipal taxes are 2%. Additionally, there is 30% Turkish corporation tax, which can be reduced through specific investment incentives, such as certain types of capital projects. Withholding tax on dividends is 10%.

18 INTERPRETATION AND CONCLUSIONS

RPA agrees with the methodology used by CBI for mineral resource and mineral reserve estimation. The estimates use well-interpreted 3D-mineralization solids, a good database, reliable assays, and block grade estimation techniques that are consistent with industry standards. All of the above has contributed to good reconciliation between mine and mill over the years. RPA confirms that the mineral resources and mineral reserves estimated at Çayeli are in accordance with National Instrument 43-101.

The major underground expansion program that includes shaft deepening (completed), excavation of the shaft new infrastructure, ore handling system, and service ramp extension is well advanced. The work associated with the development and installation of the shaft underground infrastructure, ore handling system, and service ramp extension was advancing on schedule at the time of the site visit. This program, once completed in the middle of 2006, will allow for sustained productivity from the lower parts of the deposit and will give access for exploration at depth.

In RPA's opinion, the Çayeli property has very good exploration potential both from surface and underground. CBI retains large land holdings surrounding the mine, which host almost 15 km of the stratigraphic strike equivalent to the Çayeli ore horizon. Much of this ground remains essentially unexplored at this time.

The mine is sensitive to NSR parameters such as metal prices and operating costs. Based on long term copper and zinc prices ($1.10 and $0.55, respectively), the margin between the average NSR value and the 2005 operating cost per tonne of ore is $16 ($72 vs. $56). At current prices (Cu = $2.30/lb, Zn = $1.00/lb, Ag = $9.50/oz, Au = $550/oz), the NSR value is $165, which gives a margin of approximately $110 per tonne of ore.

19 RECOMMENDATIONS

Although RPA does not see problems that could represent material issues, we recommend the following:

1. **Land Tenure.** The mine and surrounding property comprise three separate leases. One of them, Licence OIR-10627, will expire in 2006. According to the mine geologists, Licence OIR-10627 is important for exploration. RPA recommends that, in this particular case, mine geologists be directly involved in the process of extending the expiration date of this lease because the Turkish government may require that geological reports be filed.

2. **RQD Database**. The RQD (Rock Quality Designation) values are not currently in a database. RPA recommends entering RQD values into a database, which, in combination with rock types, ore types, alteration zones, and microseismic data, may be very useful for development heading planning, stope design, and ground support.

   RPA also recommends using the RQD values to assess, on a stope by stope basis, if the 'in-time' drifting, production drilling, blasting, and mucking philosophy is applicable.

3. **Density**. Density determinations are not conducted for stope definition drill holes, channel samples, delineation drill holes in which sludge is recovered, or in the development sludge. In these cases, average values are assigned based on the rock code designation.

   RPA recommends that density be determined by an interpolation technique, such as kriging, rather than assigning a density value based on rock codes. Density would then be based on more local values rather than on average or regional values.

4. **QA/QC.** CBI reports that the lower of the original and checked values is used for grade estimation and that there is no field for check assays in the database. RPA recommends using the average of the original and checked values for grade estimation and having a field in the database for check assays. A database should be set up for check assays that would contain sample type, original values, checked values, and dates. The check assay results should be included in a Mineral Resource and Mineral Reserve Estimates Technical Report. RPA also recommends that results be presented on XY graphs.

   RPA recommends that check assay programs carried out on geology samples be more closely followed by geologists. Check assays should also be requested by geologists, not only by the laboratory manager.

5. **Assay Statistics by Domain**. RPA recommends that statistics on raw data (prior to compositing) be presented for each wireframe (zone) without

application of any cut-off. Statistical reports, histograms, and probability plots should be presented as backup in a Mineral Resource and Mineral Reserve Estimates report.

6. **Variography**. RPA recommends that variography be done for silver and gold.

7. **Capping of High Grade Samples.** RPA has reviewed the Cu-Zn probability plots that have been produced for each of the mineralized domains and recommends that further consideration be given to the use of grade capping, versus outlier limitation, as a method of addressing high grade issues.

8. **NSR vs. EqCu**. Until December 2005, CBI had been using an equivalent copper grade (EqCu %) as a common parameter from which the cut-off grade was derived. The EqCu formula was derived from NSR calculations, in which a fixed head grade (generally, the reserve grade) and a fixed recovery factor were assigned to each metal.

   RPA recommended using NSR values, rather than EqCu values, for resource and reserve reporting, cut-off determination, and stope design. NSR calculations consider variable metal recovery based on head grades.

9. **Cut-Off Grade**. The current 2006 Five Year Plan operating costs are forecast at $52 per tonne, and experience is $56. In this estimate, Inmet used a cut-off grade of $46, or an EqCu of 3.3%, which, in RPA's opinion, is appropriate on an incremental basis.

10. **Block Model Grade Estimation**. Copper and zinc grades in the block model are estimated using the Ordinary Kriging interpolation method. Silver, gold, lead, and density are estimated using the Inverse Distance weighting method.

    Each domain zone is treated based on "hard" boundary limitations, which means that block grades could only be estimated using samples located inside that specific zone (for example, the Main Zone Clastic Ore (zone 1) block grades were estimated using only samples located inside zone 1).

    RPA generally agrees with the 'hard' boundary approach, which is based on ore type, however, RPA recommends being very careful in this approach in the southern part of the deposit where there is no displacement of the mineralized zone by the Scissor Fault. RPA recommends that a 'box' be outlined in the southern part of the deposit in the vicinity of the Scissor Fault where the 'hard' boundary is not applicable. For example, RPA has observed significant differences in grades for two adjacent blocks that belong to the same ore type (Clastic Ore) but different domains (one block being in the Main Zone and the other block in the Deep Zone), and where the Scissor Fault shows no apparent displacement. In this particular case, the block on the hanging wall of the fault had an estimated grade of 12.62% Zn, while the adjacent block on the footwall side of the fault had an estimated grade of 4.10% Zn. Grades are overestimated on one side of the fault and underestimated on the other.

11. **Grade Reconciliation**. CBI reports that short-term grade reconciliation (daily, weekly, monthly) between the geology model and mill grades is essentially based on the adjustment of stope broken ore grades to mill grades.

    Short-term grade reconciliation is not easy when ore is stockpiled in different piles on surface before being sent to the mill, therefore, RPA recommends the following:

    - In order to be coherent in the process of grade reconciliation, the forecast diluted grade of stopes should be compared to the mill.

    - The broken ore stockpile mine/mill grade ratio should be used to back-calculate stope grades. Stockpile grades are currently not used for grade reconciliation.

    - A more detailed study on broken ore sampling should be conducted in order to see if the sampling quantity is appropriate, or whether additional sampling is necessary. Also, broken ore sample grades are generally higher than mill grades. Such results suggest that broken ore sampling should be revisited.

    Long-term grade reconciliation should be carried out between the mill heads and the following:

    - Broken ore samples.
    - Stopes that have been surveyed using a Cavity Monitoring System (CMS).
    - Diluted block model.
    - Undiluted block model.

12. **'In-time' Stope Preparation**. The stope preparation process is based on the "in time" drifting, production drilling, blasting, and mucking philosophy. The reason for this is that, due to the relatively poor ground conditions, the excavations often deteriorate quickly.

    RPA recognizes the need to respect the potential for rapid deterioration of prematurely excavated openings. The in-time preparation philosophy is an appropriate measure to deal with this problem, however, it slows stoping productivity and ties mine resources to a given extraction area in a very unproductive way.

    RPA recommends utilizing stope delineation drilling for testing RQDs and assessing, on a stope by stope basis, if a given stope qualifies for an exemption from the "in time" development and can then be prepared and extracted in a sequential order, without interruption.

13. **Ground Support**. RPA agrees that the ground support measures adopted at the Çayeli mine are working well and, in general, are appropriate.

14. **Five Year Mine Plan**. The Çayeli operation does not have a life of mine plan. The operation developed a preliminary five-year mining plan aimed at mining one million tonnes per year of ore for the next five years. RPA recommends updating the life of mine plan for the operation based on mineral reserves.

20 REFERENCES

Babacan, H., 2001: A Mine Site Laboratory from Exploration to Mine Closure: A Case Study (Çayeli Bakir Isletmeleri). Paper presented at the 17th International Mining Congress and Exhibition of Turkey – IMCET 2001.

Inmet Mining Corporation, 2000: Review of the Russian Adit Resource - Çayeli Deposit, Çayeli, Turkey. Internal report (5 pages).

Inmet Mining Corporation, 2003: Data Check Procedure. Internal memo (2 pages).

Inmet Mining Corporation, 2003: Geology Database Procedure. Internal memo (1 page).

Inmet Mining Corporation, 2004: Technical Report on the Mineral Resource and Mineral Reserve Estimates at the Çayeli Mine, Turkey - Statement as of December 31, 2004. Internal report (85 pages).

Inmet Mining Corporation, no date: Monthly Reconciliation and Report Preparation Procedure. Internal draft memo (1 page).

21 SIGNATURE PAGE

This report titled "Technical Report on Mineral Resource and Mineral Reserve Estimates of the Çayeli Mine, Turkey", prepared for Inmet Mining Corporation, Toronto, Canada, and dated March 30, 2006, was prepared and signed by the following authors:

	(Signed and Sealed)
Dated at Toronto, Ontario March 30, 2006	Graham G. Clow, P. Eng. Principal Mining Engineer Roscoe Postle Associates Inc.
	(Signed and Sealed)
Dated at Toronto, Ontario March 30, 2006	Andrew H.Hara, P.Eng. Associate Mine Engineer Roscoe Postle Associates Inc.
	(Signed and Sealed)
Dated at Toronto, Ontario March 30, 2006	Bernard Salmon, Eng. Consulting Geological Engineer Roscoe Postle Associates Inc.

22 CERTIFICATE OF QUALIFICATIONS

GRAHAM G. CLOW

I, Graham G. Clow, P.Eng., as an author of this report entitled "Technical Report on Mineral Resource and Mineral Reserve Estimates of the Çayeli Mine, Turkey" prepared for Inmet Mining Corporation and dated March 30, 2006, do hereby certify that:

1. I am Principal Mining Engineer with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

2. I am a graduate of Queen's University, Kingston, Ontario Canada in 1972 with a Bachelor of Science degree in Geological Engineering and in 1974 with a Bachelor of Science degree in Mining Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 8750507). I have worked as a mining engineer for a total of 32 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a consultant on numerous mining operations and Projects around the world for due diligence and regulatory requirements
    - Senior Engineer to Mine Manager at seven Canadian mines and projects
    - Senior person in charge of the construction of two mines in Canada
    - Senior VP Operations in charge of five mining operations, including two in Latin America
    - President of a gold mining company with one mine in Canada
    - President of a gold mining company with one mine in Mexico

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I have not visited the Çayeli mine.

6. I am responsible for overall preparation of the Technical Report.

7. I am independent of the Issuer applying the test set out in Part 1.4 of National Instrument 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

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

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated 30th day of March, 2006.

(Signed and Sealed)
Graham G. Clow, P. Eng

BERNARD SALMON

I, Bernard Salmon, Eng., as an author of this report entitled "Technical Report on Mineral Resource and Mineral Reserve Estimates of the Çayeli Mine, Turkey" prepared for Inmet Mining Corporation and dated March 30, 2006, do hereby certify that:

1. I am Consulting Geological Engineer with Roscoe Postle Associates Inc. (RPA). My office address is 46, 18th Street, Rouyn-Noranda, Quebec, J9X 2L5.

2. I am a graduate of Ecole Polytechnique, Montreal, Qubec, Canada, in 1982 with a Bachelor of Science (Applied) in Geological Engineering.

3. I am registered as an Engineer in the Province of Quebec and I am designated as a Consulting Geologist. I have worked as a geological engineer for 23 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Mining geologist, Falconbridge Copper Corp.,Opemiska Mine, 1982 to 1987.
   - Chief geologist, Minnova Inc., Ansil Mine, 1987-1992
   - Chief-Geologist and Technical Superintendant, Inmet Mining Inc., Troilus Mine, 1992-1997.
   - Chief-Geologist, Aur Resources Inc., Louvicourt Mine, 1997-2005.
   - Consulting Geologist with RPA from 2005 to present.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I visited the Çayeli Mine on December 6-9, 2005.

6. I am responsible for preparation of Items 2 to 16, and parts of Items 1, 18, and 19.

7. I am independent of the Issuer applying the test set out in Part 1.4 of National Instrument 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

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

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated 30th day of March, 2006.

(Signed and Sealed)
Bernard Salmon, Eng.

ANDREW HARA (HARASIMOWICZ), P. ENG.

I, Andrew Hara (Harasimowicz), P.Eng., as an author of this report entitled "Technical Report on Mineral Resource and Mineral Reserve Estimates of the Çayeli Mine, Turkey" prepared for Inmet Mining Corporation and dated March 30, 2006, do hereby certify that:

1. I am Associate Mining Engineer with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave., Toronto, ON, M5J 2H7.

2. I am a graduate of Academy of Mining and Metallurgy, Krakow, Poland, in 1975 with a Bachelor of Science degree in Mining.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 90247024). I have worked as a mining engineer for a total of 30 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a consultant on a number of operations and projects around the world for due diligence, operations expansions and troubleshooting.
   - Director of Mining with a Major Canadian Mining Corporation
   - Mine Superintendent
   - Senior Project Engineer, Researcher and Production Supervisor

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I visited the Çayeli Mine on December 6-9, 2005.

6. I am responsible for preparation of Item 17 and parts of Items 1, 18, and 19

7. I am independent of the Issuer applying the test set out in Part 1.4 of National Instrument 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

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

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated 30th day of March, 2006.

(Signed and Sealed)
Andrew Hara (Harasimowicz), P. Eng.

23 APPENDIX 1

MINERALIZED ZONE CODES – CROSS SECTIONS

Red:	Zone 1 - Main Zone Clastic Ore (MOCO)
Green:	Zone 2 - Main Zone Spec Ore (MOSpec)
Blue:	Zone 3- Main Zone Stockwork Ore (MOStwk)
Red:	Zone 4 - Deep Ore Zone Clastic Ore (DOCO)
Green:	Zone 5 - Deep Ore Zone Spec Ore (DPSpec)
Blue:	Zone 6 - Deep Ore Zone Stockwork Ore (DOStwk)

FIGURE 23-1 MINERALIZED ZONES – CROSS SECTION 1560N

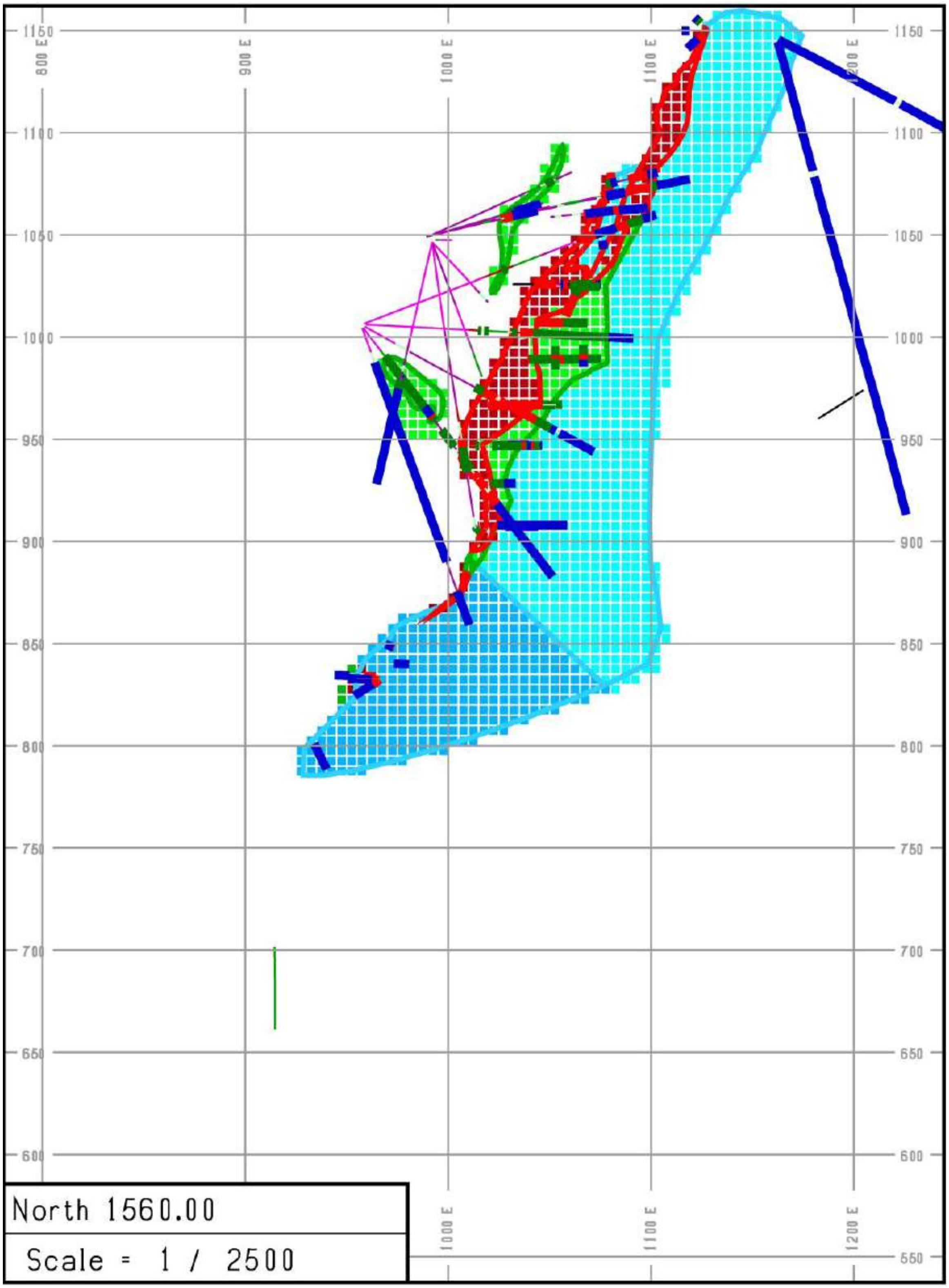


FIGURE 23-2 MINERALIZED ZONES - SECTION 1620N

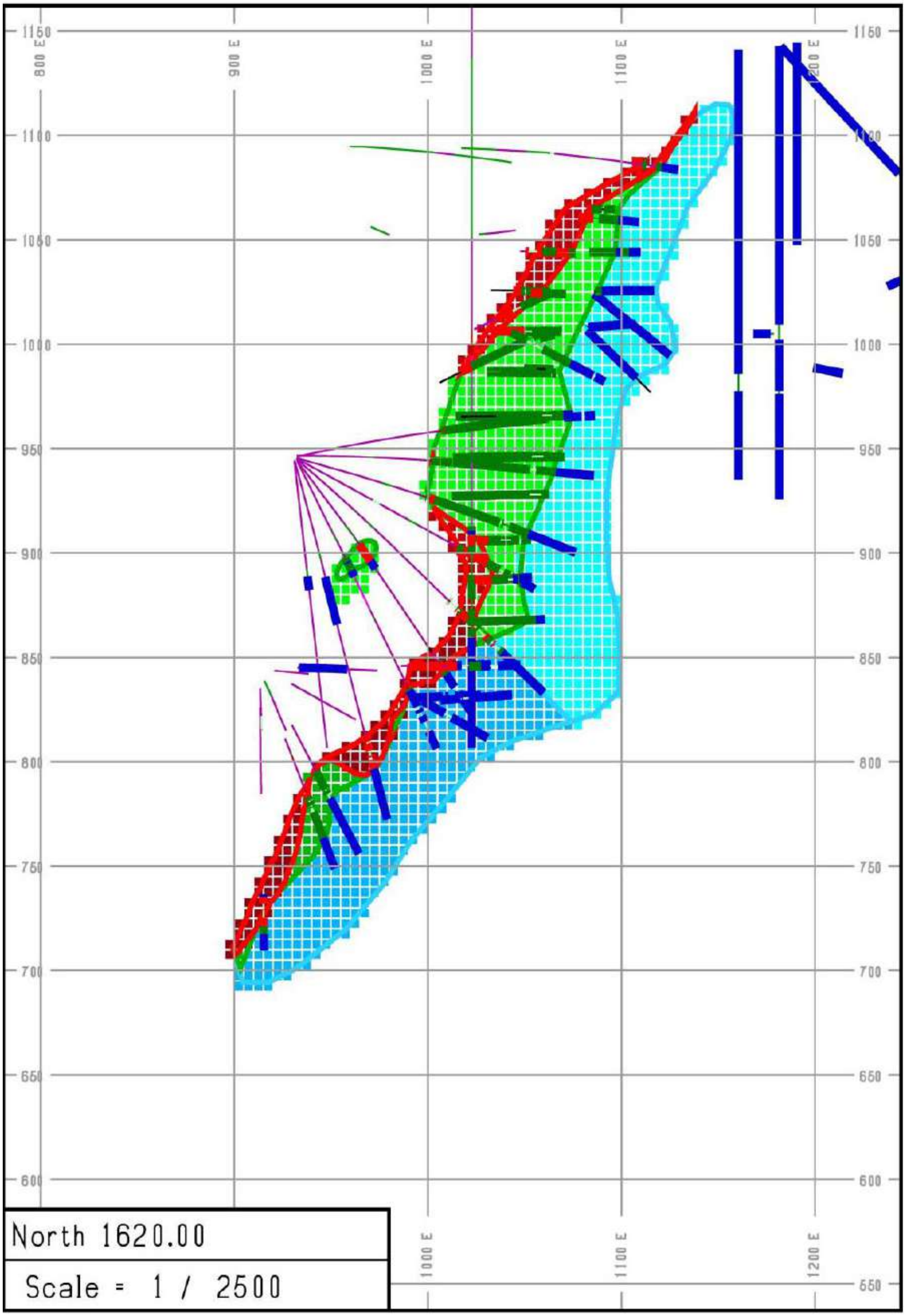


FIGURE 23-3 MINERALIZED ZONES – CROSS SECTION 1680N

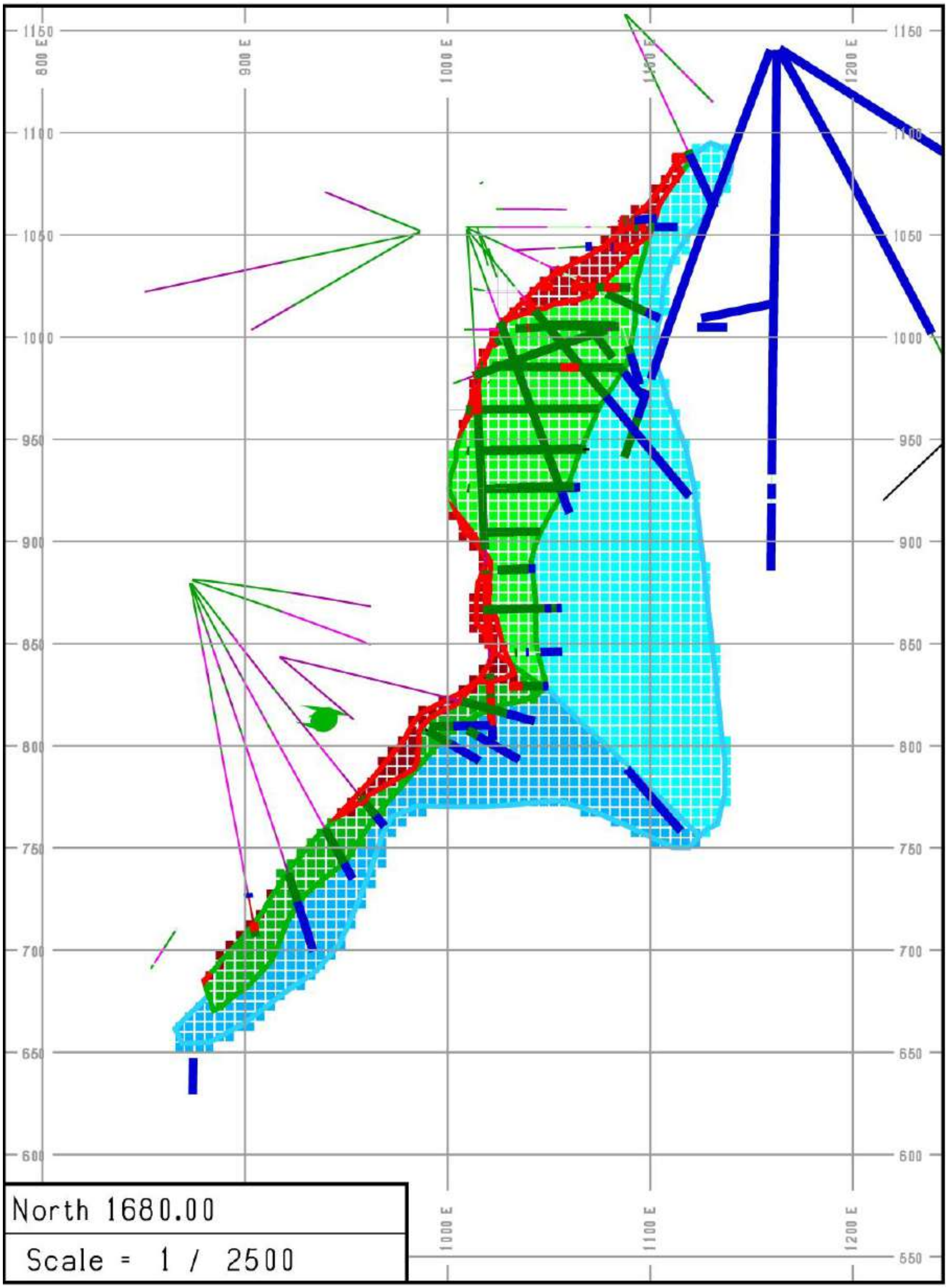


FIGURE 23-4 MINERALIZED ZONES – CROSS SECTION 1740N

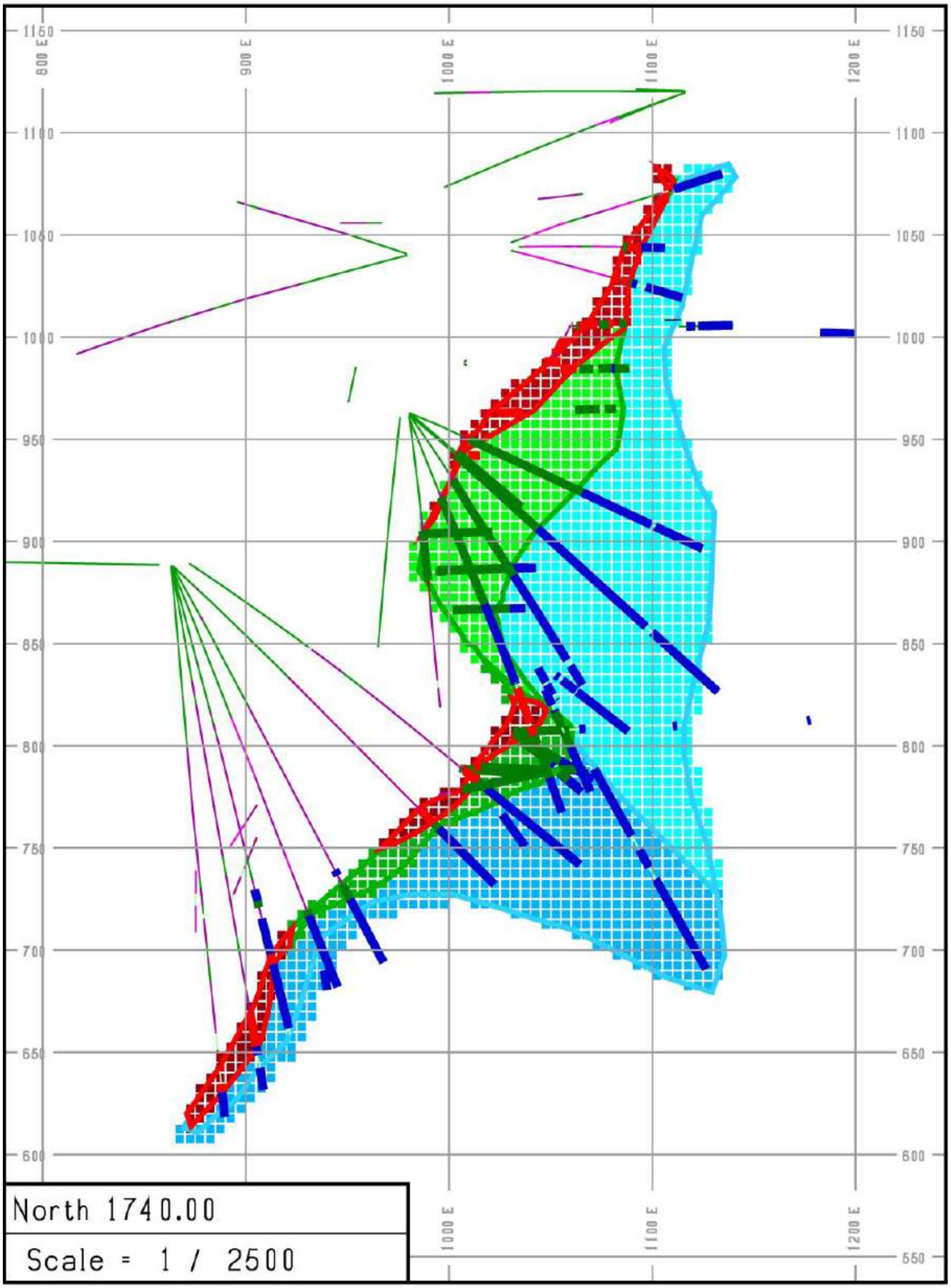


FIGURE 23-5 MINERALIZED ZONES – CROSS SECTION 1800N

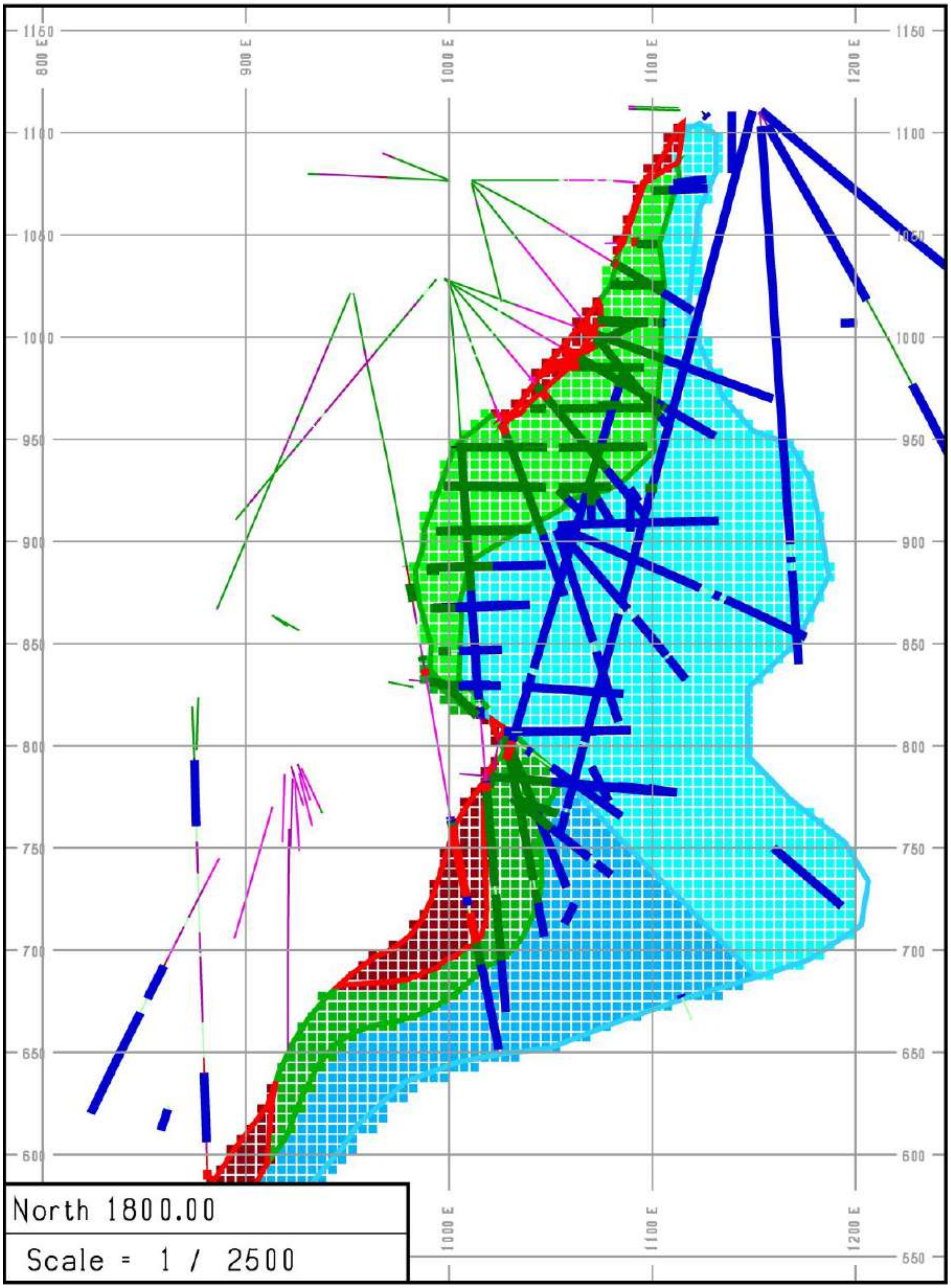


FIGURE 23-6 MINERALIZED ZONES – CROSS SECTION 1860N

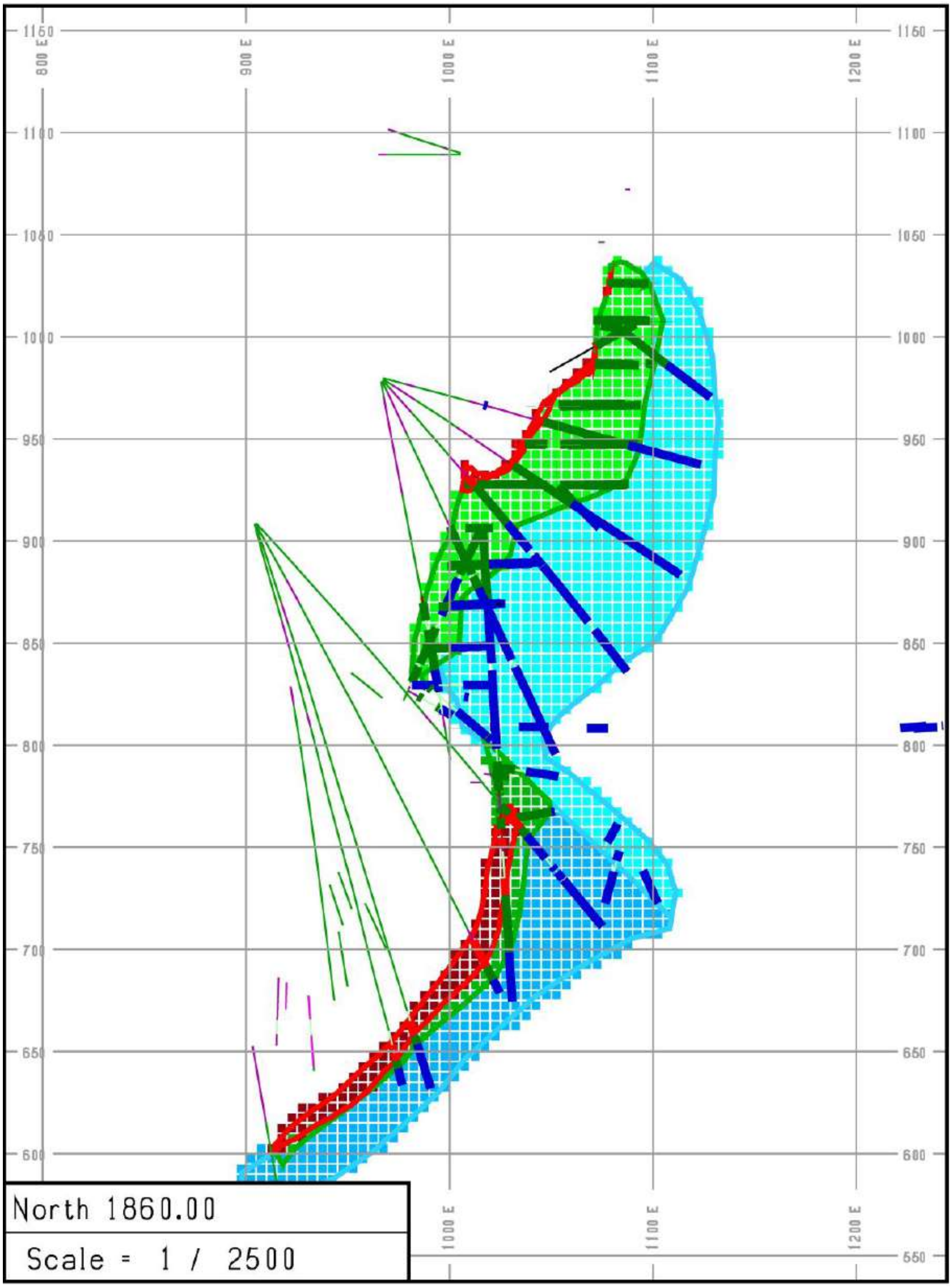


FIGURE 23-7 MINERALIZED ZONES – CROSS SECTION 1920N

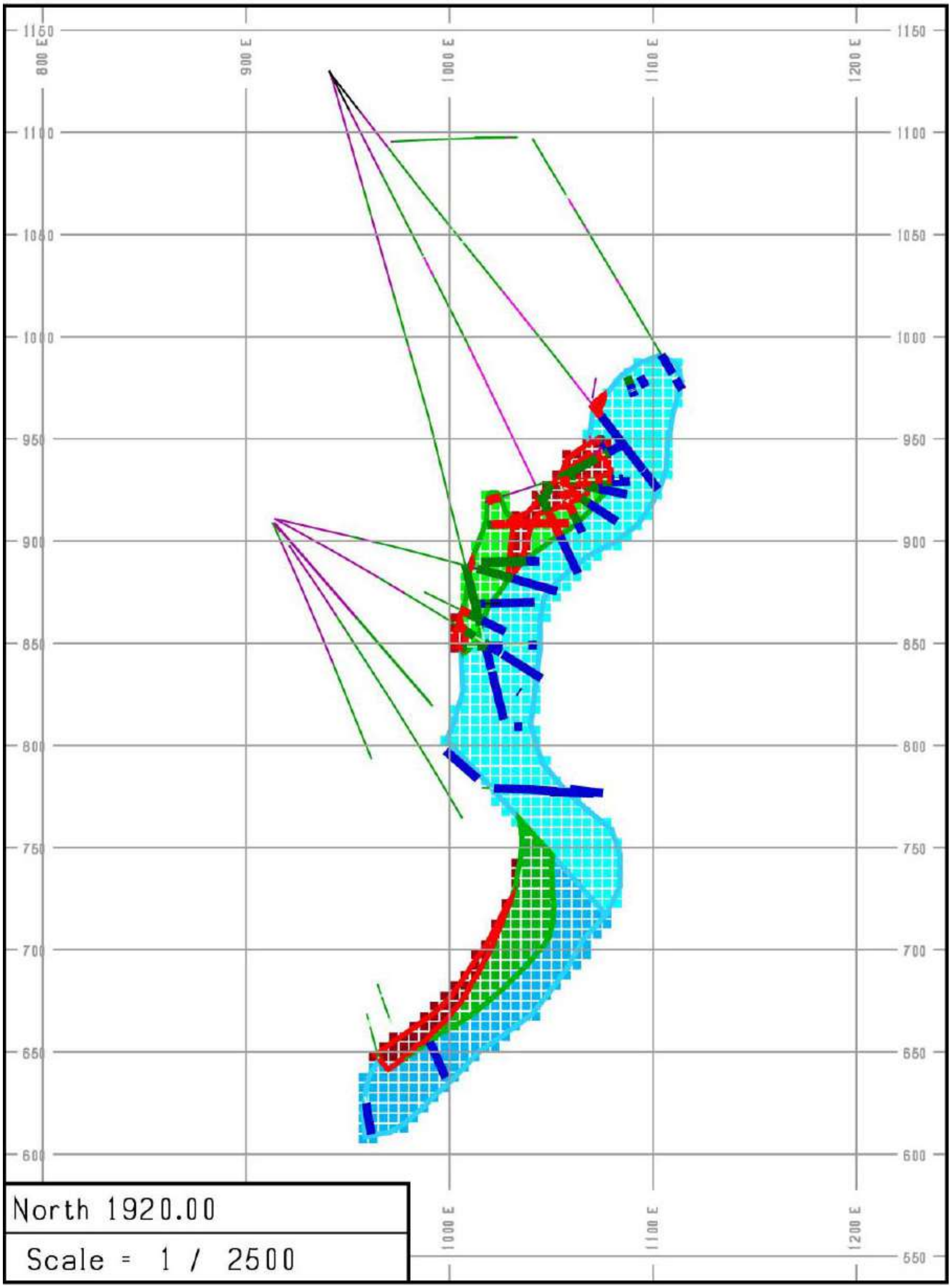


FIGURE 23-8 MINERALIZED ZONES – CROSS SECTION 1980N

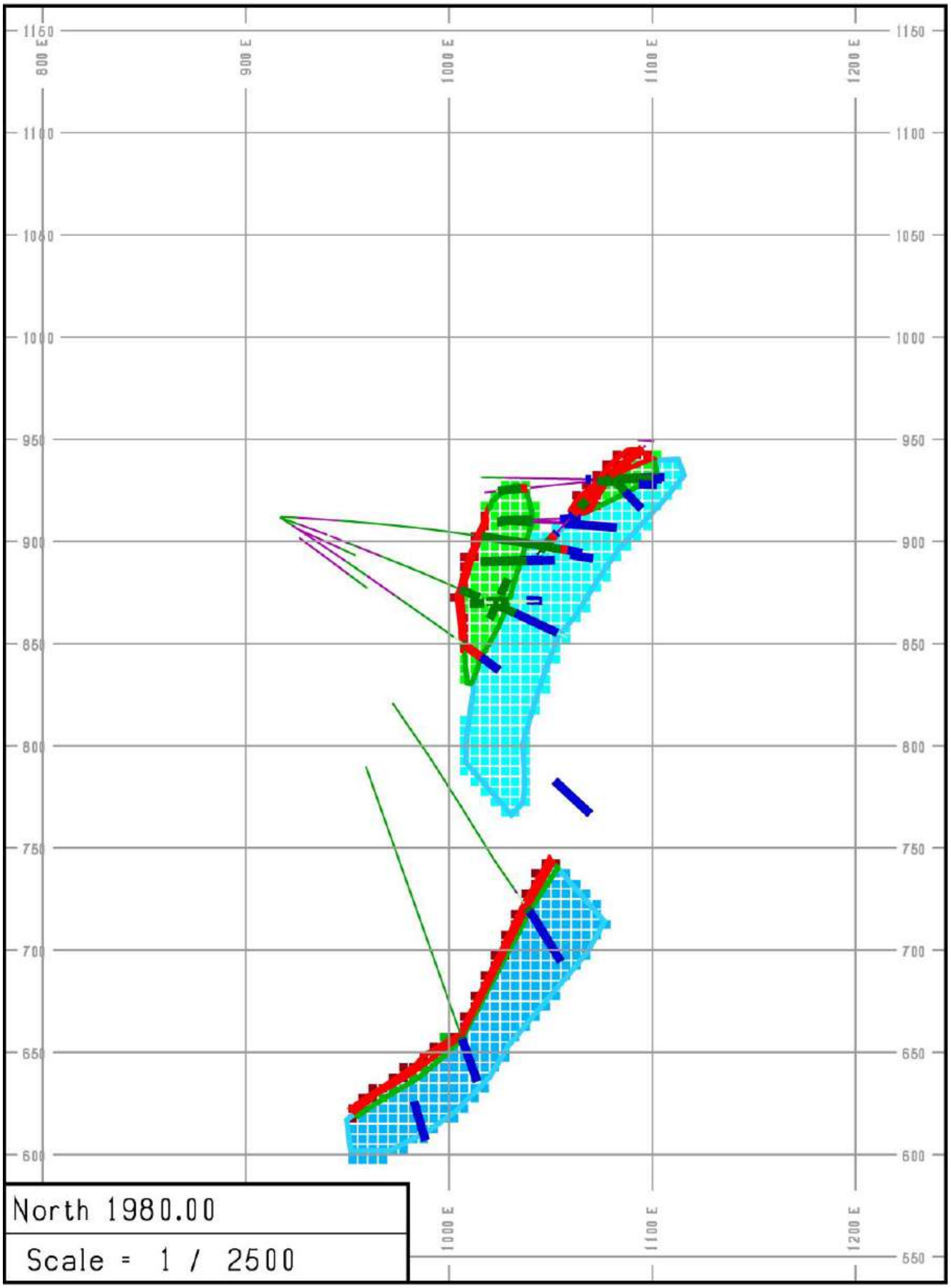


FIGURE 23-9 MINERALIZED ZONES – CROSS SECTION 2040N

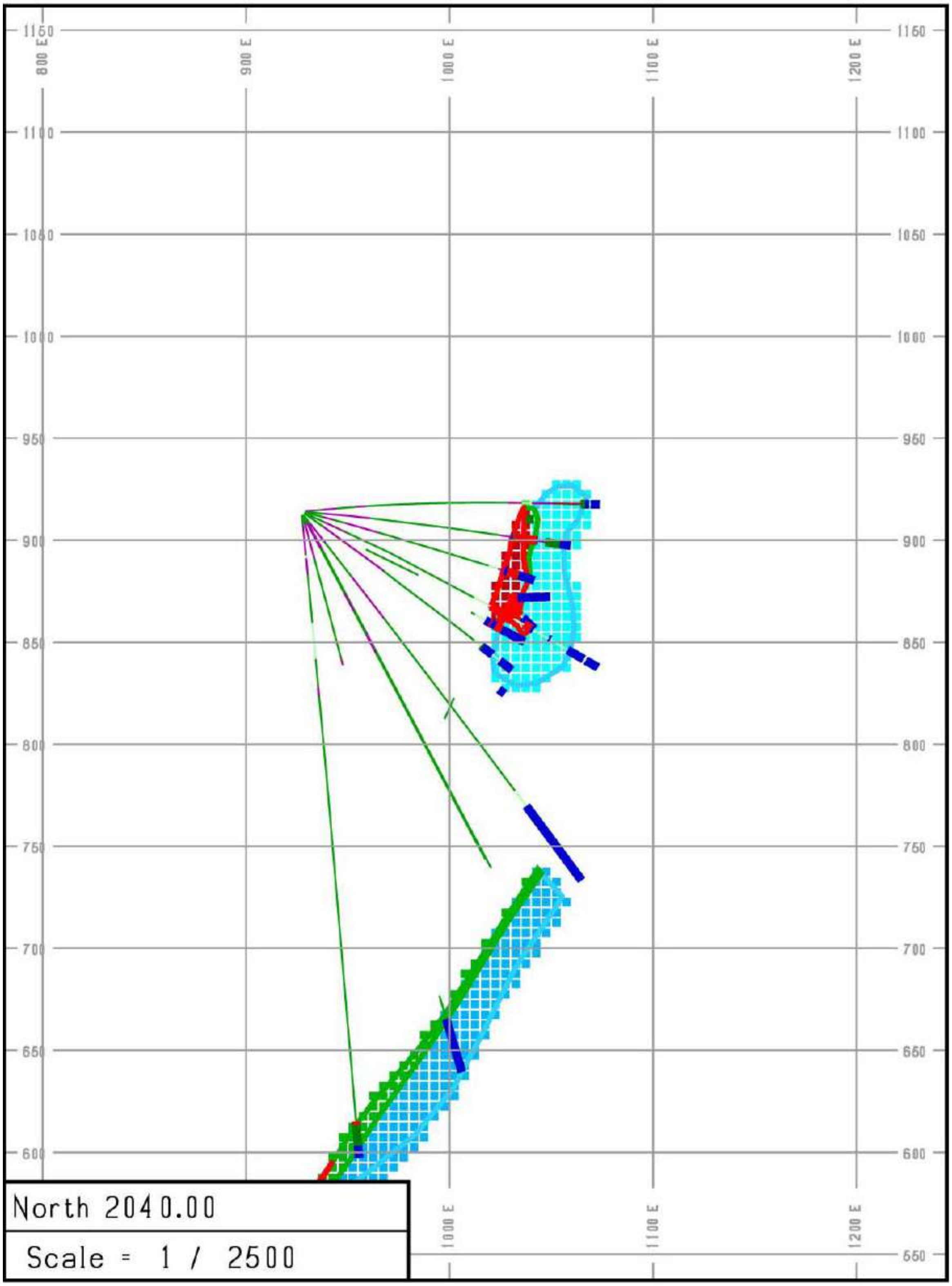
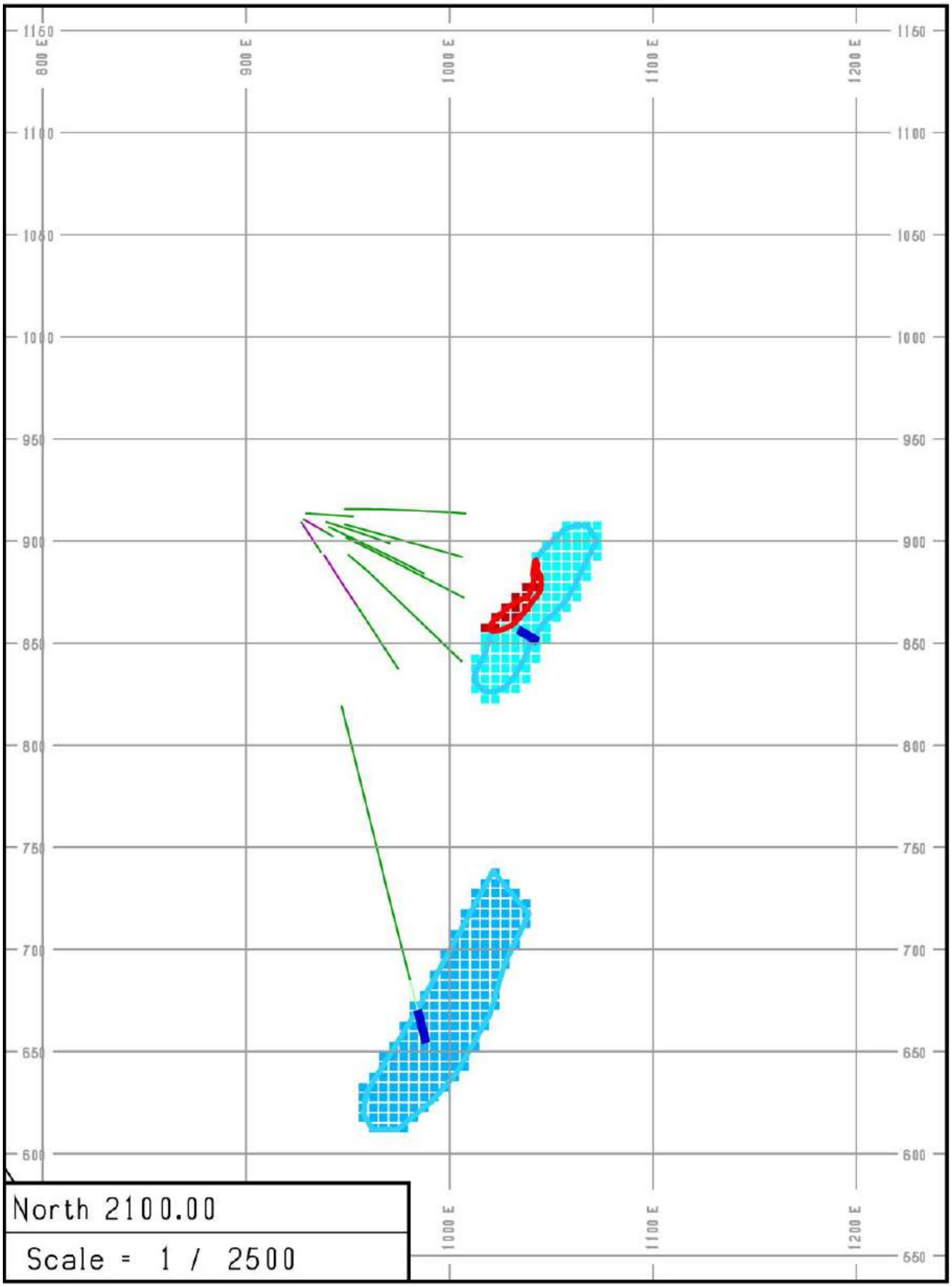


FIGURE 23-10 MINERALIZED ZONES – CROSS SECTION 2100N



24 APPENDIX 2

CORRELOGRAMS – CU & ZN

Zone 1: Main Ore Zone Clastic Ore (MOCO)

Zone 2: Main Ore Zone Spec Ore (MOSpec)

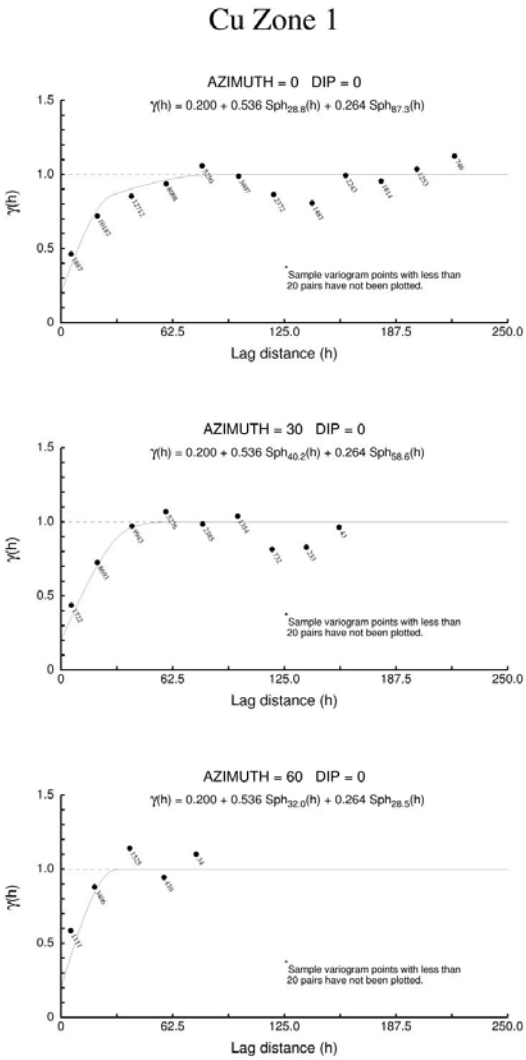
Zone 3: Main Ore Zone Stockwork Ore (MOStwk)

Zone 4: Deep Ore Zone Clastic Ore (DOCO)

Zone 5: Deep Ore Zone Spec Ore (DPSpec)

Zone 6: Deep Ore Zone Stockwork Ore (DOStwk)

FIGURE 24-1 CORRELOGRAM CU – MAIN ZONE CLASTIC ORE (ZONE 1)



Cu Zone 1

Sage95 Rotation Conventions

Nugget ==> 0.200
C1 ==> 0.536
C2 ==> 0.264

First Structure -- Spherical

RH Rotation about the Z axis ==> -35
RH Rotation about the Y' axis ==> 49
RH Rotation about the X' axis ==> -85

Range along the Z' axis ==> 40.3	Azimuth ==> 39	Dip ==> 3
Range along the Y' axis ==> 124.9	Azimuth ==> 312	Dip ==> -41
Range along the X' axis ==> 13.2	Azimuth ==> 125	Dip ==> -49

Second Structure -- Spherical

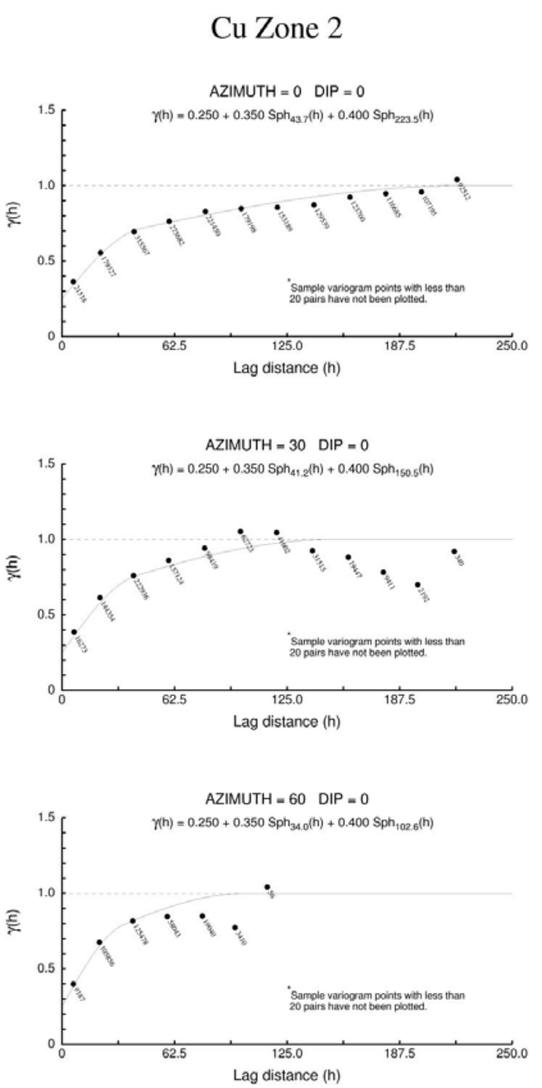
RH Rotation about the Z axis ==> -11
RH Rotation about the Y' axis ==> 33
RH Rotation about the X' axis ==> 35

Range along the Z' axis ==> 207.7	Azimuth ==> 153	Dip ==> 43
Range along the Y' axis ==> 107.6	Azimuth ==> 32	Dip ==> 28
Range along the X' axis ==> 18.4	Azimuth ==> 101	Dip ==> -33

Modeling Criteria

Minimum number pairs req'd ==> 20
Sample variogram points weighted by # pairs

FIGURE 24-2 CORRELOGRAM CU – MAIN ZONE SPEC ORE (ZONE 2)



Cu Zone 2

Sage95 Rotation Conventions

Nugget ==> 0.250
C1 ==> 0.350
C2 ==> 0.400

First Structure -- Spherical

RH Rotation about the Z axis ==> -10
RH Rotation about the Y' axis ==> 7
RH Rotation about the X' axis ==> 25
Range along the Z' axis ==> 67.5 Azimuth ==> 176 Dip ==> 64
Range along the Y' axis ==> 41.7 Azimuth ==> 13 Dip ==> 25
Range along the X' axis ==> 30.2 Azimuth ==> 100 Dip ==> -7

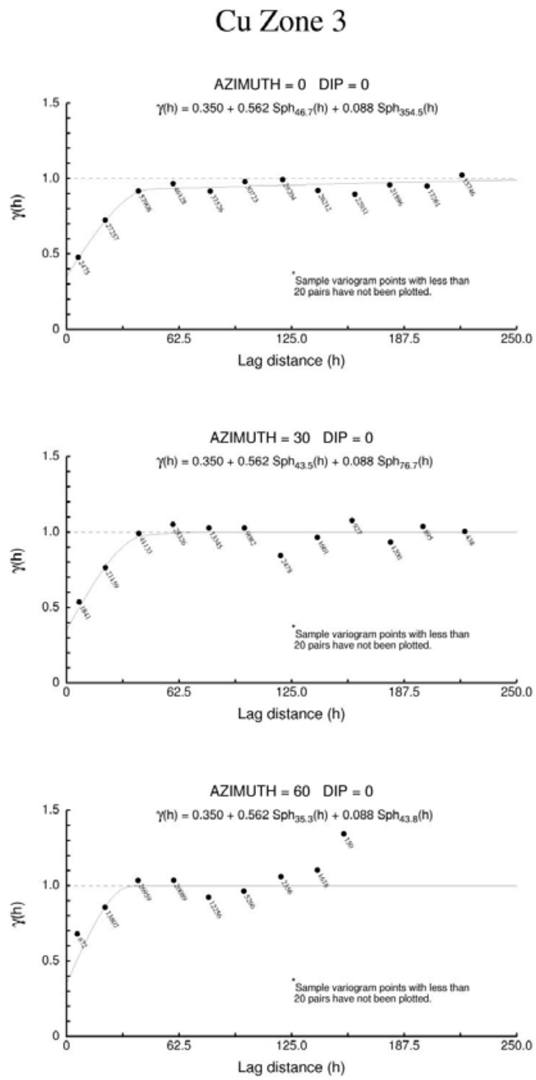
Second Structure -- Spherical

RH Rotation about the Z axis ==> -10
RH Rotation about the Y' axis ==> -35
RH Rotation about the X' axis ==> 12
Range along the Z' axis ==> 80.1 Azimuth ==> 260 Dip ==> 54
Range along the Y' axis ==> 243.5 Azimuth ==> 3 Dip ==> 9
Range along the X' axis ==> 97.8 Azimuth ==> 100 Dip ==> 35

Modeling Criteria

Minimum number pairs req'd ==> 20
Sample variogram points weighted by # pairs

FIGURE 24-3 CORRELOGRAM CU – MAIN ZONE STOCKWORK ORE (ZONE 3)



Cu Zone 3

Sage95 Rotation Conventions

Nugget ==> 0.350
C1 ==> 0.562
C2 ==> 0.088

First Structure -- Spherical

RH Rotation about the Z axis ==> -21
RH Rotation about the Y' axis ==> 73
RH Rotation about the X' axis ==> -20
Range along the Z' axis ==> 34.6 Azimuth ==> 90 Dip ==> 16
Range along the Y' axis ==> 48.3 Azimuth ==> 1 Dip ==> -6
Range along the X' axis ==> 18.0 Azimuth ==> 111 Dip ==> -73

Second Structure -- Spherical

RH Rotation about the Z axis ==> -1
RH Rotation about the Y' axis ==> -6
RH Rotation about the X' axis ==> 107
Range along the Z' axis ==> 455.9 Azimuth ==> 180 Dip ==> -16
Range along the Y' axis ==> 159.4 Azimuth ==> 201 Dip ==> 72
Range along the X' axis ==> 37.3 Azimuth ==> 91 Dip ==> 6

Modeling Criteria

Minimum number pairs req'd ==> 20
Sample variogram points weighted by # pairs

FIGURE 24-4 CORRELOGRAM CU – DEEP ZONE CLASTIC ORE (ZONE 4)

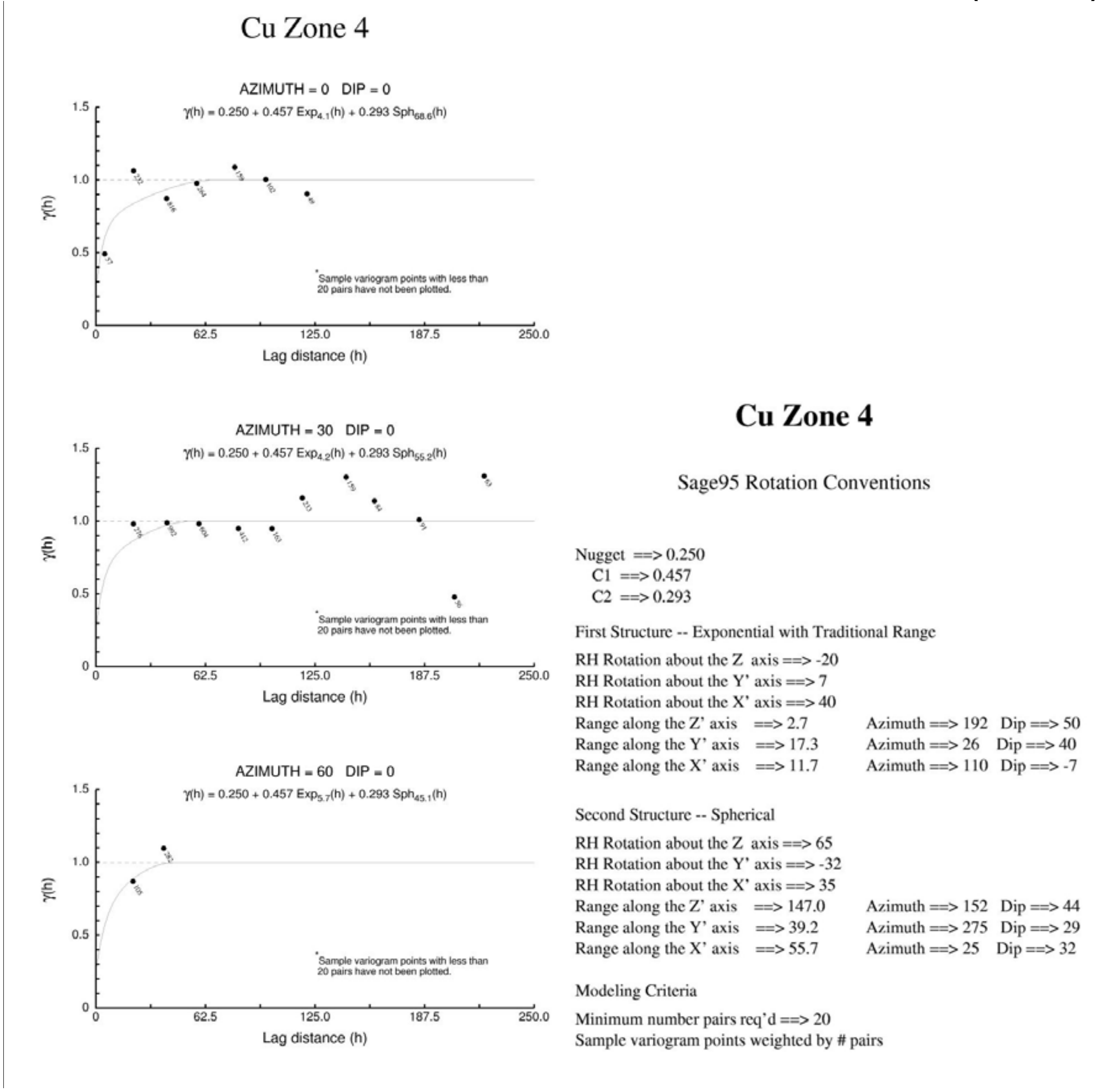
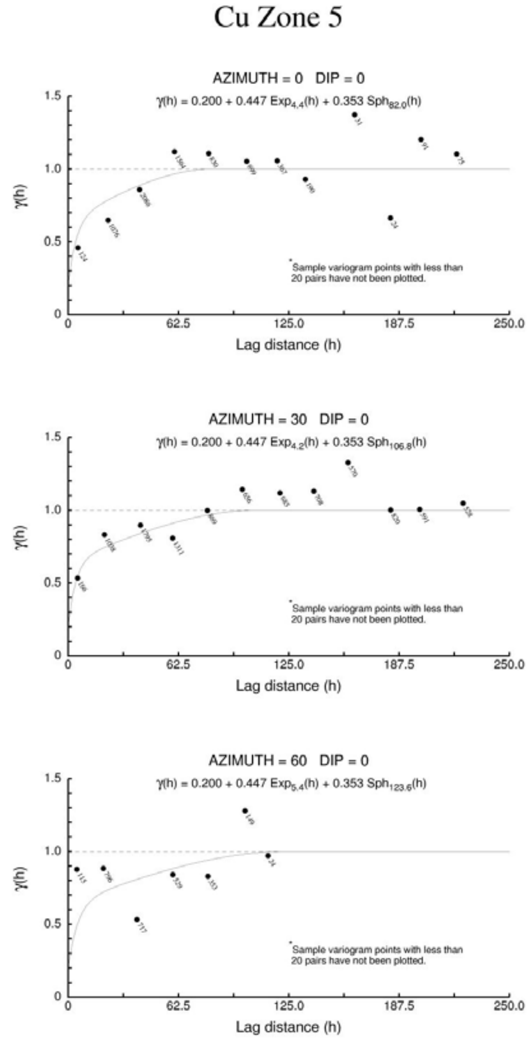


FIGURE 24-5 CORRELOGRAM CU – DEEP ZONE SPEC ORE (ZONE 5)



Cu Zone 5

Sage95 Rotation Conventions

Nugget ==> 0.200
C1 ==> 0.447
C2 ==> 0.353

First Structure -- Exponential with Traditional Range

RH Rotation about the Z axis ==> -15
RH Rotation about the Y' axis ==> 42
RH Rotation about the X' axis ==> 6
Range along the Z' axis ==> 13.5 Azimuth ==> 114 Dip ==> 48
Range along the Y' axis ==> 4.1 Azimuth ==> 19 Dip ==> 5
Range along the X' axis ==> 32.8 Azimuth ==> 105 Dip ==> -42

Second Structure -- Spherical

RH Rotation about the Z axis ==> 105
RH Rotation about the Y' axis ==> -68
RH Rotation about the X' axis ==> 51
Range along the Z' axis ==> 94.6 Azimuth ==> 112 Dip ==> 14
Range along the Y' axis ==> 193.0 Azimuth ==> 206 Dip ==> 17
Range along the X' axis ==> 33.6 Azimuth ==> 345 Dip ==> 68

Modeling Criteria

Minimum number pairs req'd ==> 20
Sample variogram points weighted by # pairs

FIGURE 24-6 CORRELOGRAM CU – DEEP ZONE STOCKWORK ORE (ZONE 6)

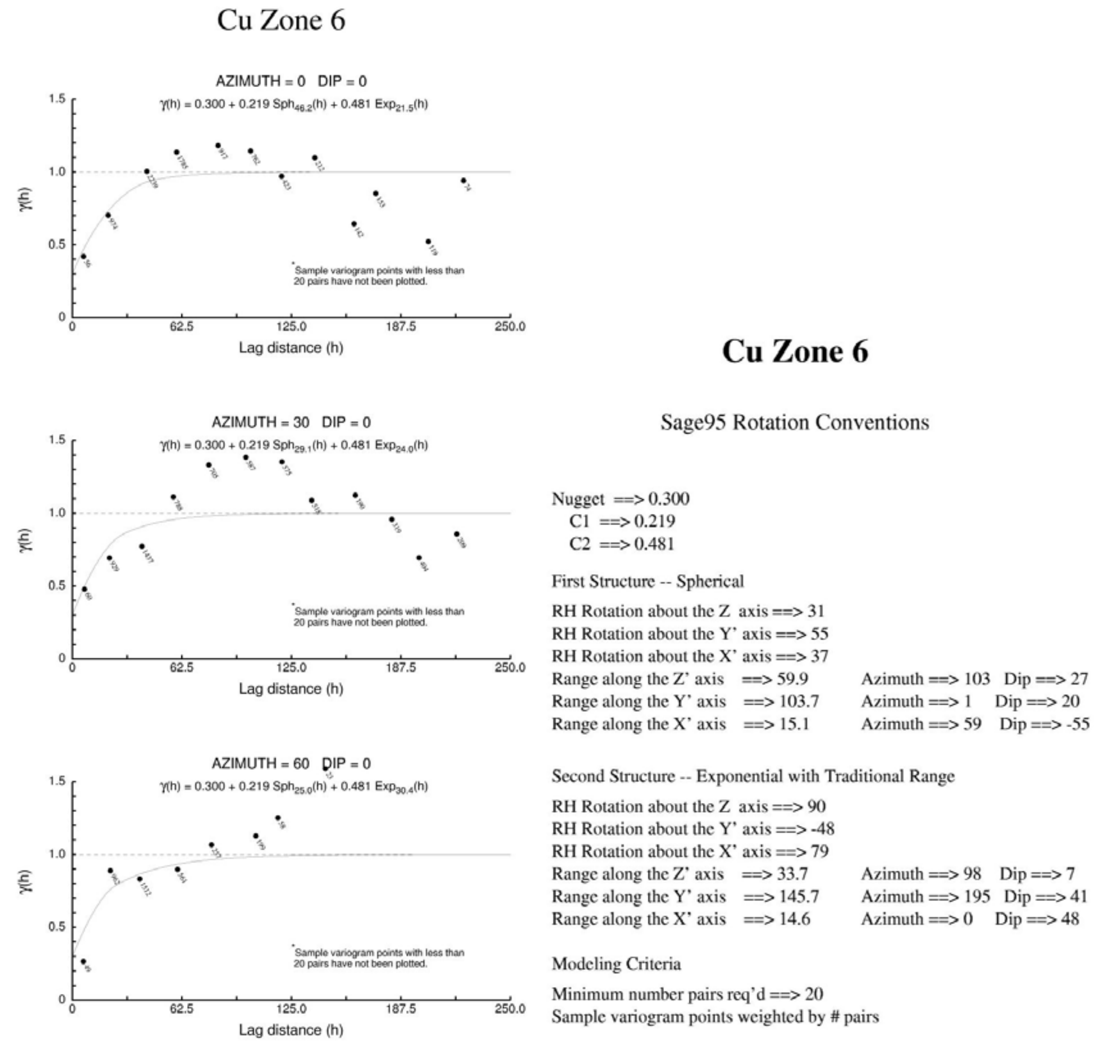


FIGURE 24-7 CORRELOGRAM ZN – MAIN ZONE CLASTIC ORE (ZONE 1)

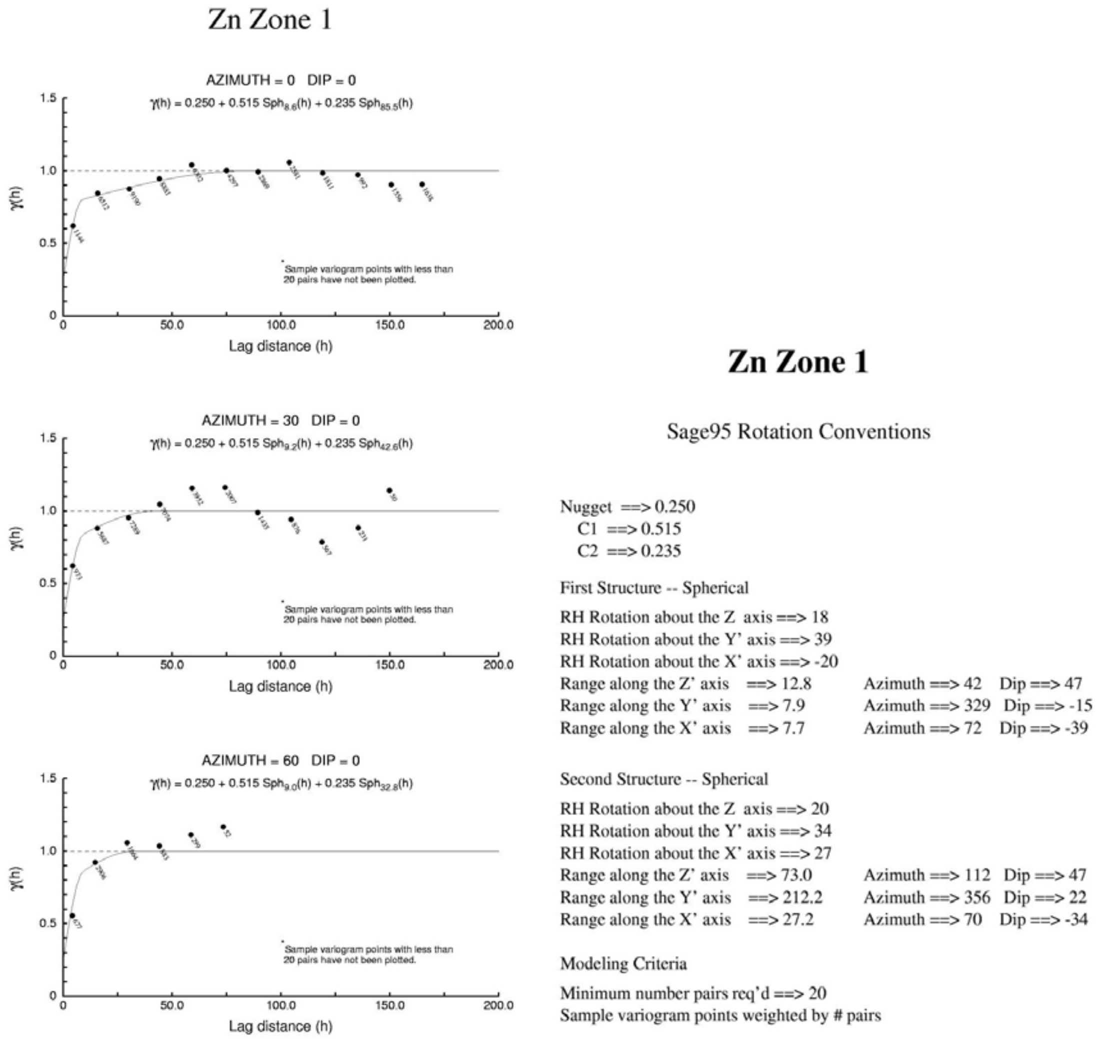


FIGURE 24-8 CORRELOGRAM ZN – MAIN ZONE SPEC ORE (ZONE 2)

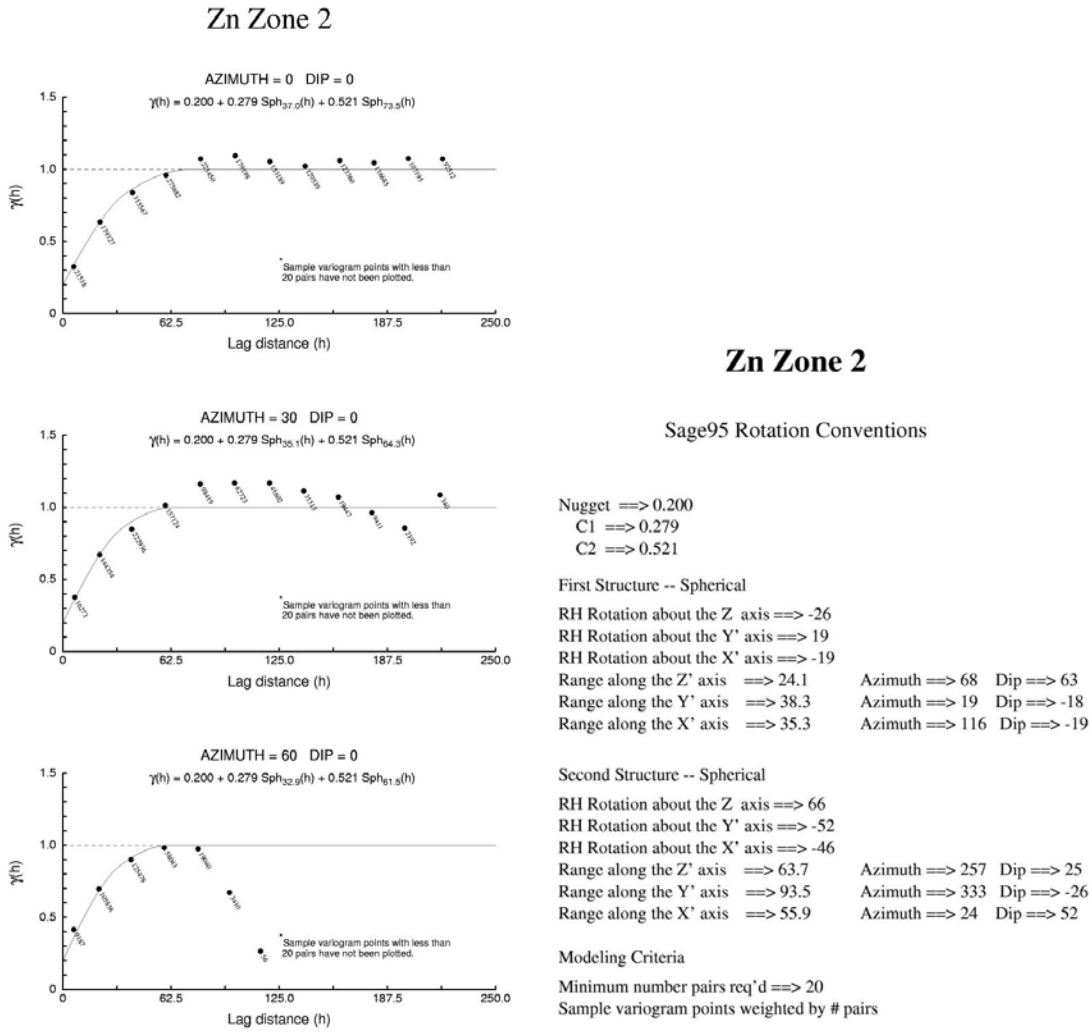
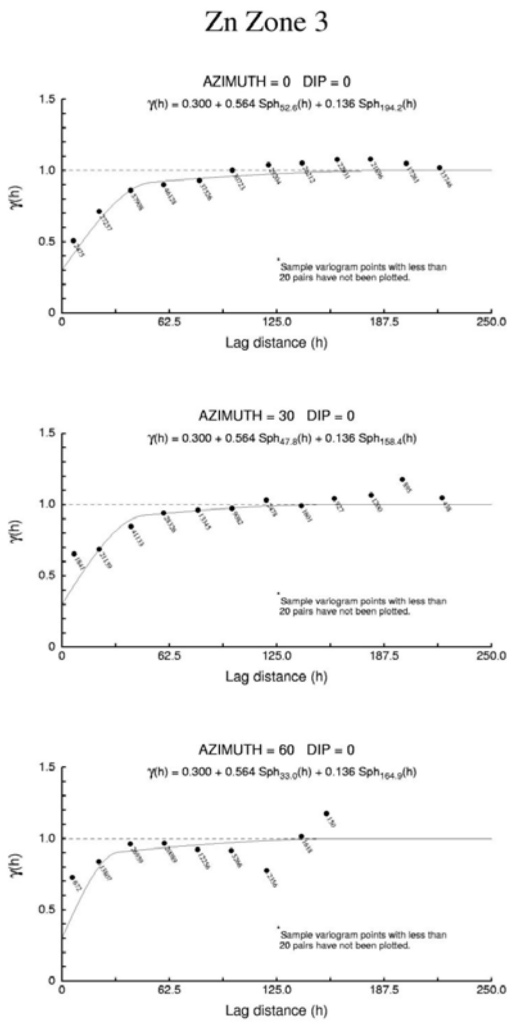


FIGURE 24-9 CORRELOGRAM ZN – MAIN ZONE STOCKWORK ORE (ZONE 3)



Zn Zone 3

Sage95 Rotation Conventions

Nugget ==> 0.300
C1 ==> 0.564
C2 ==> 0.136

First Structure -- Spherical

RH Rotation about the Z axis ==> -4
RH Rotation about the Y' axis ==> 60
RH Rotation about the X' axis ==> 10
Range along the Z' axis ==> 29.6 Azimuth ==> 106 Dip ==> 30
Range along the Y' axis ==> 56.4 Azimuth ==> 13 Dip ==> 5
Range along the X' axis ==> 22.4 Azimuth ==> 94 Dip ==> -60

Second Structure -- Spherical

RH Rotation about the Z axis ==> 41
RH Rotation about the Y' axis ==> 34
RH Rotation about the X' axis ==> -18
Range along the Z' axis ==> 164.4 Azimuth ==> 19 Dip ==> 52
Range along the Y' axis ==> 522.4 Azimuth ==> 309 Dip ==> -15
Range along the X' axis ==> 152.8 Azimuth ==> 49 Dip ==> -34

Modeling Criteria

Minimum number pairs req'd ==> 20
Sample variogram points weighted by # pairs

FIGURE 24-10 CORRELOGRAM ZN – DEEP ZONE CLASTIC ORE (ZONE 4)

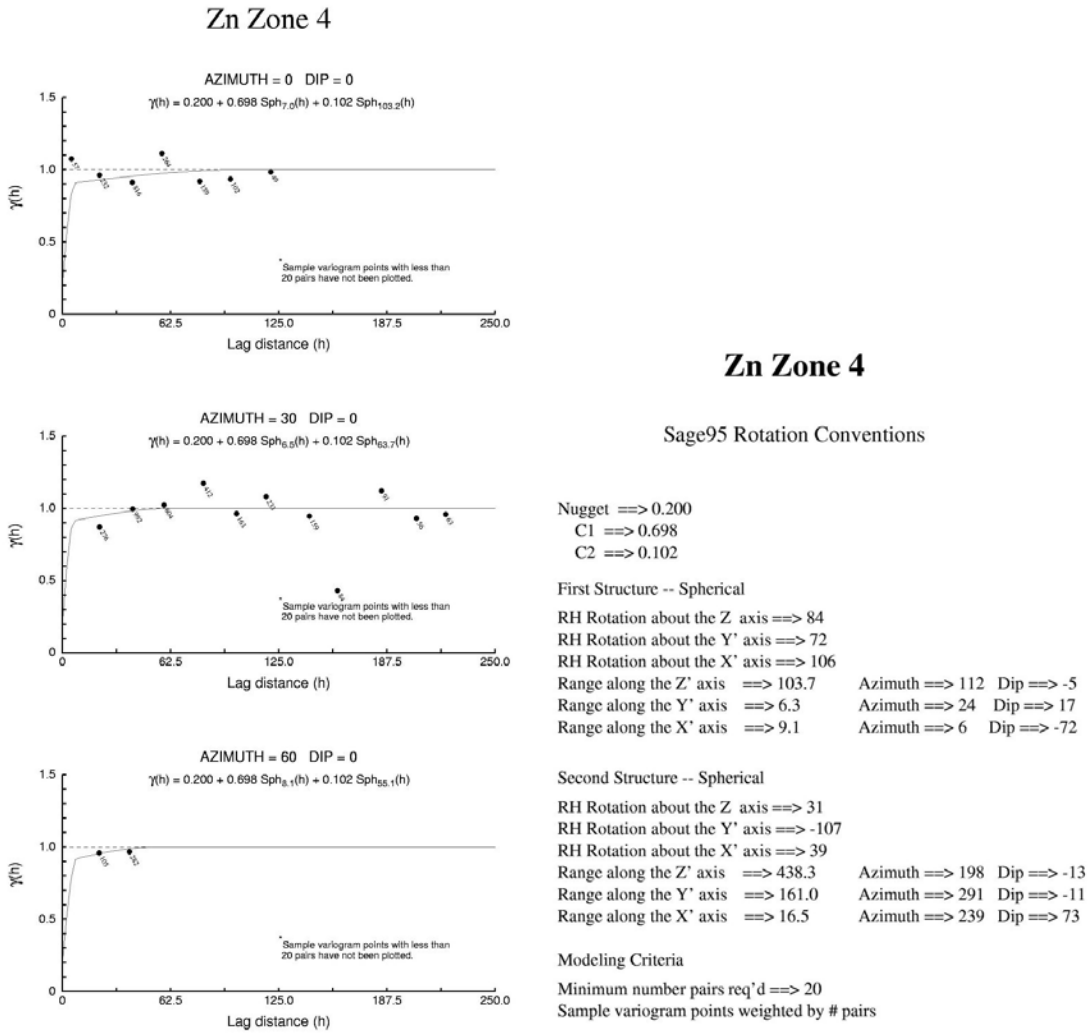


FIGURE 24-11 CORRELOGRAM ZN – DEEP ZONE SPEC ORE (ZONE 5)

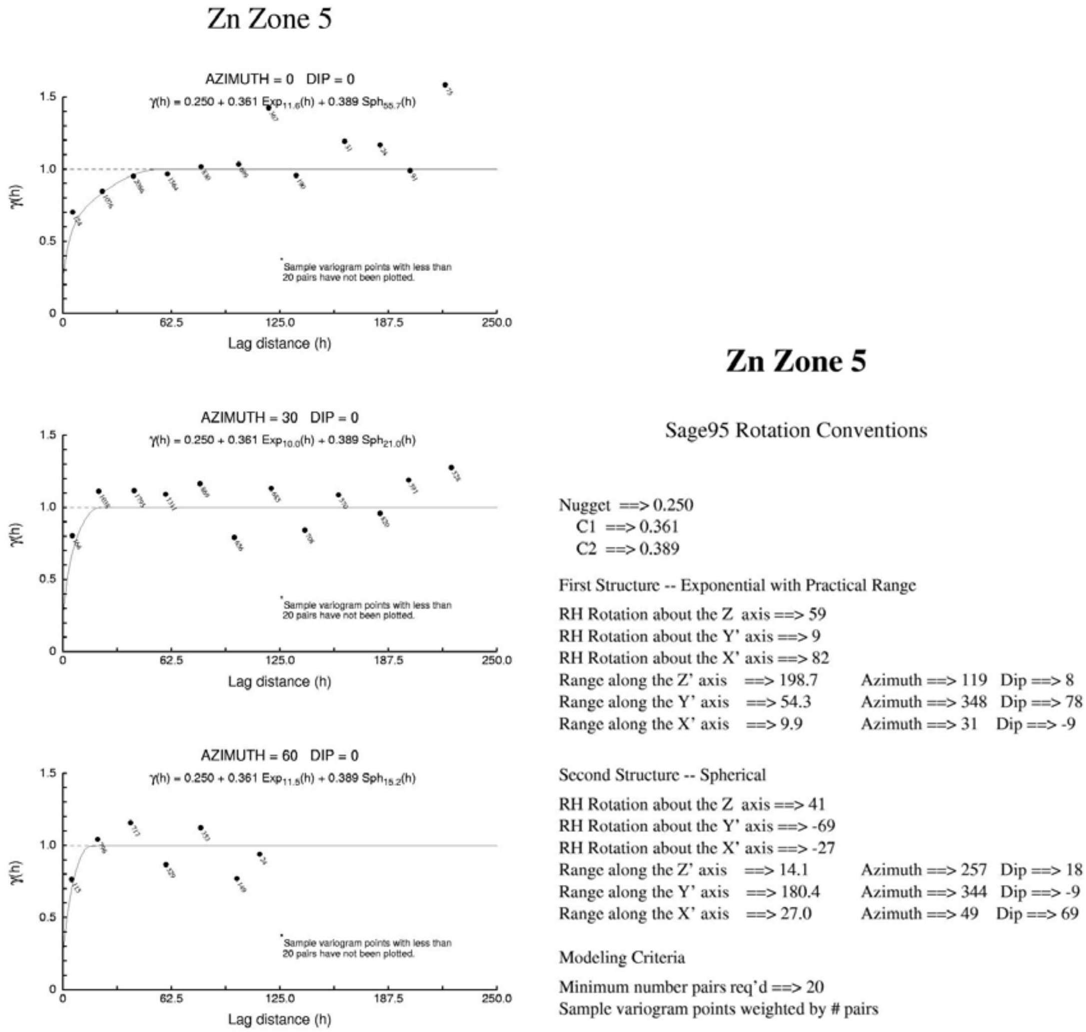
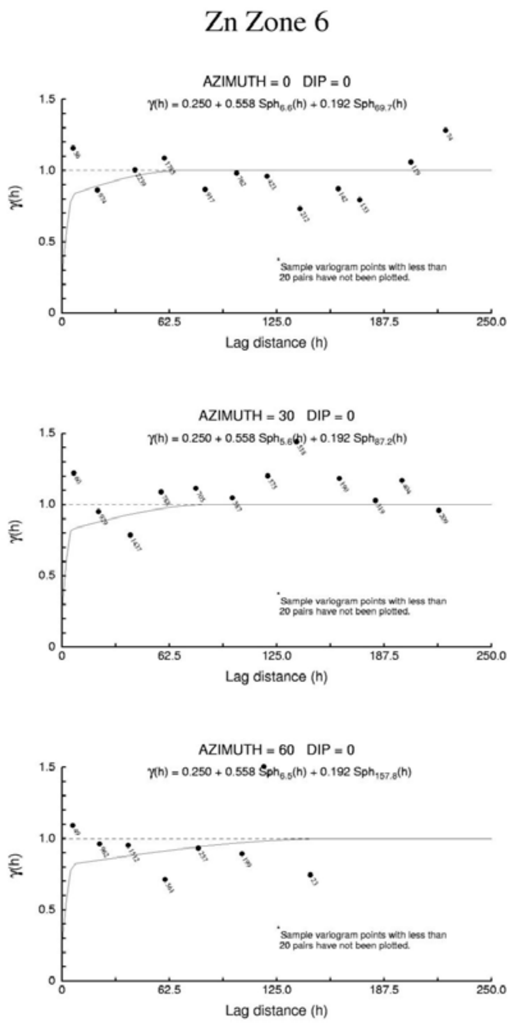


FIGURE 24-12 CORRELOGRAM ZN – DEEP ZONE STOCKWORK ORE (ZONE 6)



Zn Zone 6

Sage95 Rotation Conventions

Nugget ==> 0.250
C1 ==> 0.558
C2 ==> 0.192

First Structure -- Spherical

RH Rotation about the Z axis ==> 18
RH Rotation about the Y' axis ==> 79
RH Rotation about the X' axis ==> -41

Range along the Z' axis ==> 5.6	Azimuth ==> 31	Dip ==> 8
Range along the Y' axis ==> 97.2	Azimuth ==> 302	Dip ==> -7
Range along the X' axis ==> 14.3	Azimuth ==> 72	Dip ==> -79

Second Structure -- Spherical

RH Rotation about the Z axis ==> -78
RH Rotation about the Y' axis ==> 33
RH Rotation about the X' axis ==> 50

Range along the Z' axis ==> 194.9	Azimuth ==> 234	Dip ==> 33
Range along the Y' axis ==> 443.8	Azimuth ==> 111	Dip ==> 40
Range along the X' axis ==> 58.3	Azimuth ==> 168	Dip ==> -33

Modeling Criteria

Minimum number pairs req'd ==> 20
Sample variogram points weighted by # pairs

25 APPENDIX 3

PROBABILITY PLOTS – CU & ZN

Zone 1: Main Ore Zone Clastic Ore (MOCO)

Zone 2: Main Ore Zone Spec Ore (MOSpec)

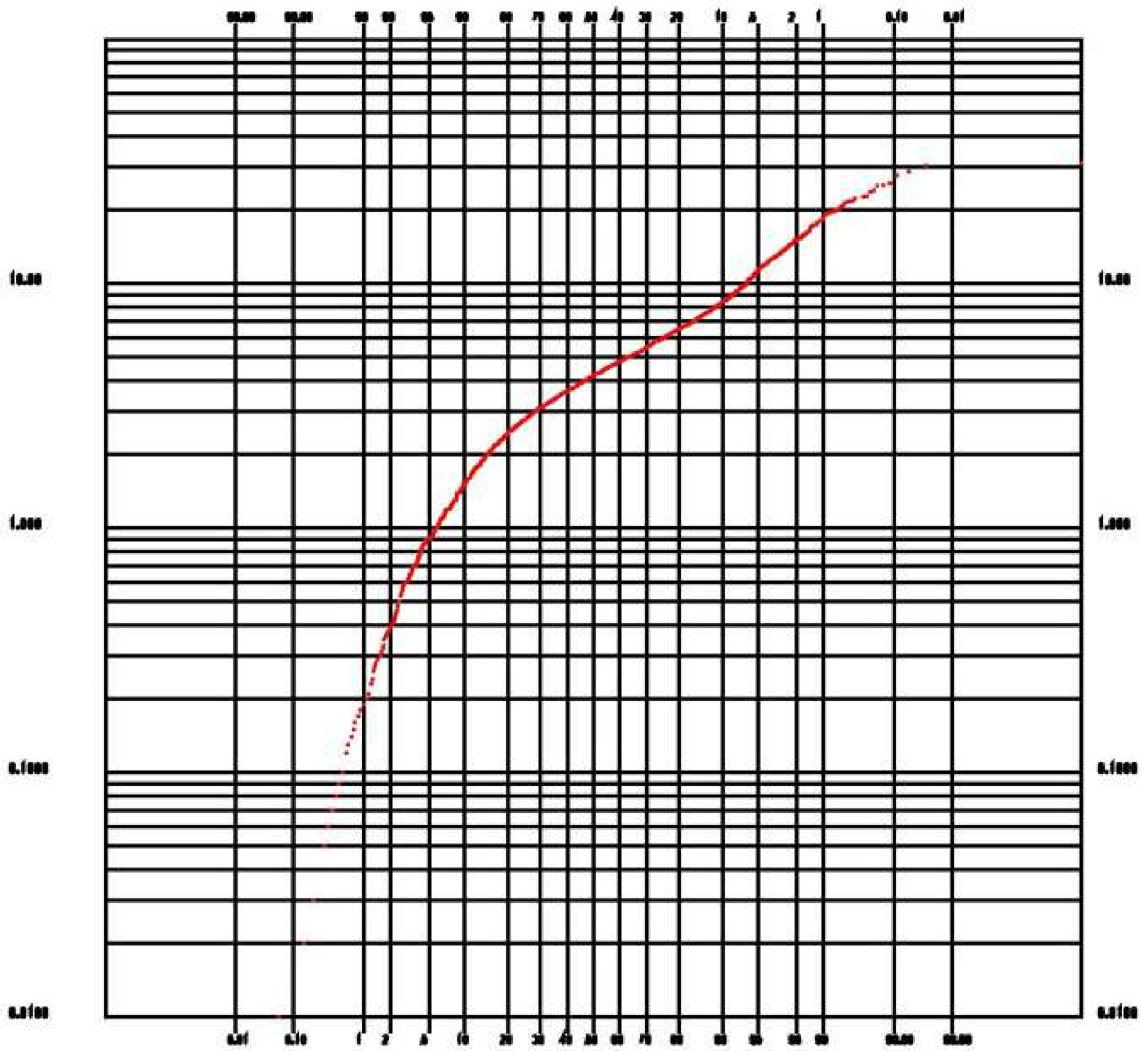
Zone 3: Main Ore Zone Stockwork Ore (MOStwk)

Zone 4: Deep Ore Zone Clastic Ore (DOCO)

Zone 5: Deep Ore Zone Spec Ore (DPSpec)

Zone 6: Deep Ore Zone Stockwork Ore (DOStwk)

FIGURE 25-1 PROBABILITY PLOT CU – MAIN ZONE CLASTIC ORE (ZONE 1)

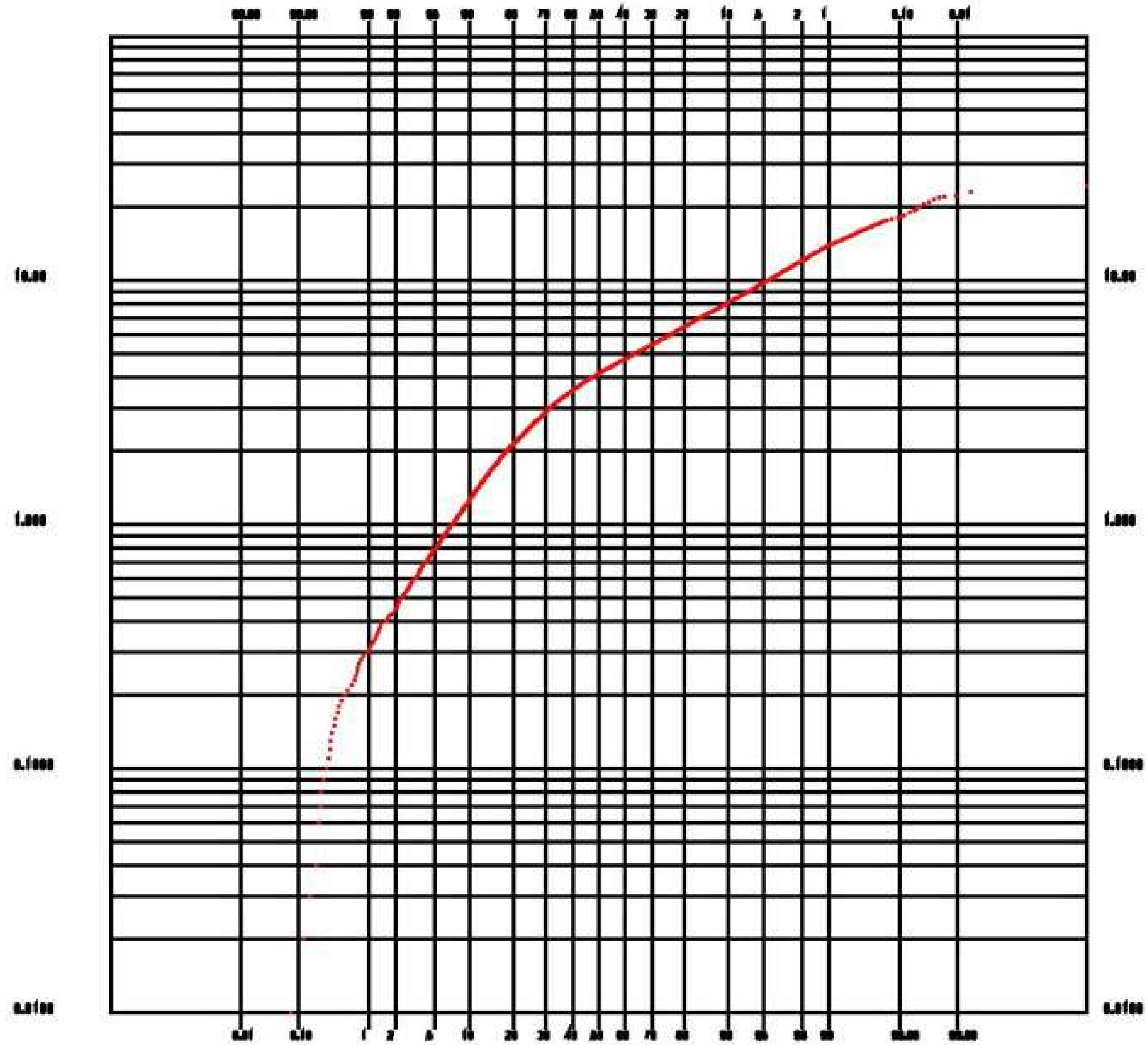


** PROBABILITY DISTRIBUTION PLOT OF CU **

ITEM	CU	NATURAL LOGS	
NUMBER	3391	NUMBER	3391
MEAN	4.8620	MEAN	1.3300
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	31.1600	MAXIMUM	3.4390
VARIANCE	11.7010	VARIANCE	0.6590
ST.DEV.	3.4210	ST.DEV.	0.8120

Cu Zone 1

FIGURE 25-2 PROBABILITY PLOT CU – MAIN ZONE SPEC ORE (ZONE 2)

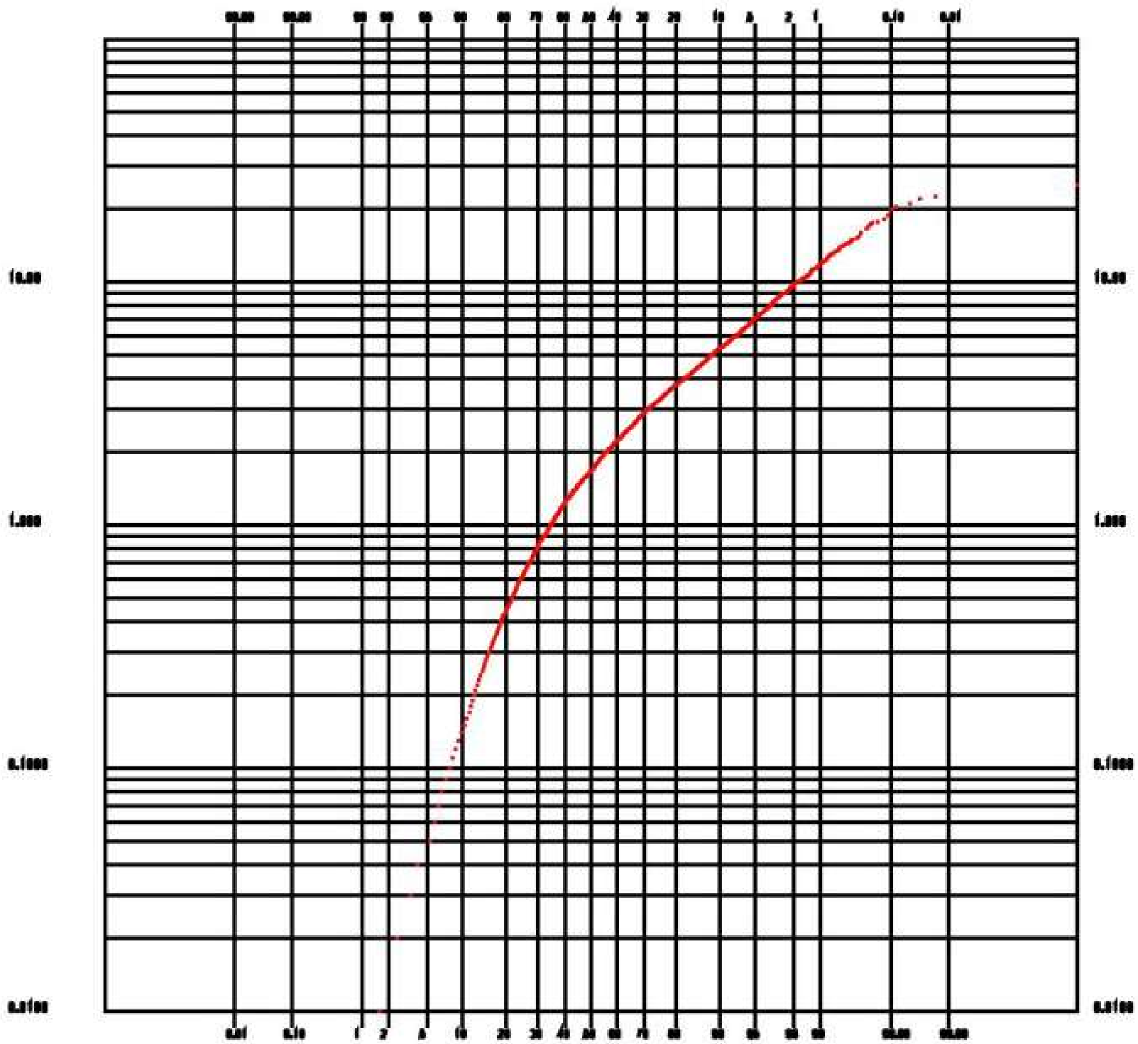


** PROBABILITY DISTRIBUTION PLOT OF CU **

ITEM	CU	NATURAL LOGS	
NUMBER	18735	NUMBER	18735
MEAN	4.6450	MEAN	1.2780
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	24.4000	MAXIMUM	3.1950
VARIANCE	8.0410	VARIANCE	0.6230
ST.DEV.	2.8360	ST.DEV.	0.7890

Cu Zone 2

FIGURE 25-3 PROBABILITY PLOT CU – MAIN ZONE STOCKWORK ORE (ZONE 3)

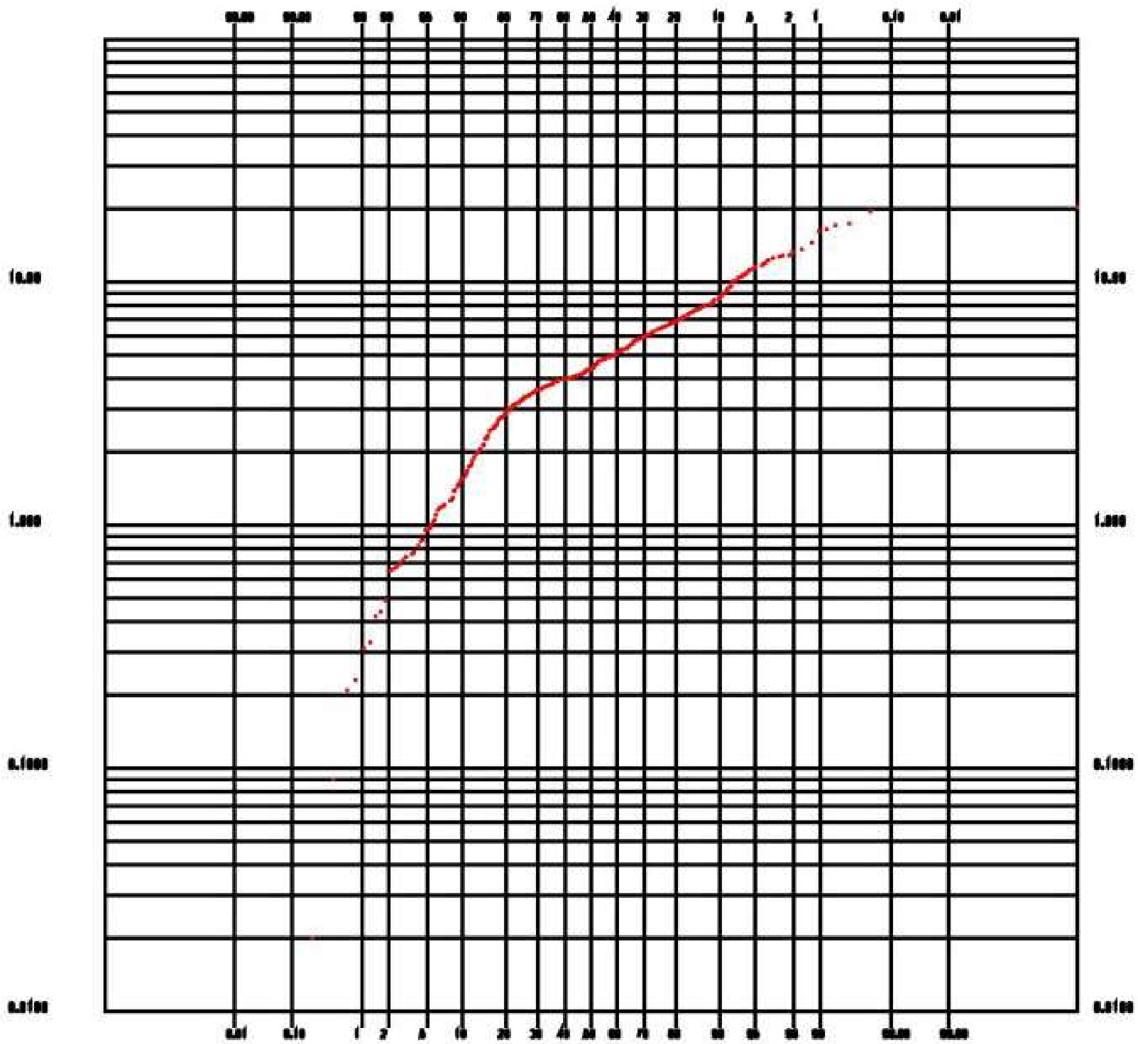


** PROBABILITY DISTRIBUTION PLOT OF CU **

ITEM	CU	NATURAL LOGS	
NUMBER	6282	NUMBER	6282
MEAN	2.3850	MEAN	0.1880
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	25.0100	MAXIMUM	3.2190
VARIANCE	6.2630	VARIANCE	2.1340
ST.DEV.	2.5030	ST.DEV.	1.4610

Cu Zone 3

FIGURE 25-4 PROBABILITY PLOT CU – DEEP ZONE CLASTIC ORE (ZONE 4)

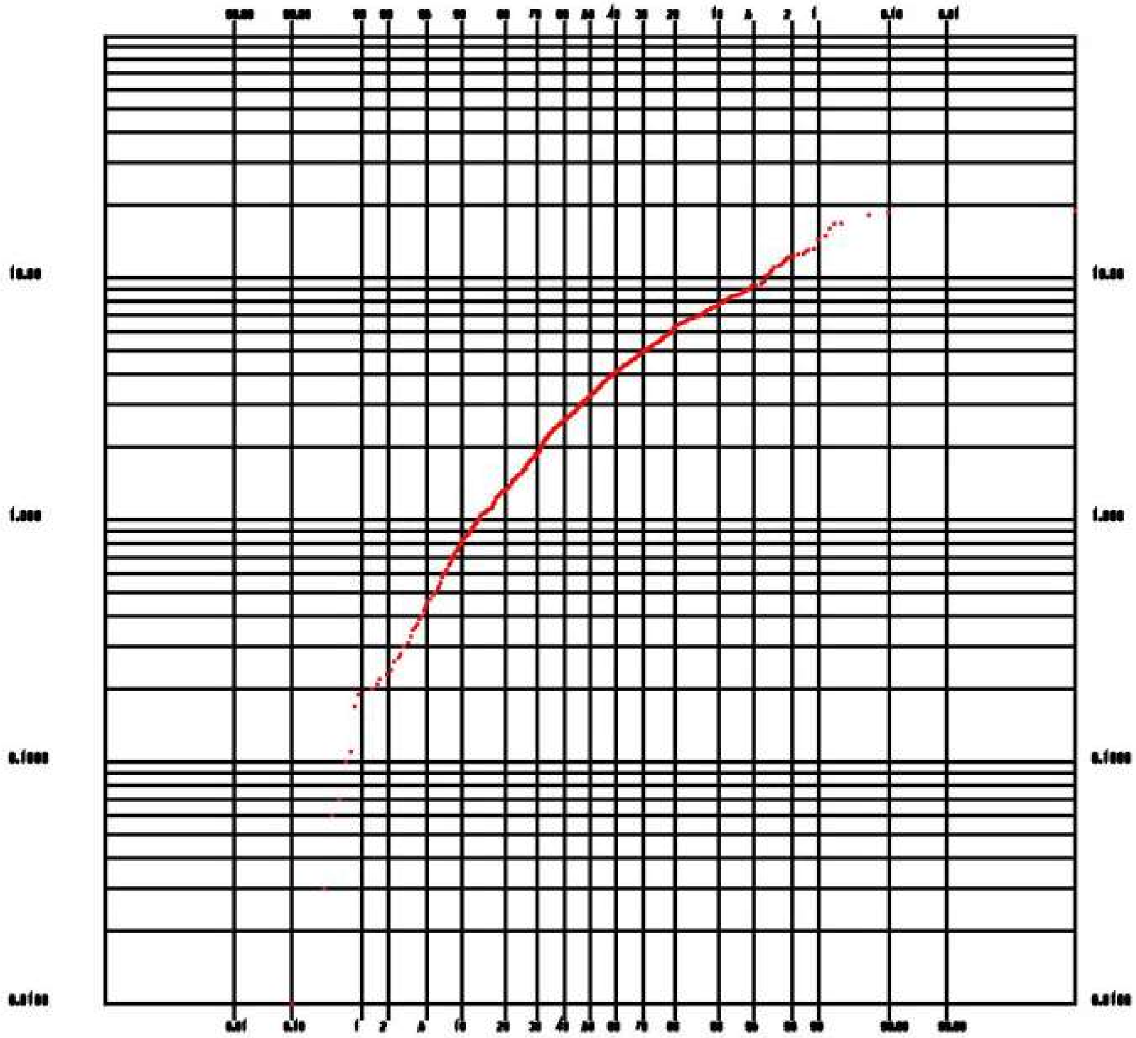


** PROBABILITY DISTRIBUTION PLOT OF CU **

ITEM	CU	NATURAL LOGS	
NUMBER	480	NUMBER	480
MEAN	6.1170	MEAN	1.4180
MINIMUM	0.0200	MINIMUM	-3.9120
MAXIMUM	20.3100	MAXIMUM	3.0110
VARIANCE	9.6820	VARIANCE	0.5760
ST.DEV.	3.1120	ST.DEV.	0.7590

Cu Zone 4

FIGURE 25-5 PROBABILITY PLOT CU – DEEP ZONE SPEC ORE (ZONE 5)

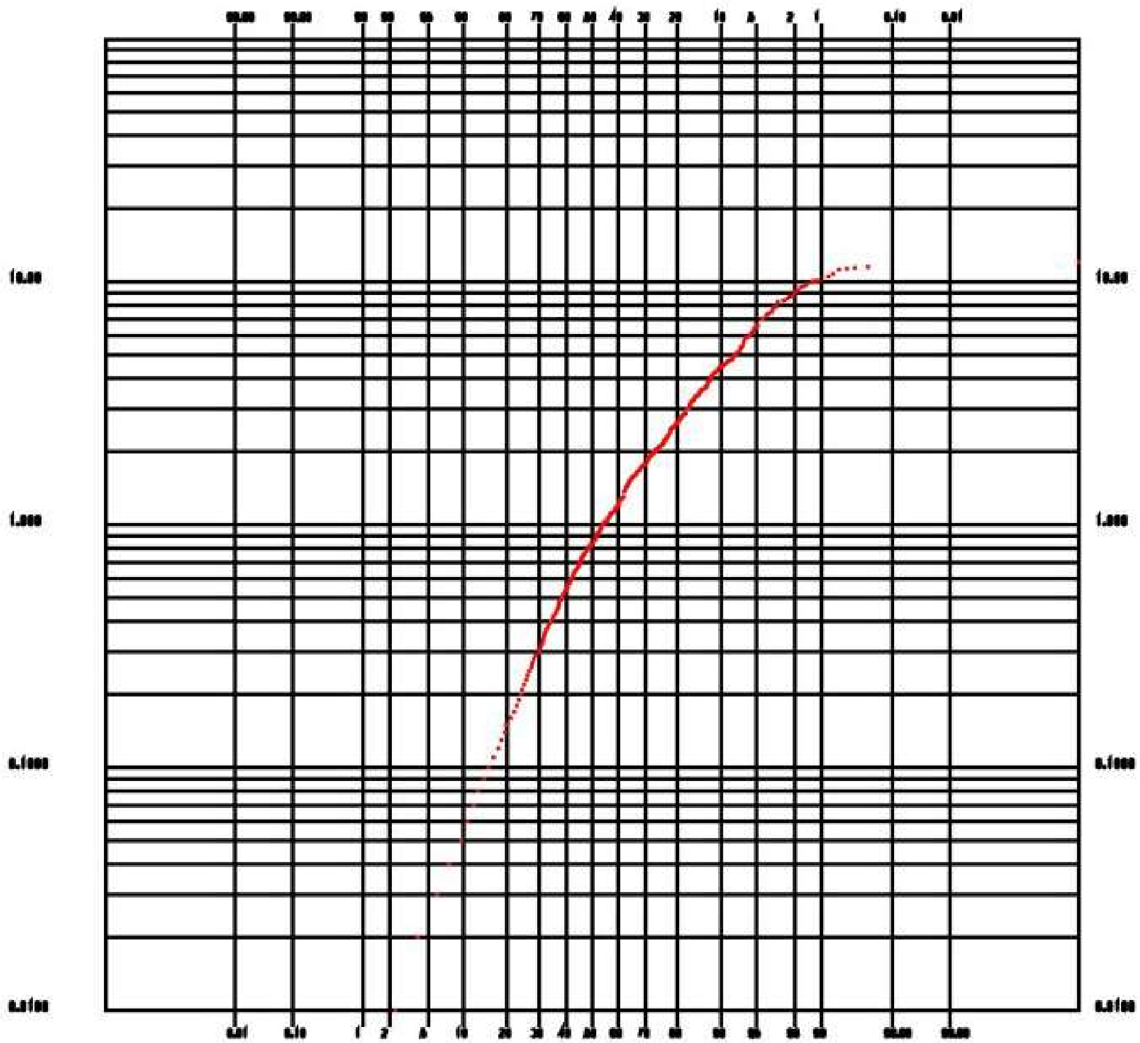


** PROBABILITY DISTRIBUTION PLOT OF CU **

ITEM	CU	NATURAL LOGS	
NUMBER	974	NUMBER	974
MEAN	3.9600	MEAN	1.0220
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	18.9300	MAXIMUM	2.9410
VARIANCE	9.2460	VARIANCE	0.9280
ST.DEV.	3.0410	ST.DEV.	0.9630

Cu Zone 5

FIGURE 25-6 PROBABILITY PLOT CU – DEEP ZONE STOCKWORK ORE (ZONE 6)

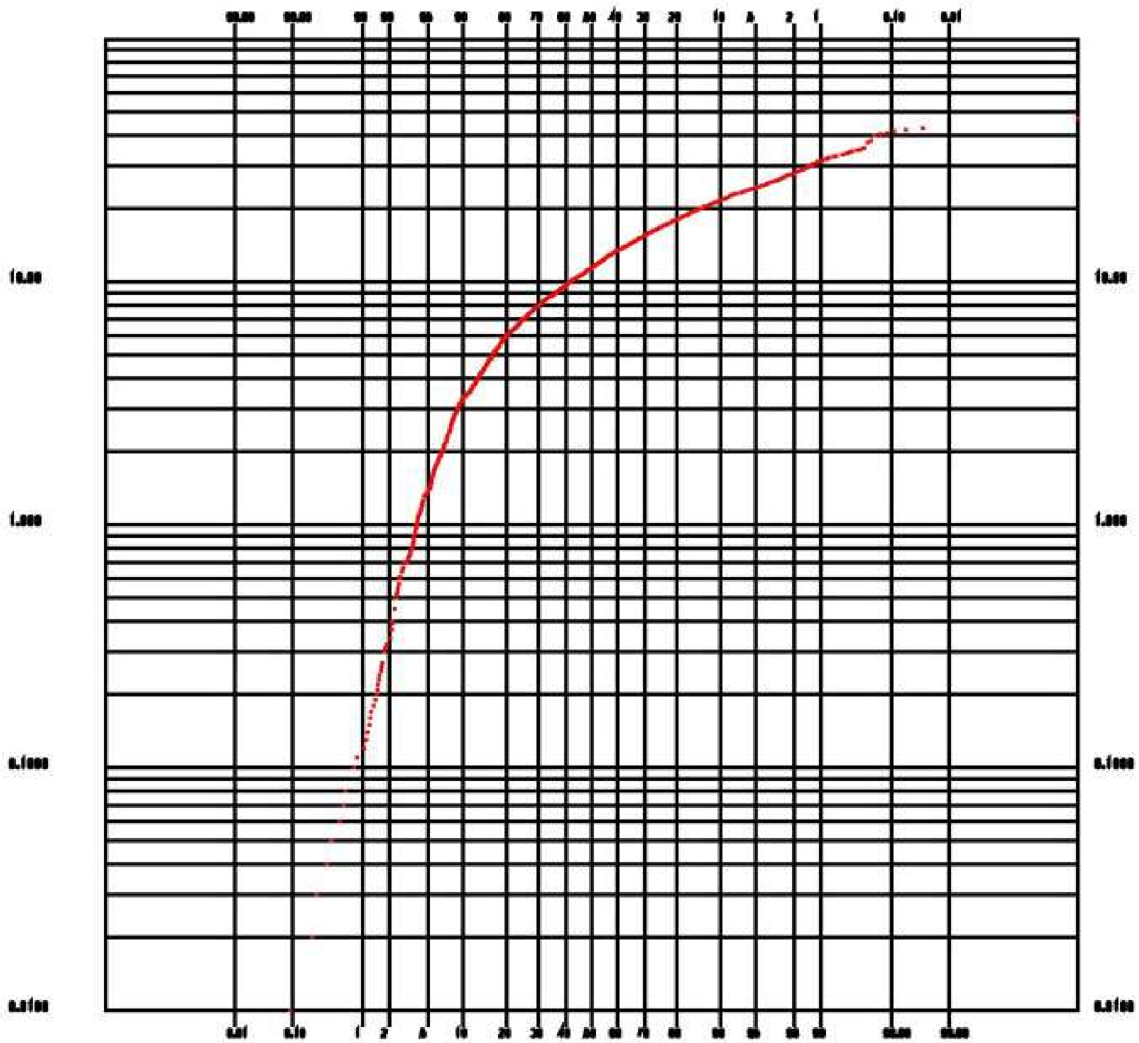


** PROBABILITY DISTRIBUTION PLOT OF CU **

ITEM	CU	NATURAL LOGS	
NUMBER	843	NUMBER	843
MEAN	1.6780	MEAN	-0.4360
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	12.0300	MAXIMUM	2.4830
VARIANCE	4.8230	VARIANCE	2.6620
ST.DEV.	2.1960	ST.DEV.	1.6320

Cu Zone 6

FIGURE 25-7 PROBABILITY PLOT ZN – MAIN ZONE CLASTIC ORE (ZONE 1)

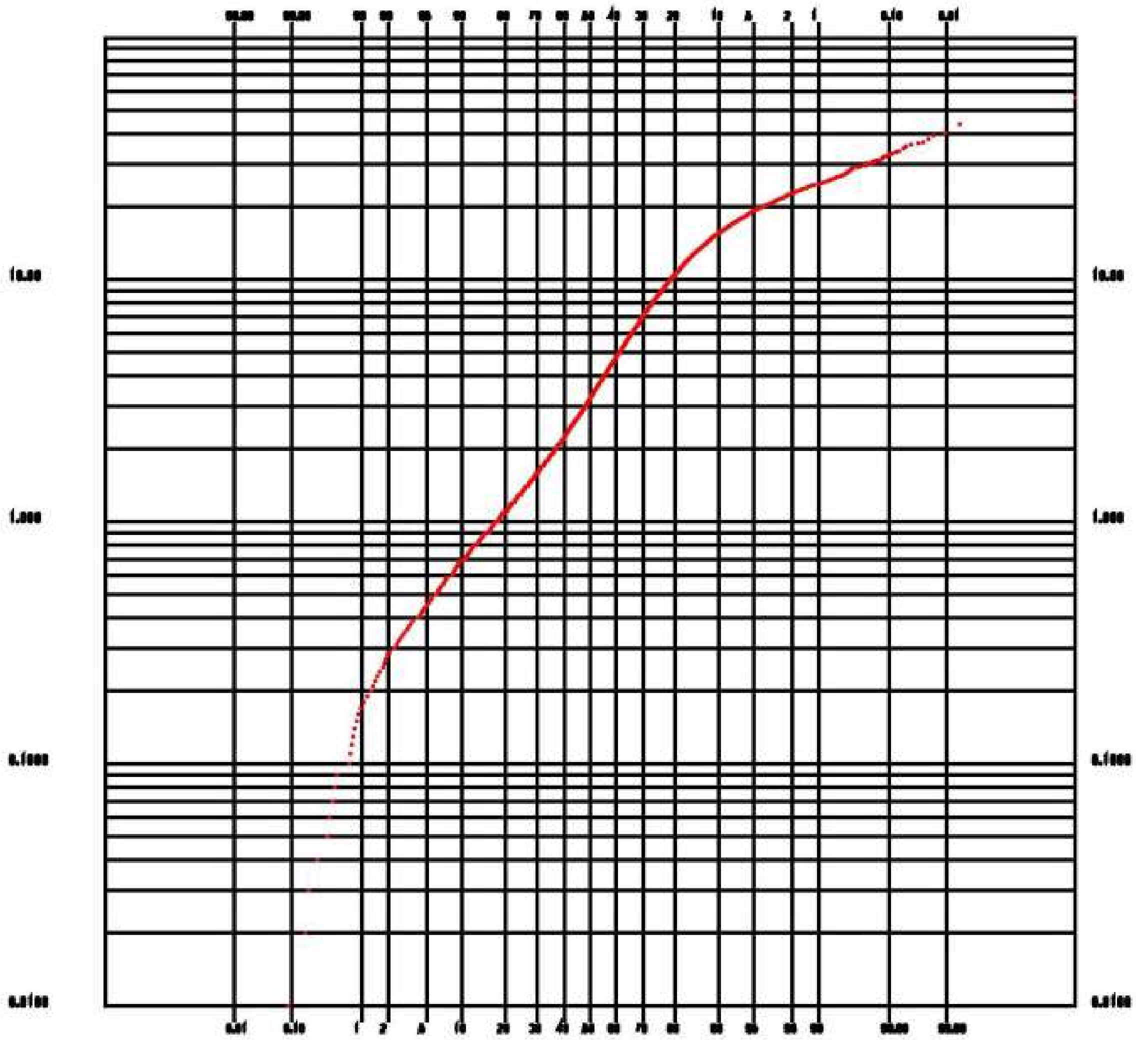


** PROBABILITY DISTRIBUTION PLOT OF ZN **

ITEM	ZN	NATURAL LOGS	
NUMBER	3392	NUMBER	3392
MEAN	12.2080	MEAN	2.2160
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	47.1000	MAXIMUM	3.8520
VARIANCE	50.4940	VARIANCE	0.9870
ST.DEV.	7.1060	ST.DEV.	0.9940

Zn Zone 1

FIGURE 25-8 PROBABILITY PLOT ZN – MAIN ZONE SPEC ORE (ZONE 2)

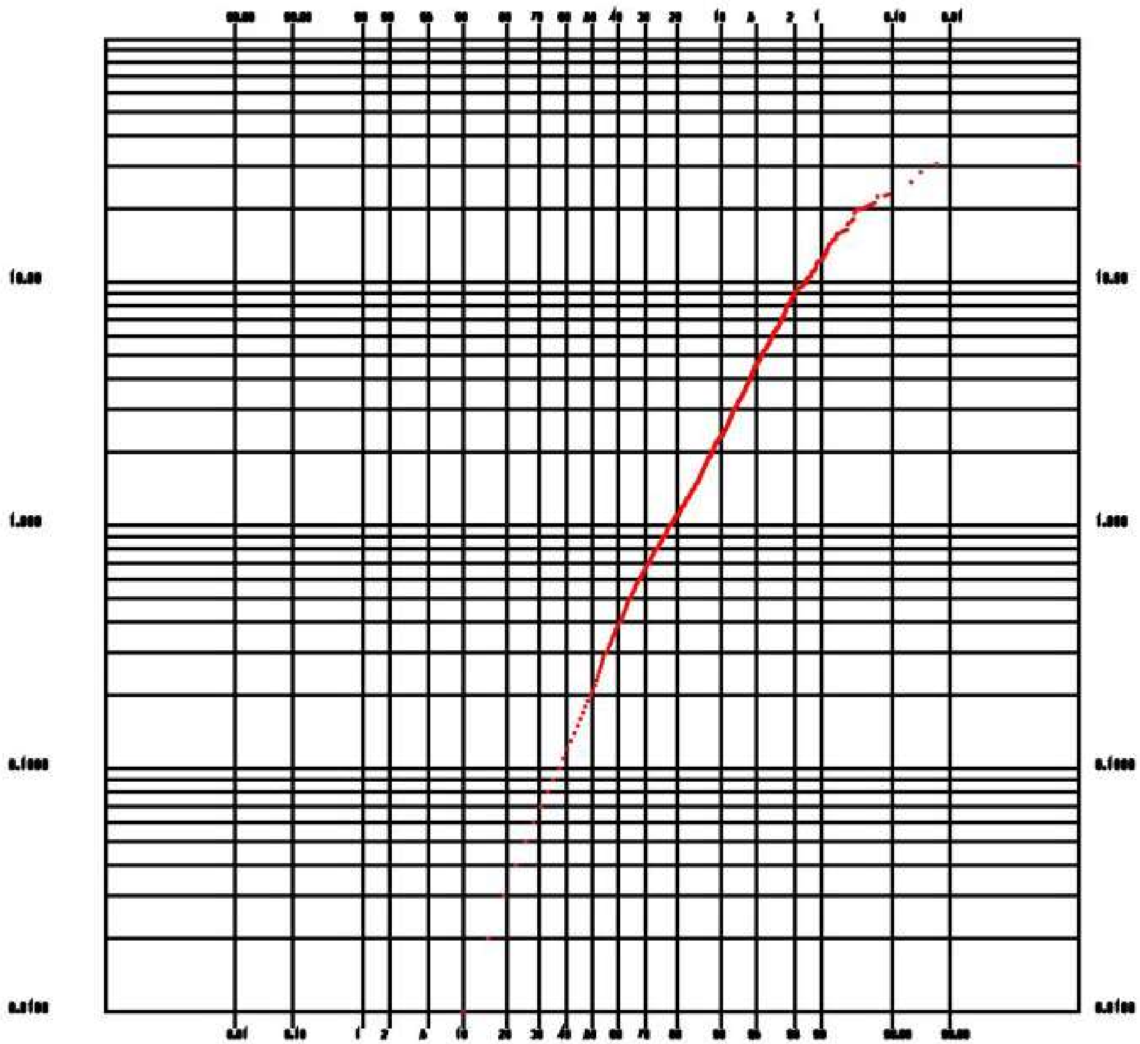


** PROBABILITY DISTRIBUTION PLOT OF ZN **

ITEM	ZN	NATURAL LOGS	
NUMBER	18713	NUMBER	18713
MEAN	5.8930	MEAN	1.1650
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	56.3000	MAXIMUM	4.0310
VARIANCE	38.4290	VARIANCE	1.4620
ST.DEV.	6.1990	ST.DEV.	1.2090

Zn Zone 2

FIGURE 25-9 PROBABILITY PLOT ZN – MAIN ZONE STOCKWORK ORE (ZONE 3)

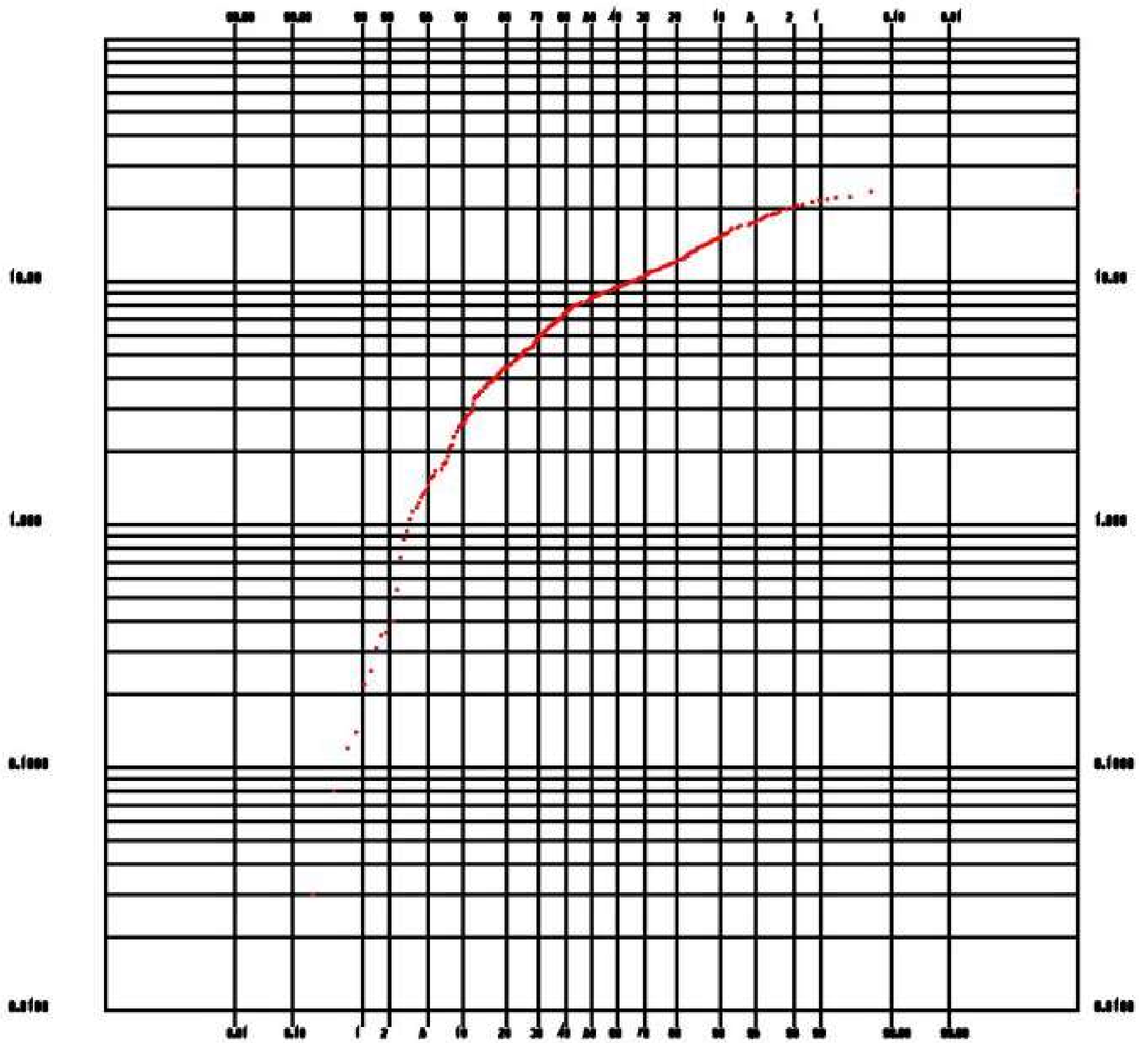


** PROBABILITY DISTRIBUTION PLOT OF ZN **

ITEM	ZN	NATURAL LOGS	
NUMBER	6100	NUMBER	6100
MEAN	0.9990	MEAN	-1.5370
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	30.6800	MAXIMUM	3.4240
VARIANCE	5.7390	VARIANCE	3.4250
ST.DEV.	2.3960	ST.DEV.	1.8510

Zn Zone 3

FIGURE 25-10 PROBABILITY PLOT ZN – DEEP ZONE CLASTIC ORE (ZONE 4)

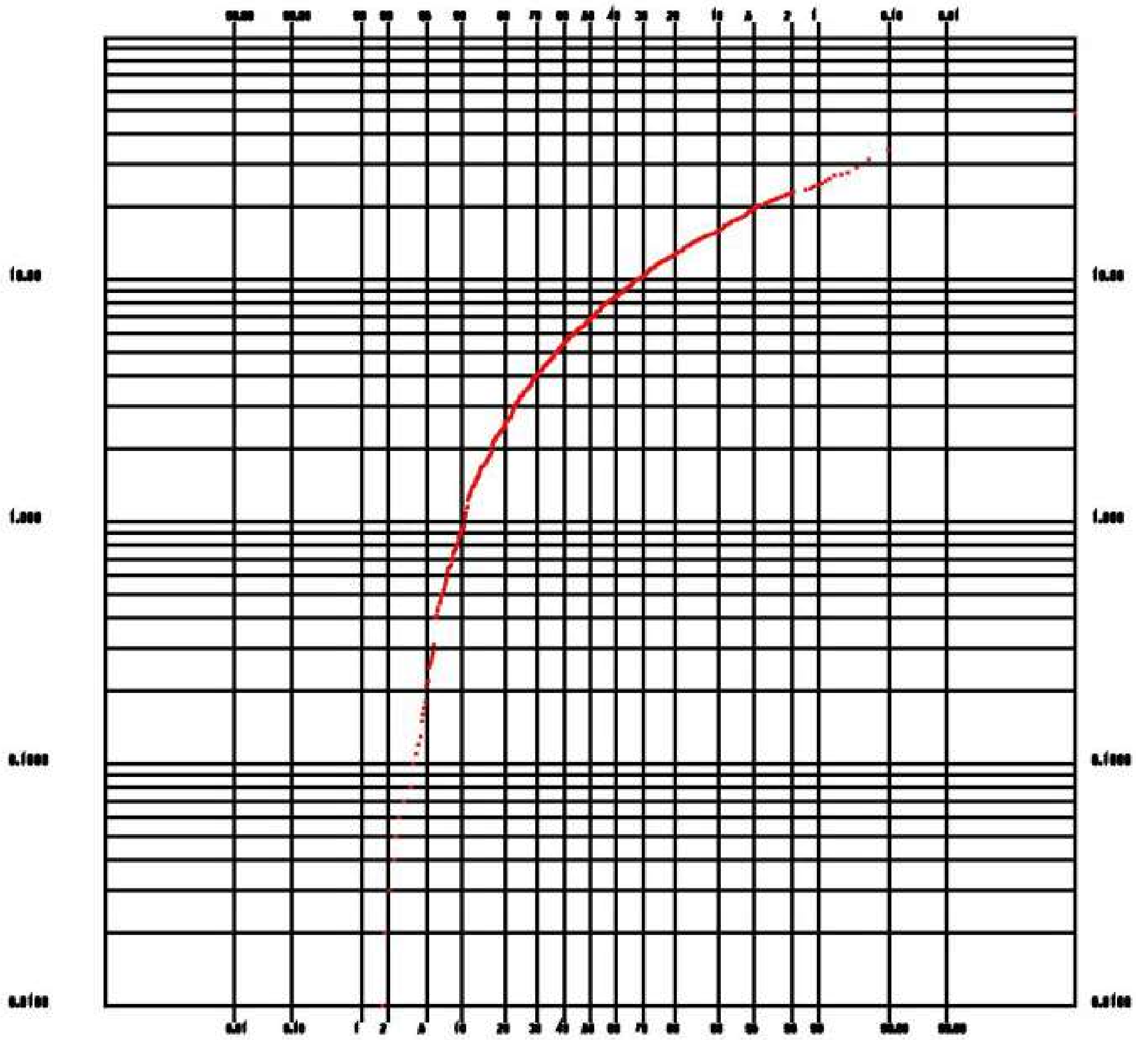


** PROBABILITY DISTRIBUTION PLOT OF ZN **

ITEM	ZN	NATURAL LOGS	
NUMBER	480	NUMBER	480
MEAN	8.7720	MEAN	1.9390
MINIMUM	0.0300	MINIMUM	-3.5070
MAXIMUM	23.6800	MAXIMUM	3.1650
VARIANCE	23.2210	VARIANCE	0.7280
ST.DEV.	4.8190	ST.DEV.	0.8530

Zn Zone 4

FIGURE 25-11 PROBABILITY PLOT ZN – DEEP ZONE SPEC ORE (ZONE 5)

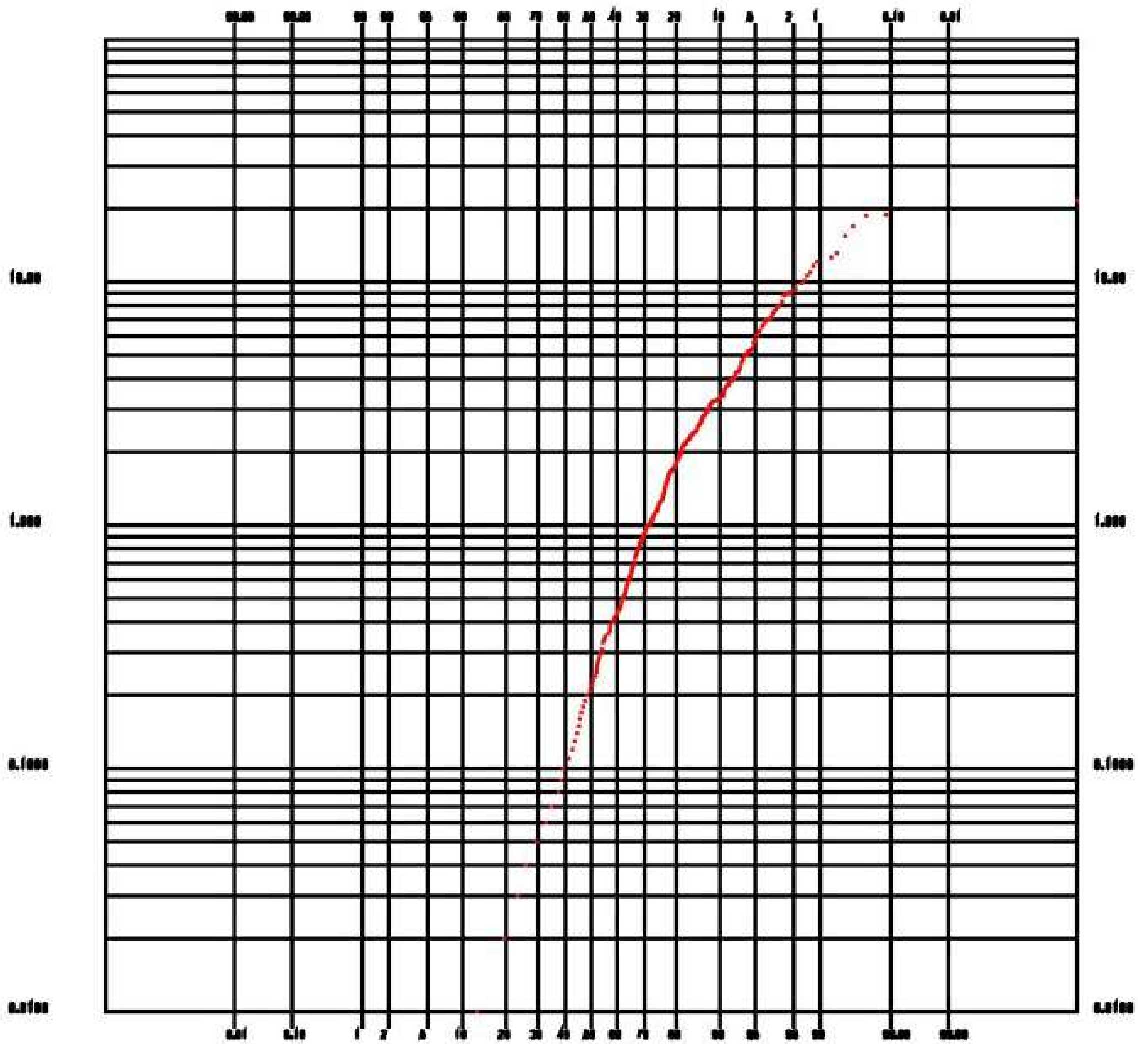


** PROBABILITY DISTRIBUTION PLOT OF ZN **

ITEM	ZN	NATURAL LOGS	
NUMBER	973	NUMBER	973
MEAN	8.0190	MEAN	1.6620
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	48.6000	MAXIMUM	3.8840
VARIANCE	37.0220	VARIANCE	2.0720
ST.DEV.	6.0850	ST.DEV.	1.4390

Zn Zone 5

FIGURE 25-12 PROBABILITY PLOT ZN – DEEP ZONE STOCKWORK ORE (ZONE 6)



** PROBABILITY DISTRIBUTION PLOT OF ZN **

ITEM	ZN	NATURAL LOGS	
NUMBER	832	NUMBER	832
MEAN	1.2300	MEAN	-1.4890
MINIMUM	0.0100	MINIMUM	-4.6050
MAXIMUM	21.6500	MAXIMUM	3.0750
VARIANCE	5.9920	VARIANCE	4.2610
ST.DEV.	2.4480	ST.DEV.	2.0640

Zn Zone 6

26 APPENDIX 4

EqCU DISTRIBUTION – CROSS SECTIONS

Red:	EqCu > 5.0%
Yellow:	3.4% < EqCu < 5.0%
Green:	2.5% < EqCu < 3.4%

FIGURE 26-1 EqCU DISTRIBUTION – CROSS SECTION 1560N

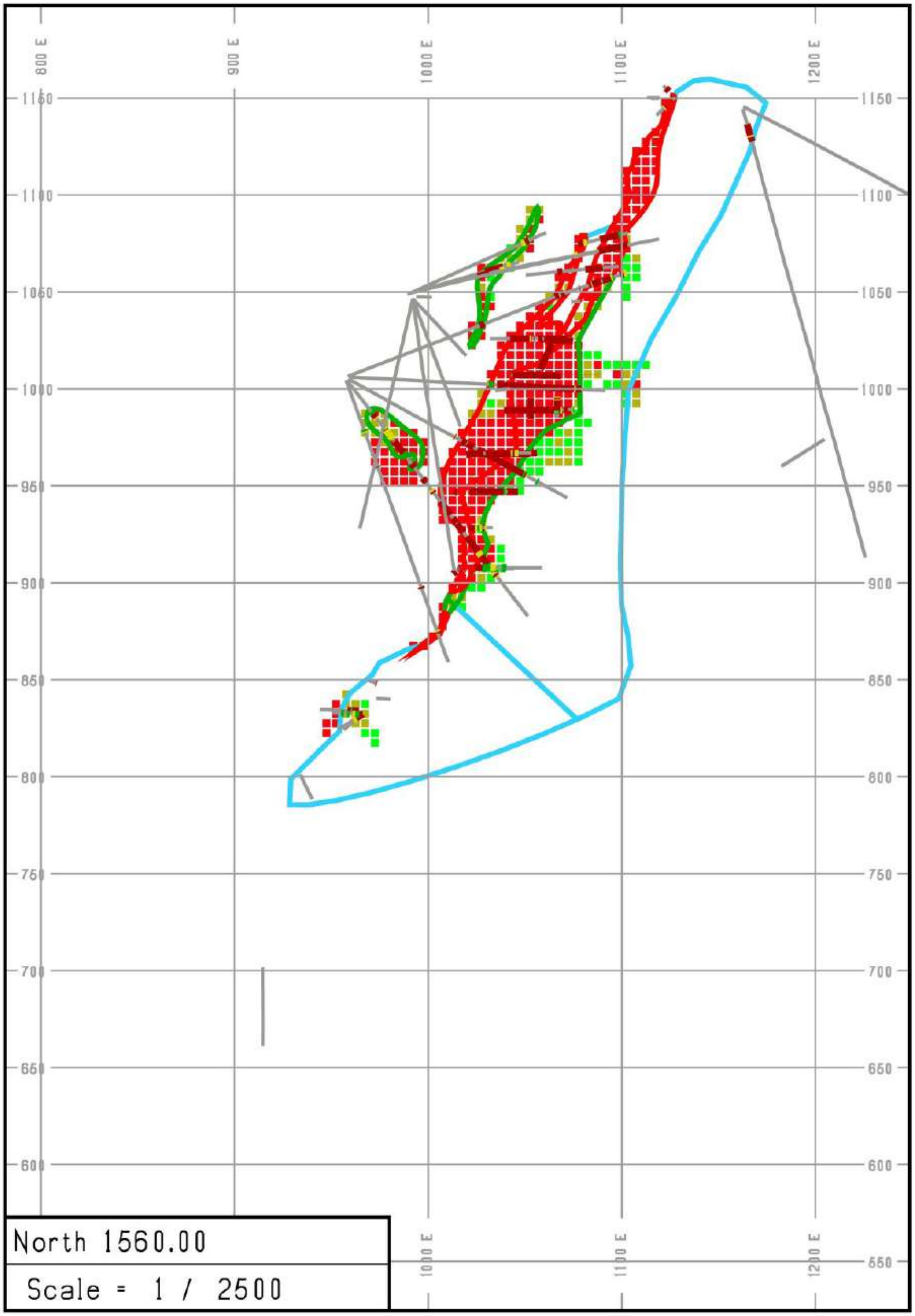


FIGURE 26-2 EqCU DISTRIBUTION – CROSS SECTION 1620N

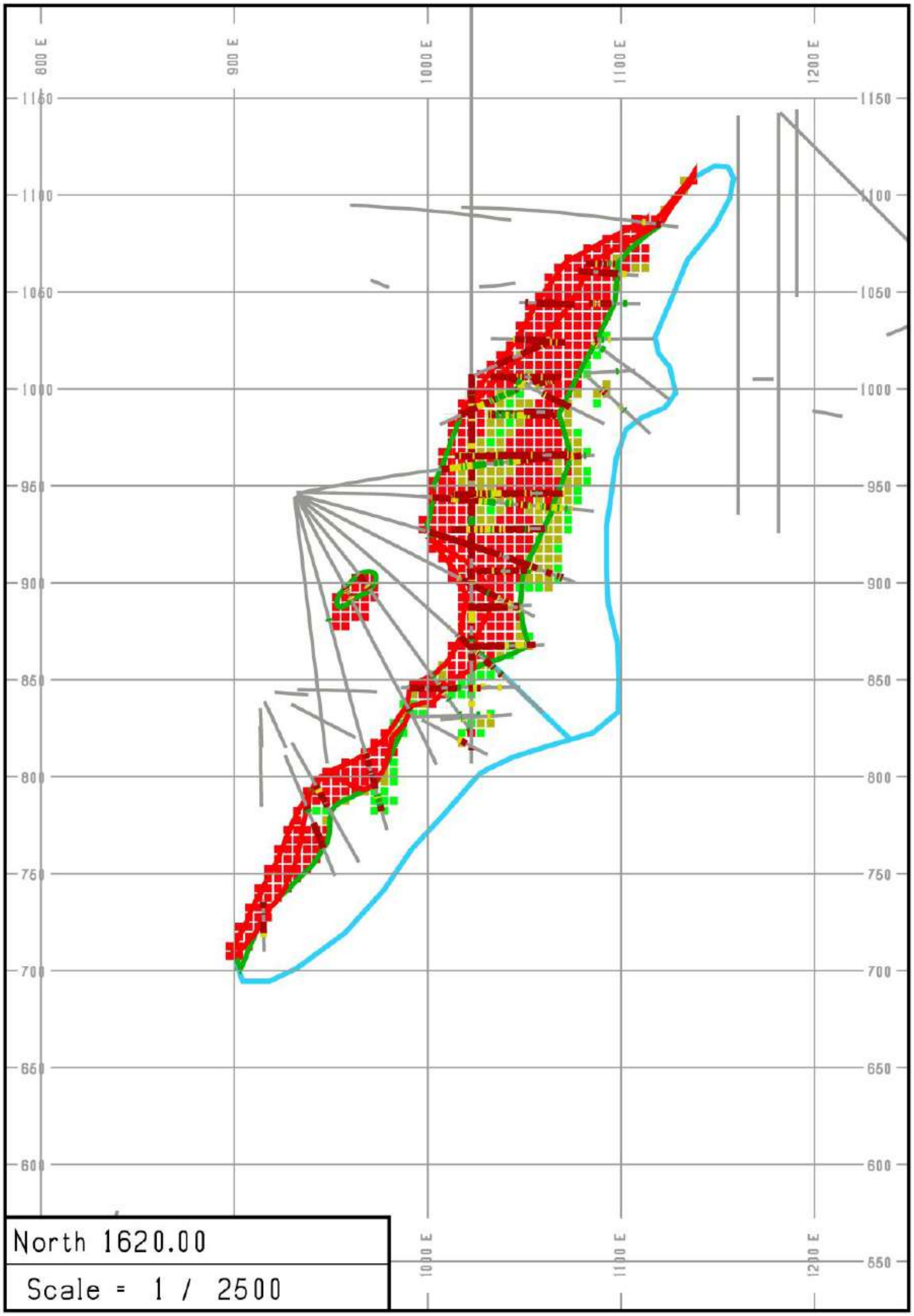


FIGURE 26-3 EqCU DISTRIBUTION – CROSS SECTION 1680N

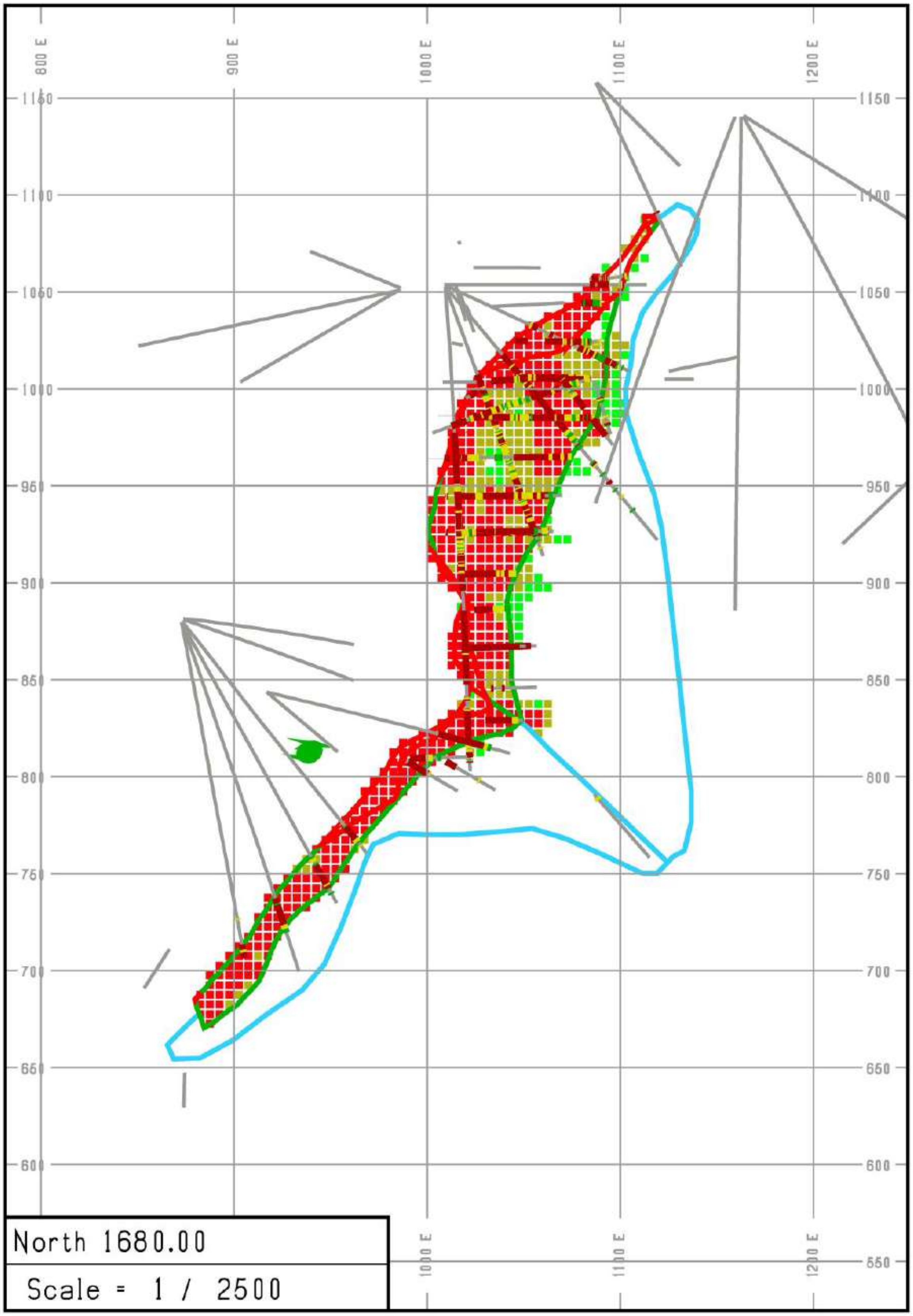


FIGURE 26-4 EqCU DISTRIBUTION – CROSS SECTION 1740N

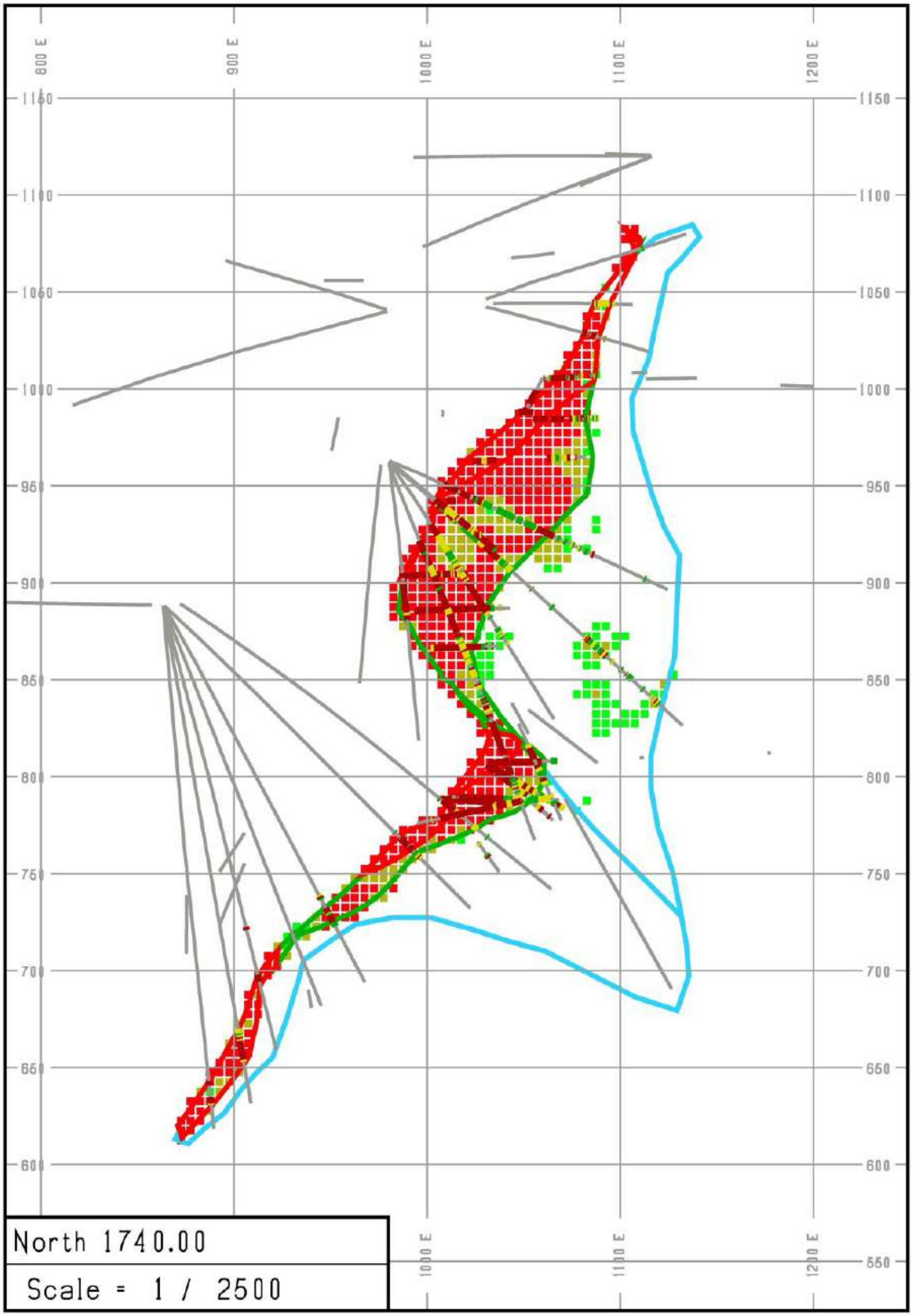


FIGURE 26-5 EqCU DISTRIBUTION – CROSS SECTION 1800N

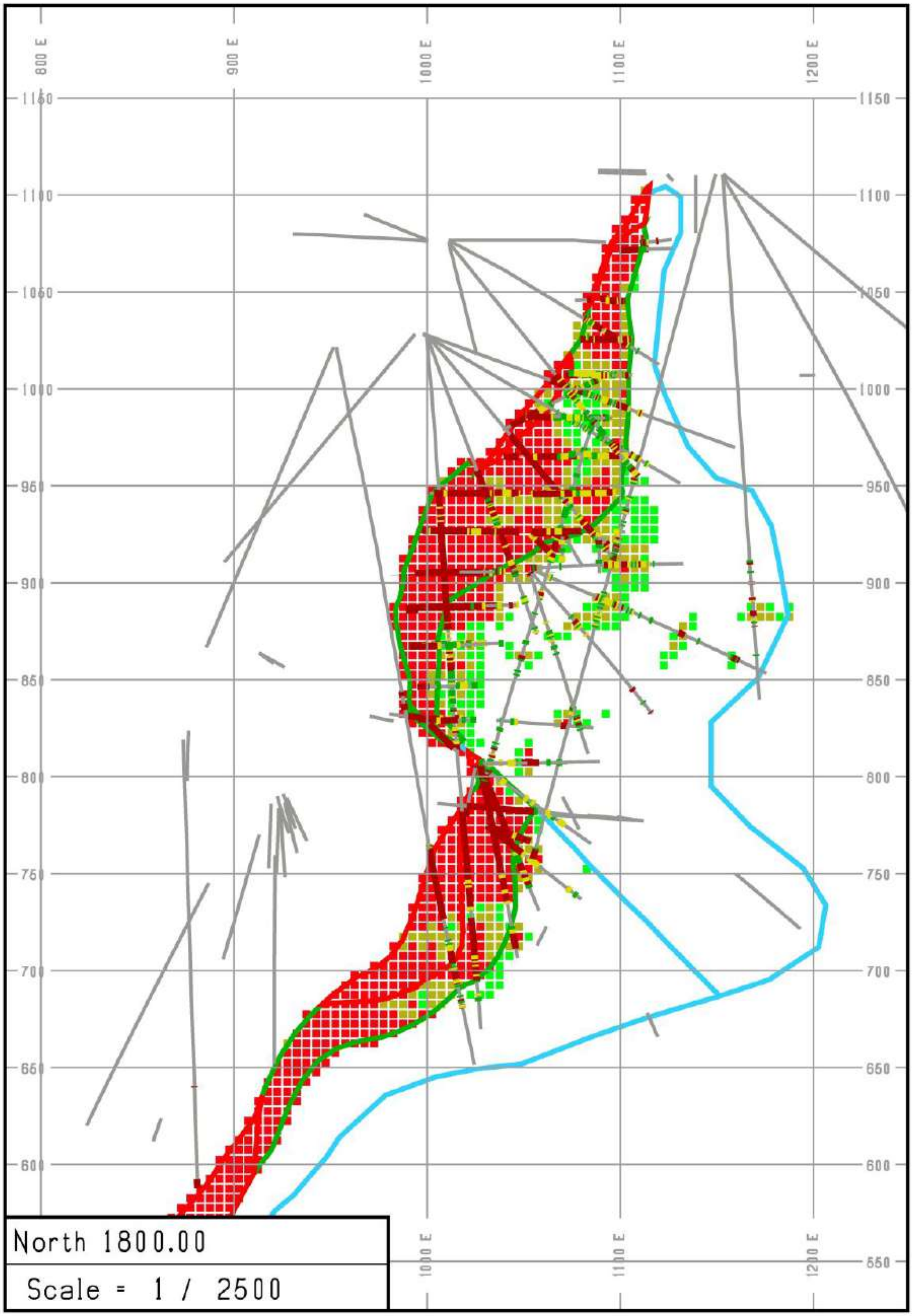


FIGURE 26-6 EqCU DISTRIBUTION – CROSS SECTION 1860N

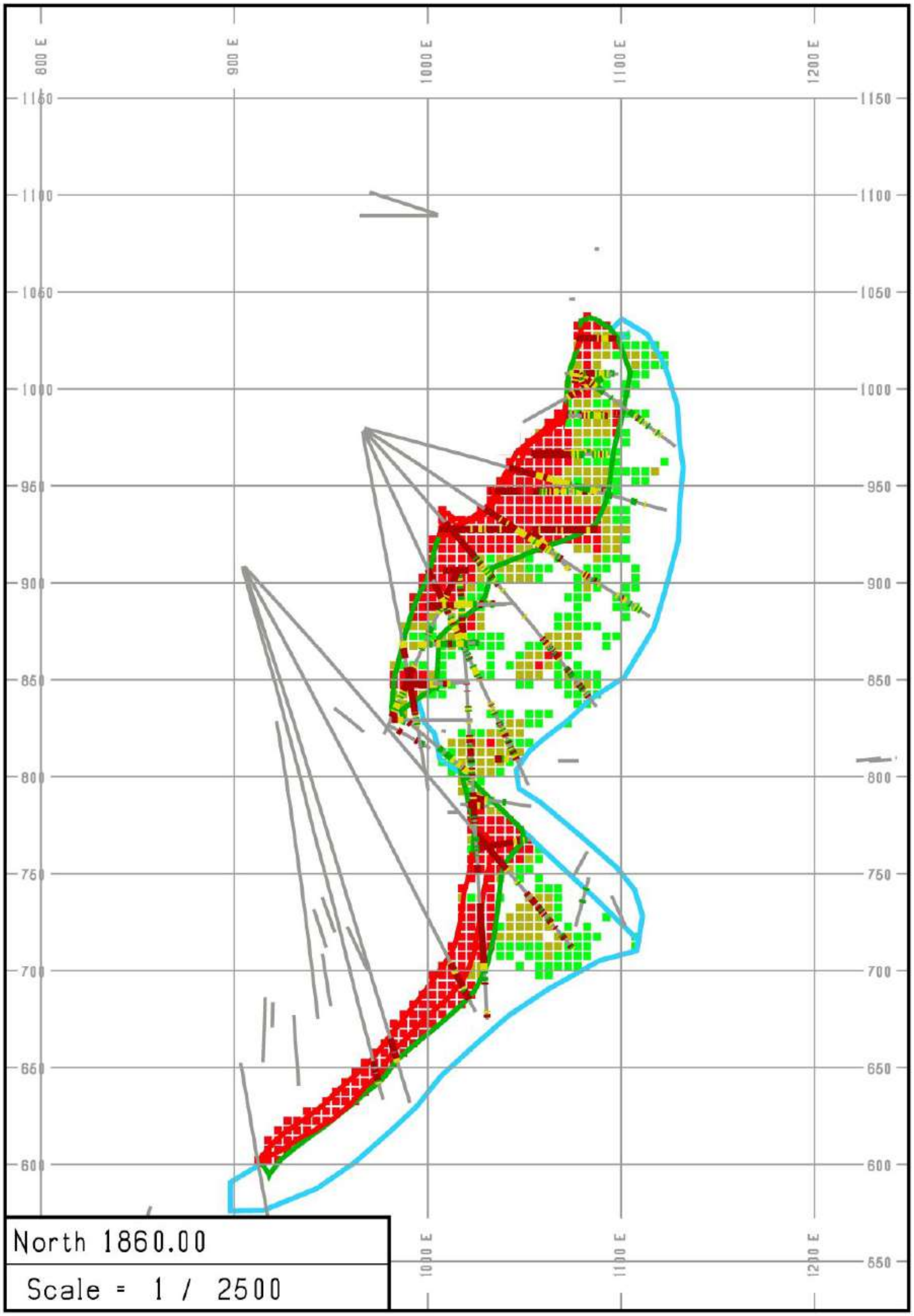


FIGURE 26-7 EqCU DISTRIBUTION – CROSS SECTION 1920N

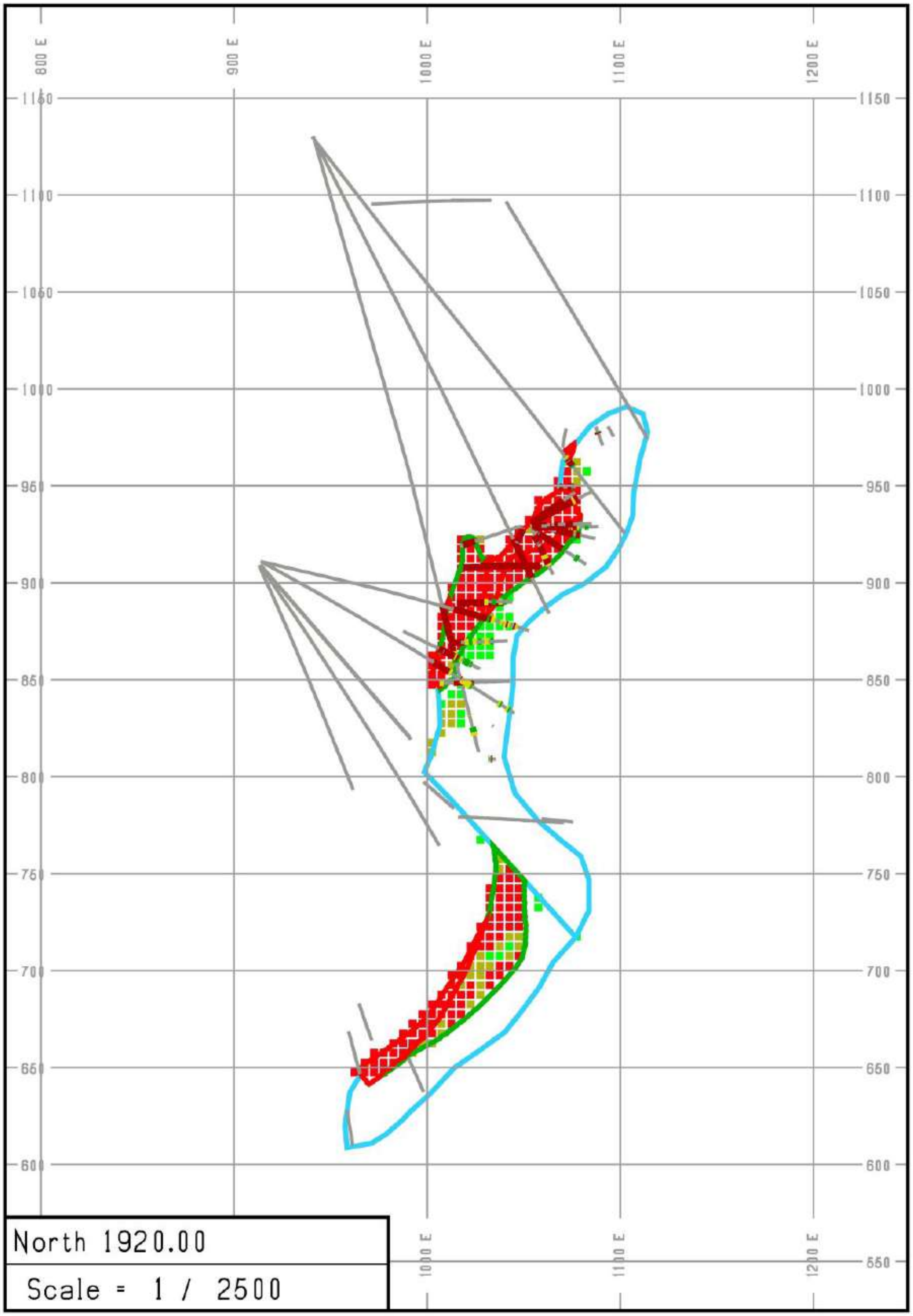


FIGURE 26-8 EqCU DISTRIBUTION – CROSS SECTION 1980N

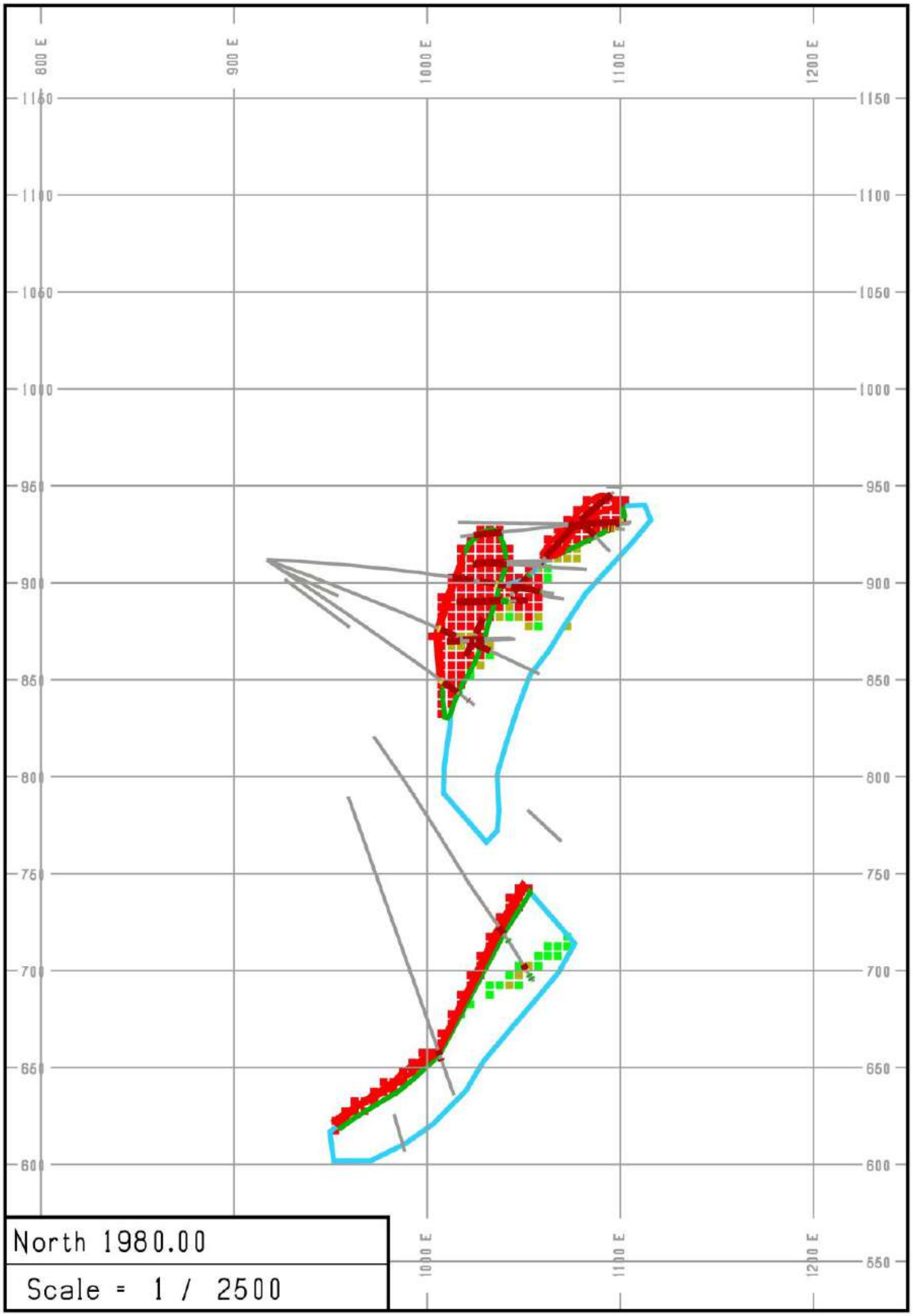


FIGURE 26-9 EqCU DISTRIBUTION – CROSS SECTION 2040N

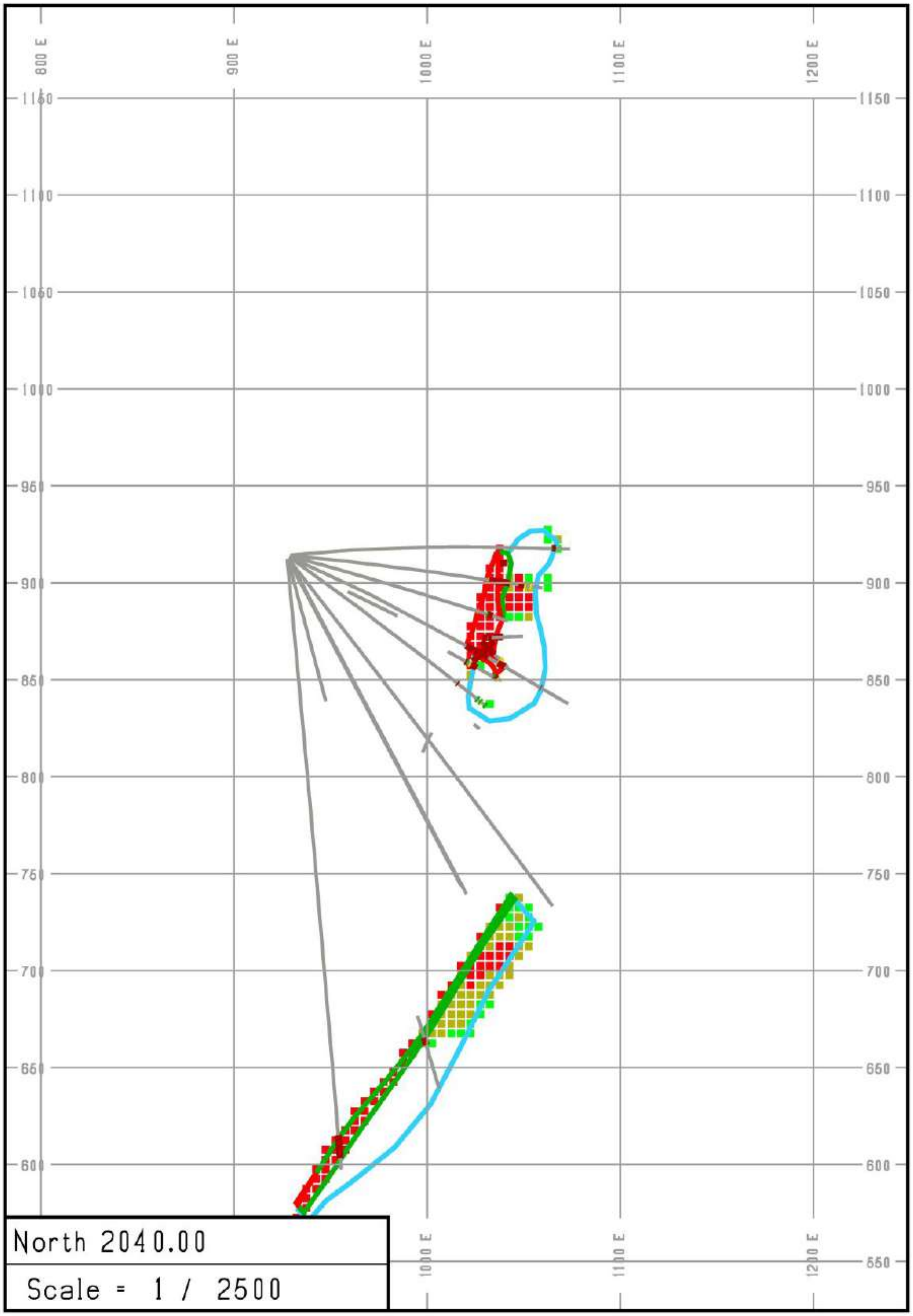
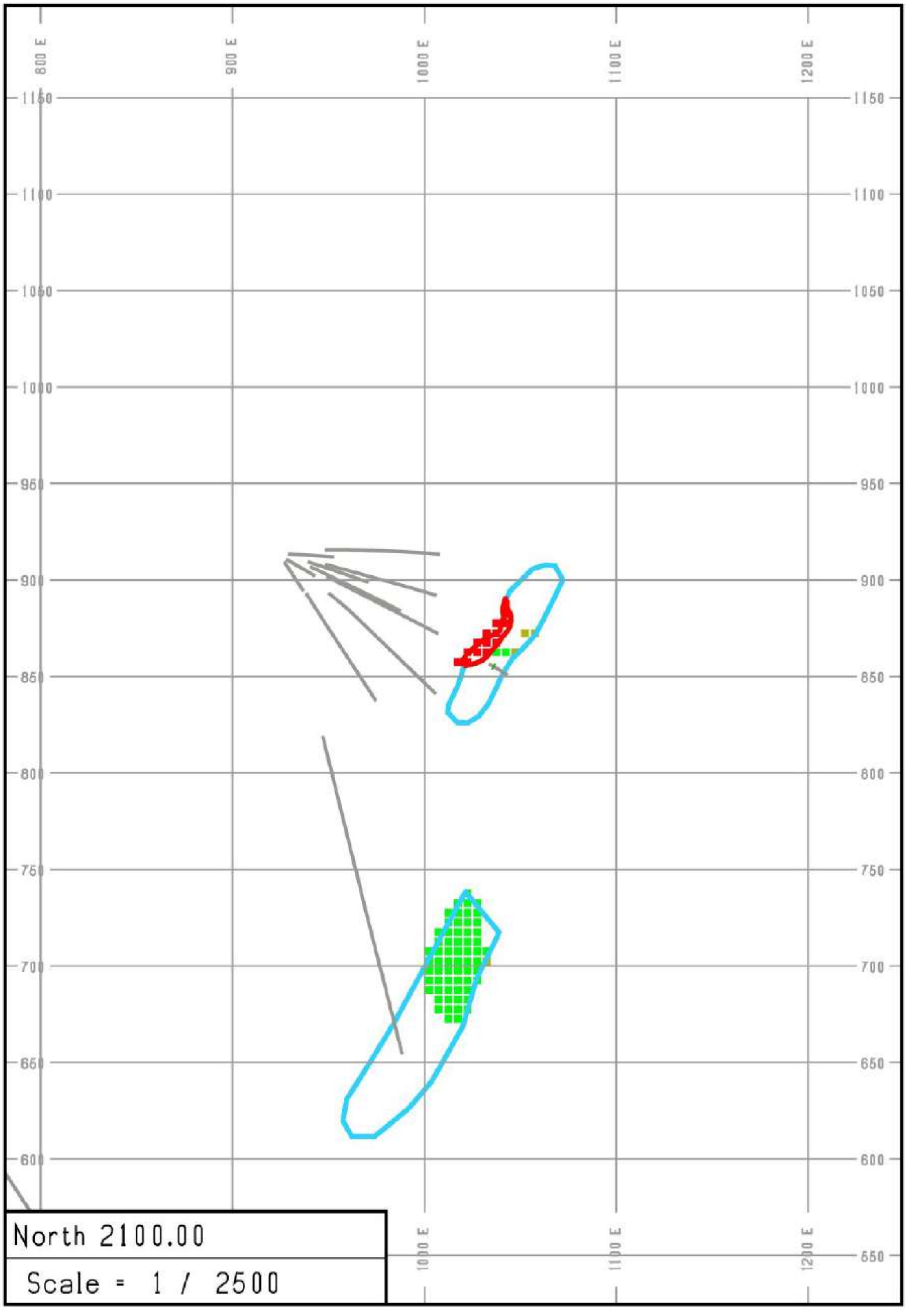


FIGURE 26-10 EqCU DISTRIBUTION – CROSS SECTION 2100N



27 APPENDIX 5

RESOURCE CLASSIFICATION – CROSS SECTIONS

Red:	Measured
Yellow:	Indicated
Green:	Inferred

FIGURE 27-1 RESOURCE CLASSIFICATION – CROSS SECTION 1560N

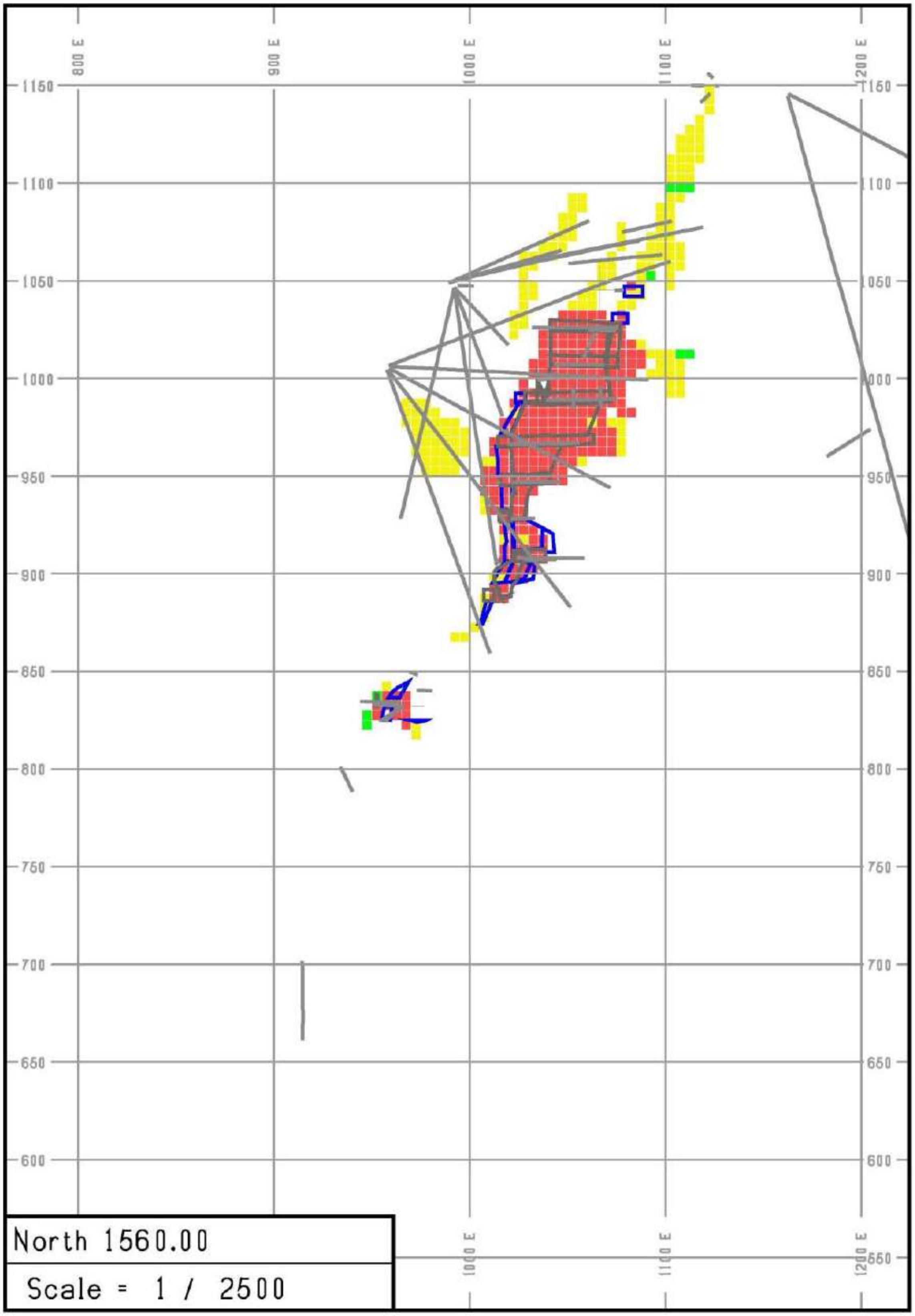


FIGURE 27-2 RESOURCE CLASSIFICATION – CROSS SECTION 1620N

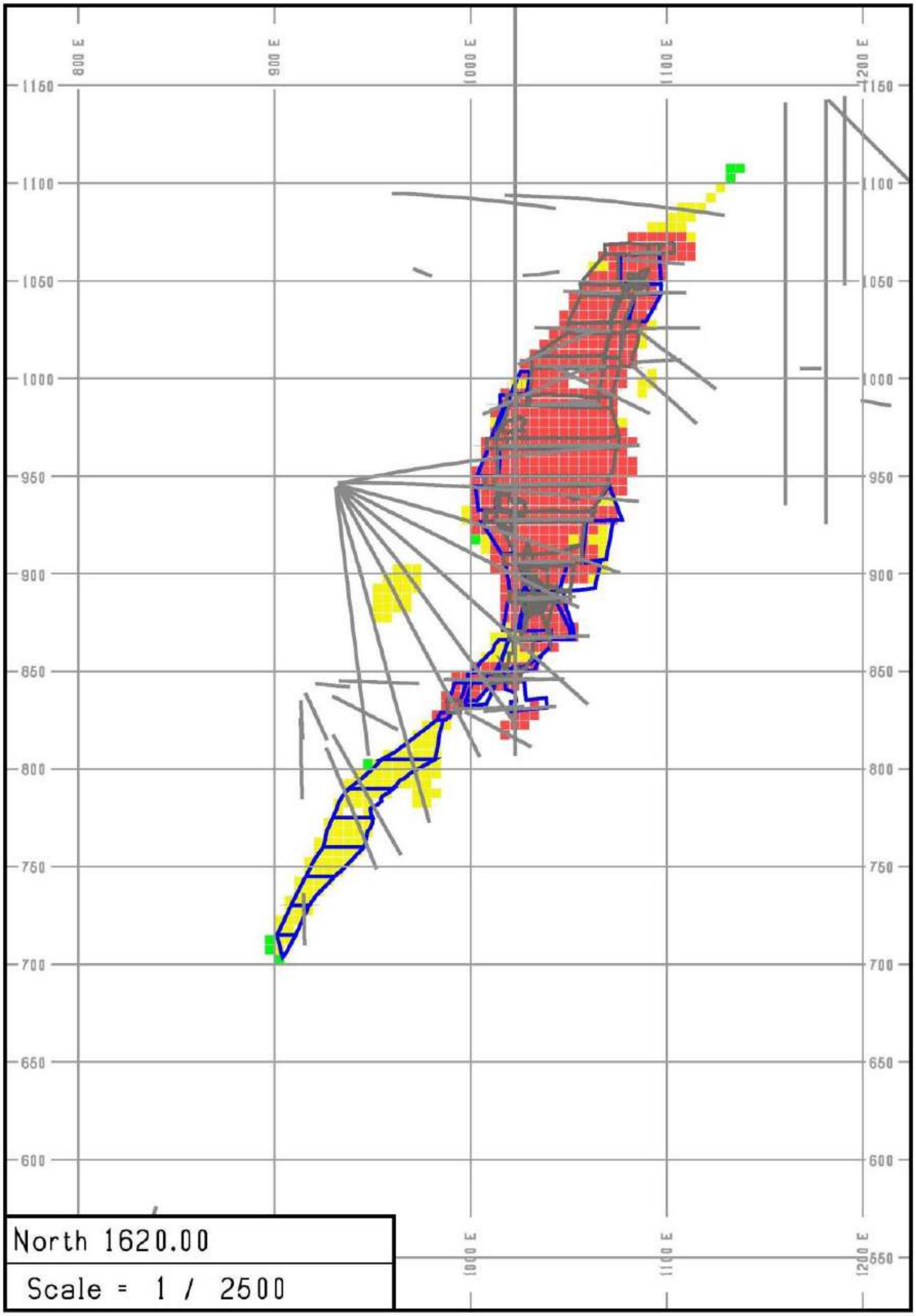


FIGURE 27-3 RESOURCE CLASSIFICATION – CROSS SECTION 1680N

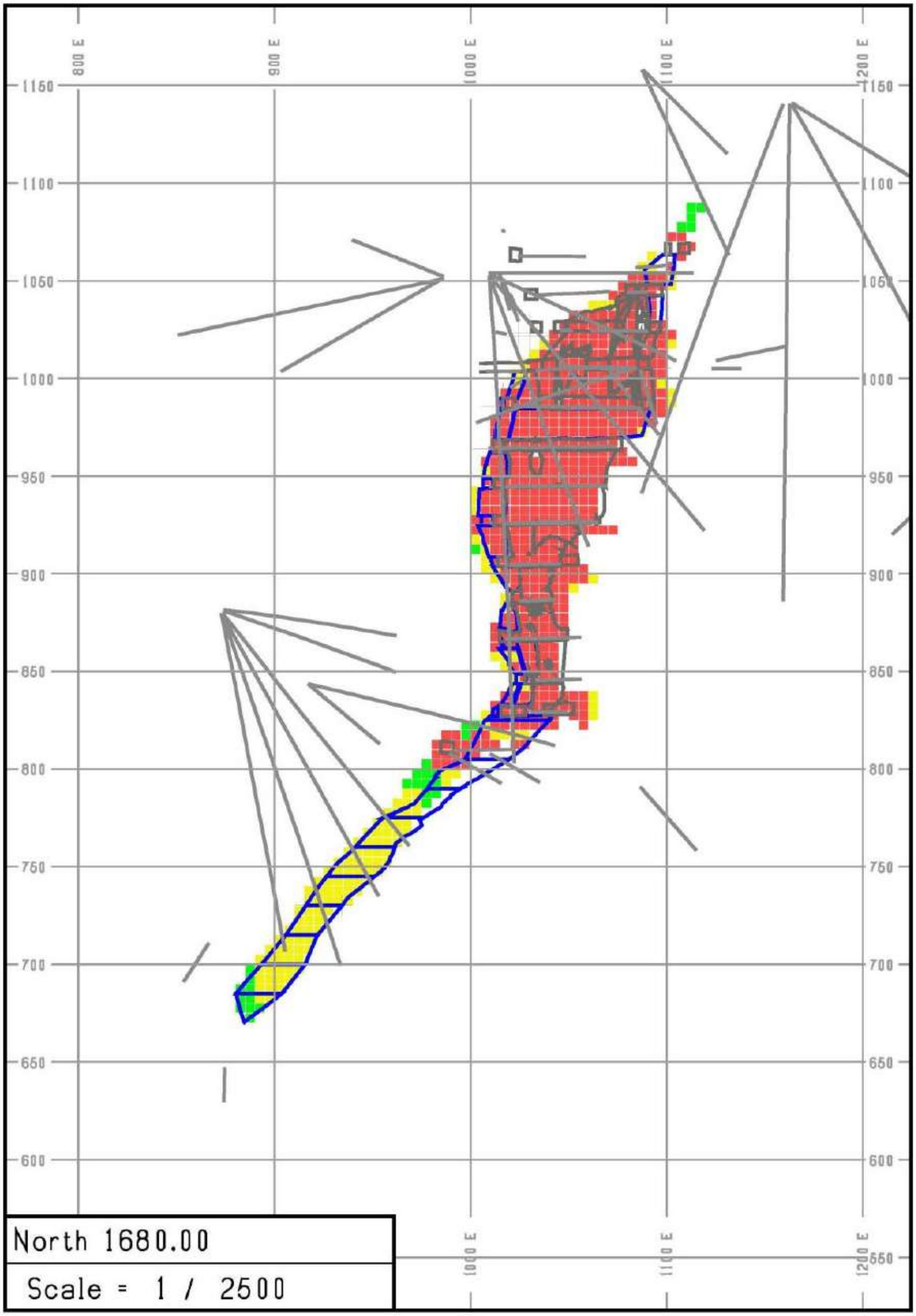


FIGURE 27-4 RESOURCE CLASSIFICATION – CROSS SECTION 1740N

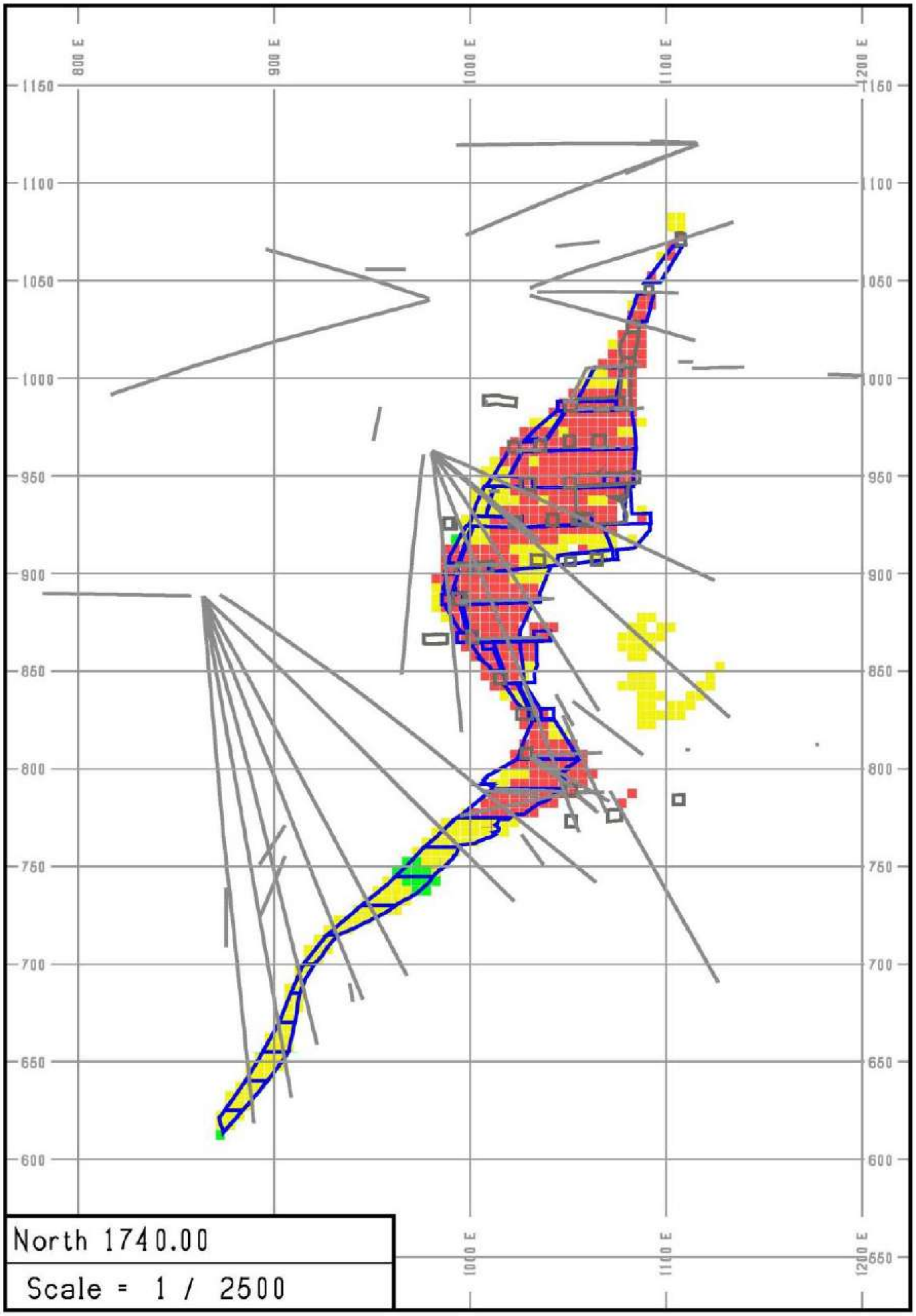


FIGURE 27-5 RESOURCE CLASSIFICATION – CROSS SECTION 1800N

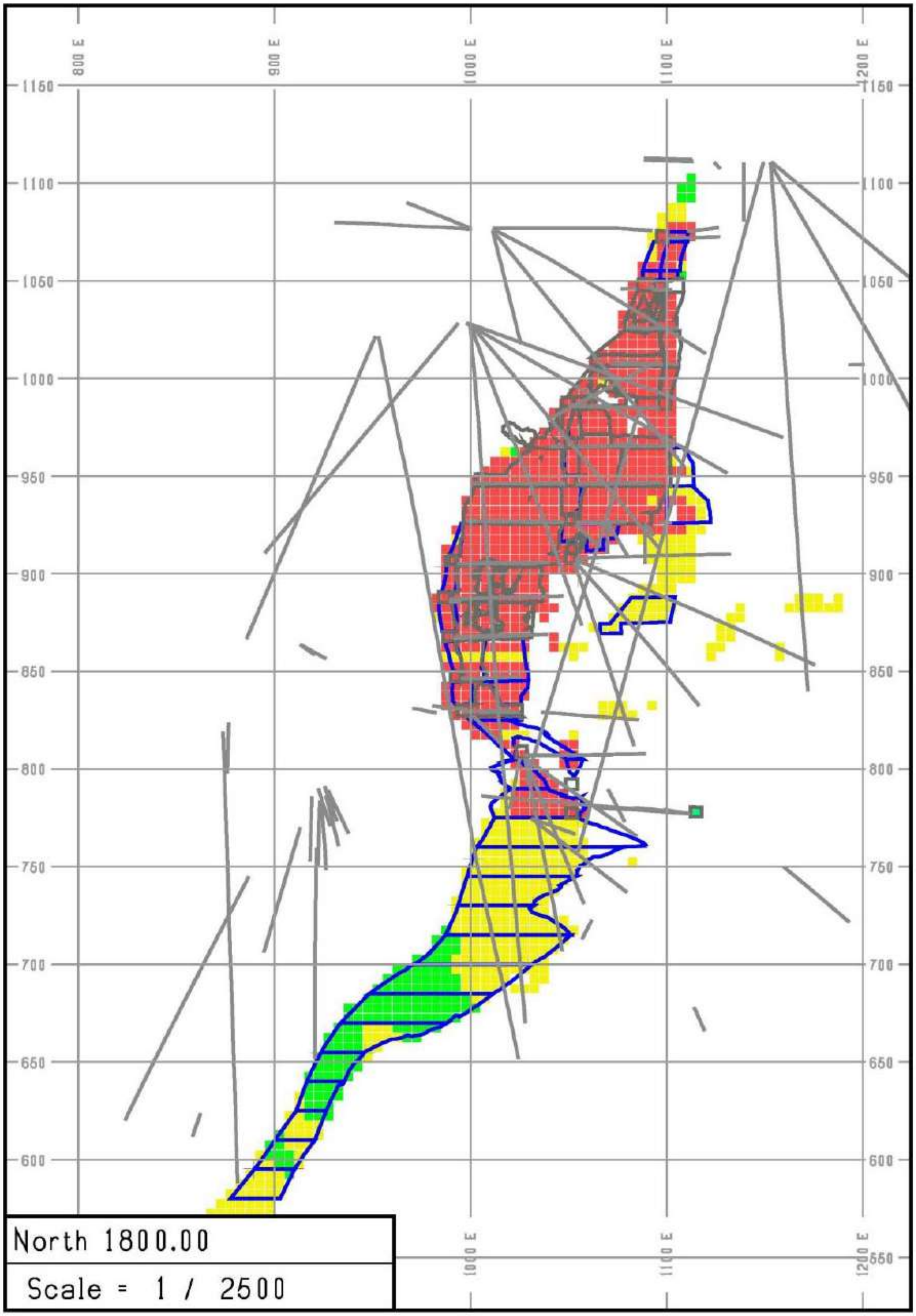


FIGURE 27-6 RESOURCE CLASSIFICATION – CROSS SECTION 1860N

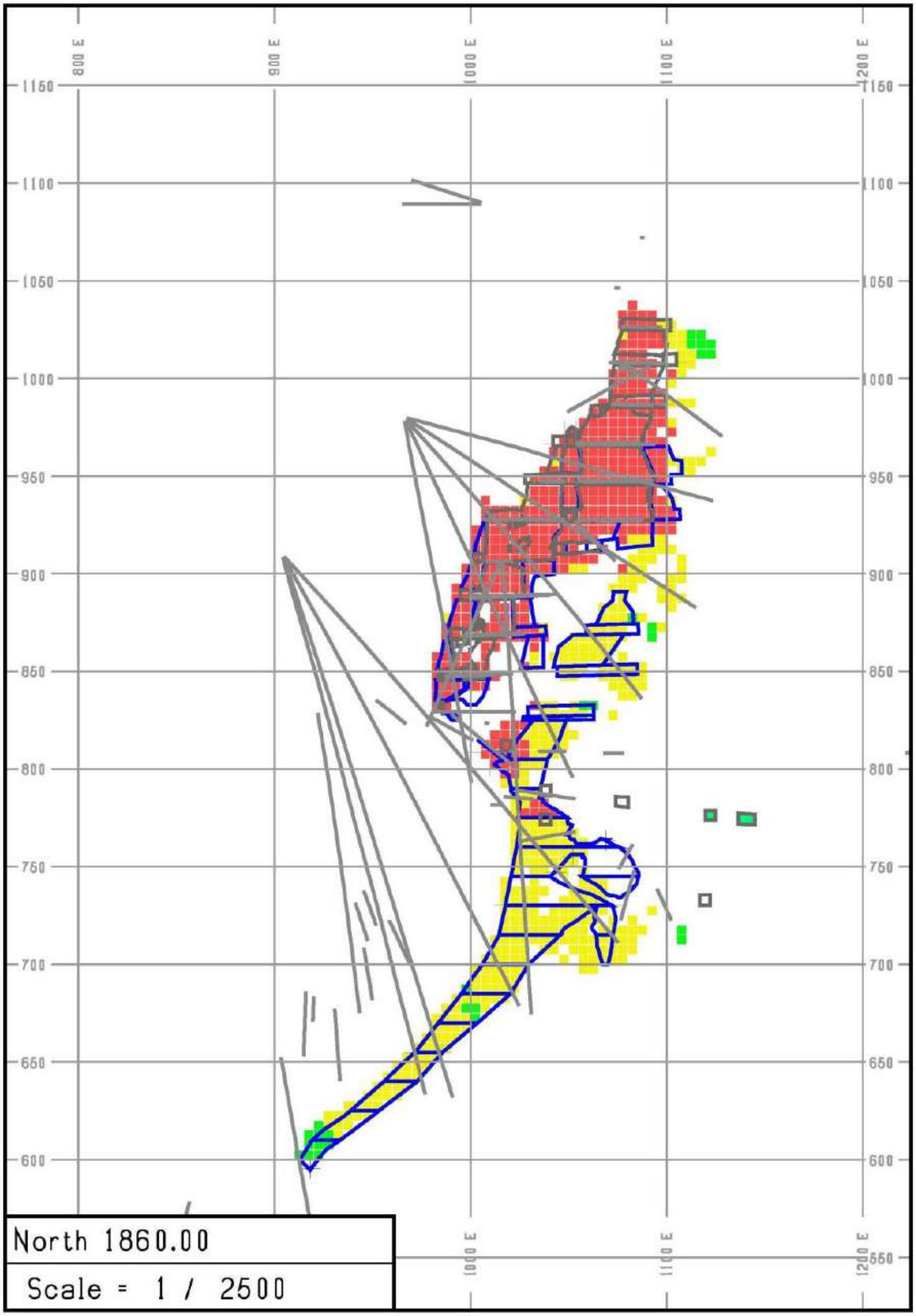


FIGURE 27-7 RESOURCE CLASSIFICATION – CROSS SECTION 1920N

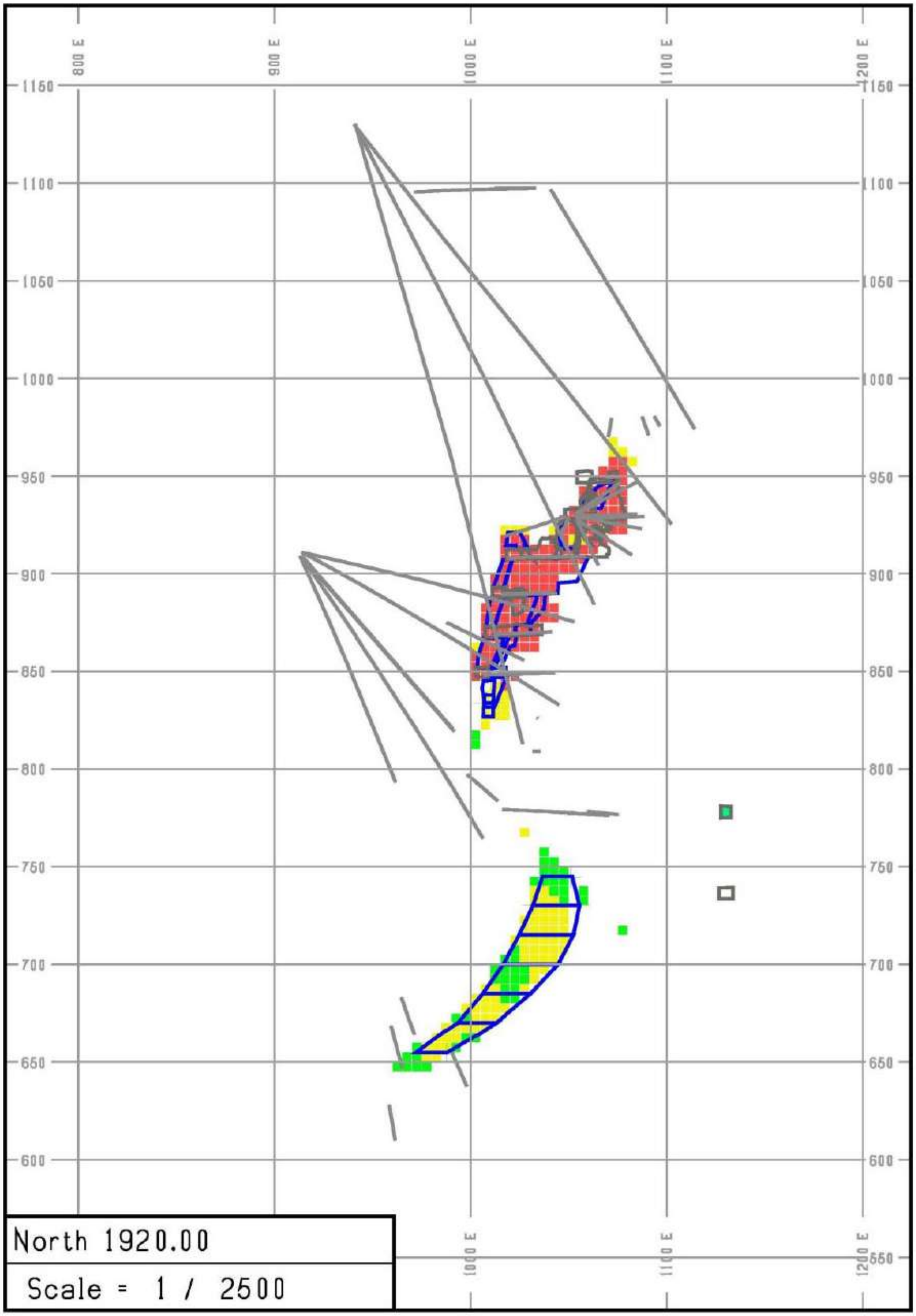


FIGURE 27-8 RESOURCE CLASSIFICATION – CROSS SECTION 1980N

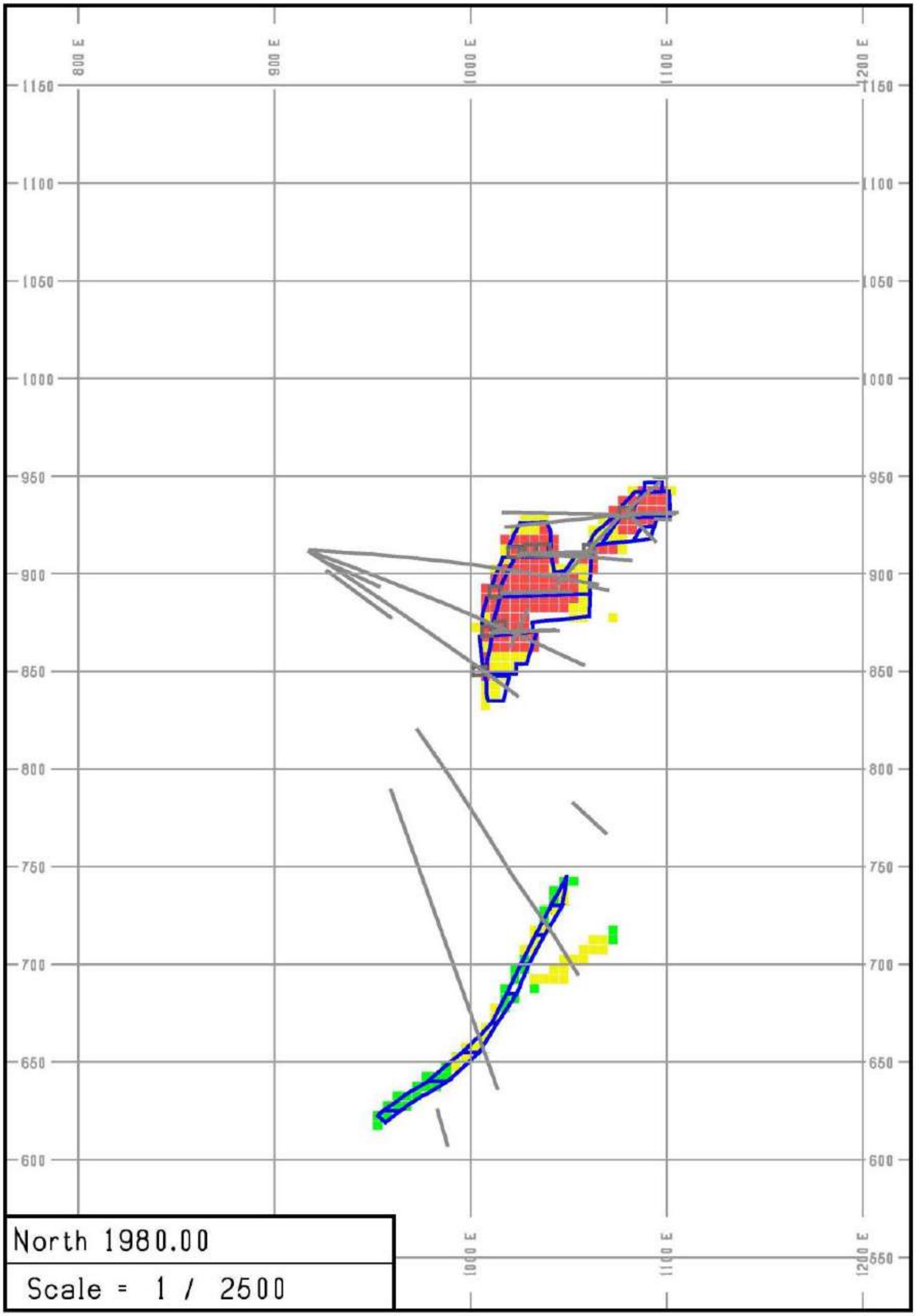


FIGURE 27-9 RESOURCE CLASSIFICATION – CROSS SECTION 2040N

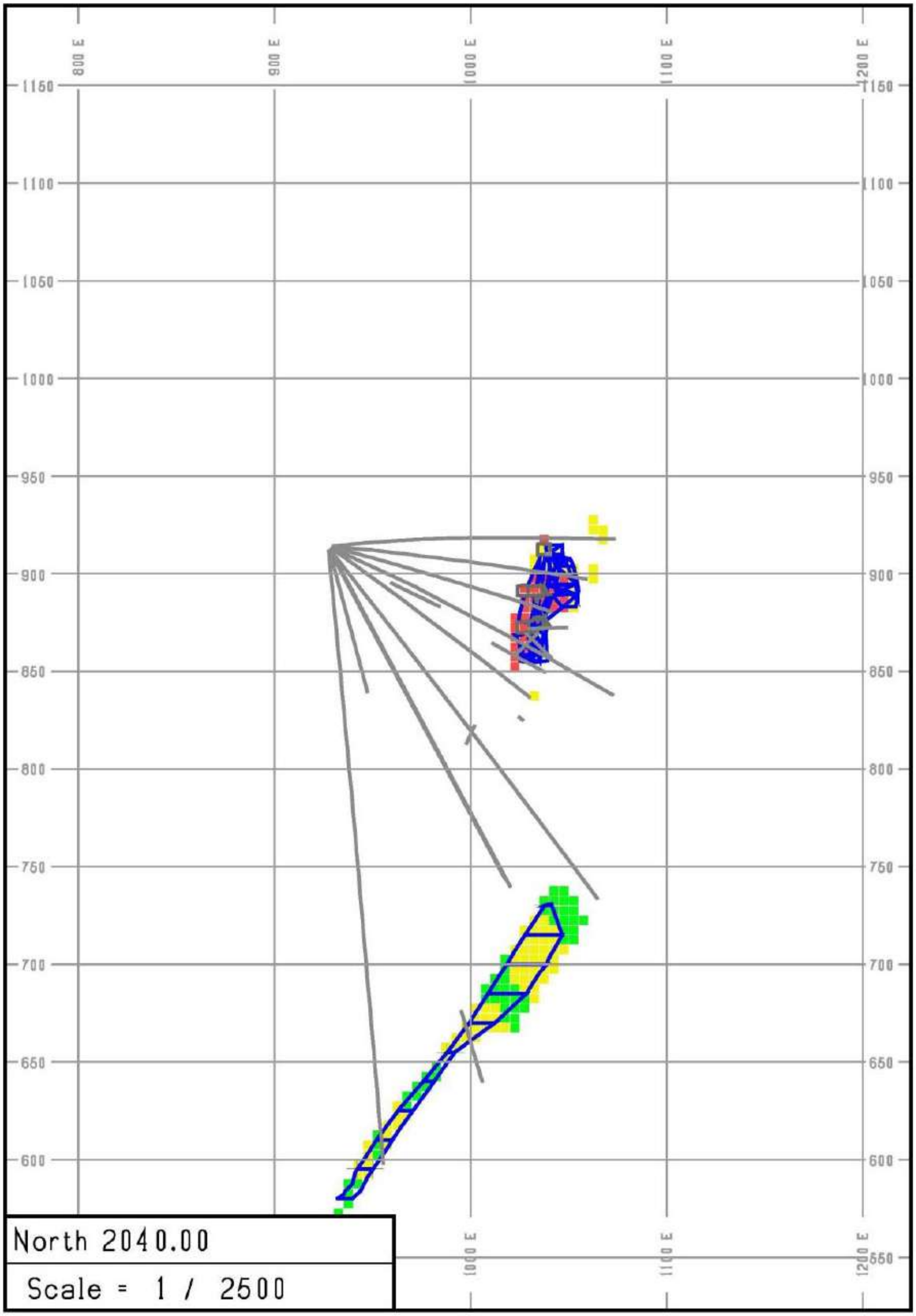


FIGURE 27-10 RESOURCE CLASSIFICATION – CROSS SECTION 2100N