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YAVA TECHNOLOGIES INC.

GEOLOGIC AND HYDROGEOLOGIC INVESTIGATIONS AT THE SILVER MINE PROPERTY CAPE BRETON COUNTY, NOVA SCOTIA

For the year ending 30 April 2006

Prepared by
Robert E. A. Boudreau, Ph.D., CD.

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EXECUTIVE SUMMARY

The current program involved the continuation of research into the geology and hydrogeology of the Yava Technologies Inc. (YTI) claims centered on Silver Mine, Cape Breton County, Nova Scotia. The geology and geochemistry of the Windsor Formation, and the hydrogeology of the aquifer within the Silver Mine Formation sandstones were investigated. The resulting preliminary modeling of the hydrogeologic properties of the involved aquifer, within the host sandstones, led to the design of a well-field for in-situ leach mining (ISLM). Further work involved investigating and compiling the geology and hydrogeology of the YTI property for the Jacques Whitford Environmental Ltd. report for environmental registration entitled "Environmental Registration for the Proposed Silver Mine Project". Finally, detailed mapping of the geology and hydrogeology at the YTI claim site, assay analysis of mudstone section of the core DDH 03-05 (partially reported in CETI 2004), permeability studies on mudstone from core DDH 03-05 and prospecting, analysis and mapping of the Windsor Formation were carried out during the fiscal year ending April 2006.

1. INTRODUCTION

The Yava lead deposit, located at the former settlement of Silver Mine approximately 40 km southwest of Sydney, Nova Scotia (Figure 1), consists of three low-grade areas of mineralization in Pennsylvanian-age sandstone, known as the east, central and west zones. An attempt was made in 1978 - 1981 to mine the highest grade west zone, however, it proved to be uneconomic using a traditional underground mining approach. The similarity of the Yava deposit to several low-grade uranium deposits located in sandstone formations in the United States prompted Centaur Mining Exploration Ltd (Centaur) to investigate the potential feasibility of employing a similar ***In Situ Leach Mining*** (ISLM) method for development of the Yava site.

With the ISLM technology, a leach solution is injected into the mineralized interval through a series of injection wells. The leach solution then flows through the porous sandstone toward a series of extraction wells. While in the subsurface, the leach solution dissolves the target metals and carries them in solution. The metal bearing leach solution is recovered with the extraction wells and is pipelined to the recovery plant, where the metals are stripped from the solution.

A 5-spot test well system was installed in 1992 at the Silver Mine Property, within the central zone, to examine the feasibility of leaching the lead mineralization. In the fall of 2003, the original 1992 test site was expanded by Yava Technologies Inc. (YTI) with the drilling of eleven additional test wells plus three shallower monitoring wells (CETI 2004). A hydrogeological investigation was initiated at the test site in December of 2003 to determine the hydrogeological parameters necessary for the design of a commercial-scale ISLM well field. The field program was suspended in January 2004 due to weather conditions.

The principal objective of the work program in 2005 was to continue the hydrogeological characterization of the sandstone in order to model and develop a working prototype well-field that could be tested during the demonstration phase.

2. LOCATION AND ACCESS

The YTI property is centered by the former settlement of Silver Mine (now a single deserted farm), in Cape Breton County, Nova Scotia as indicated in figure 1. Its geographical co-ordinates are 45° 51.6'N, 60° 24.3'W. It is approximately 42 km by road southwest of Sydney; and 38 km from the junction of Highways 125 and 327. The first 16 km of Hwy. 327, from the Sydney Bypass (Hwy. 125) to Marion Bridge, and the 12 km of Sandfield Road, from Marion Bridge to the Salmon River Road, are paved. The remaining 10 km along the Salmon River Road to Silver Mine is gravel.

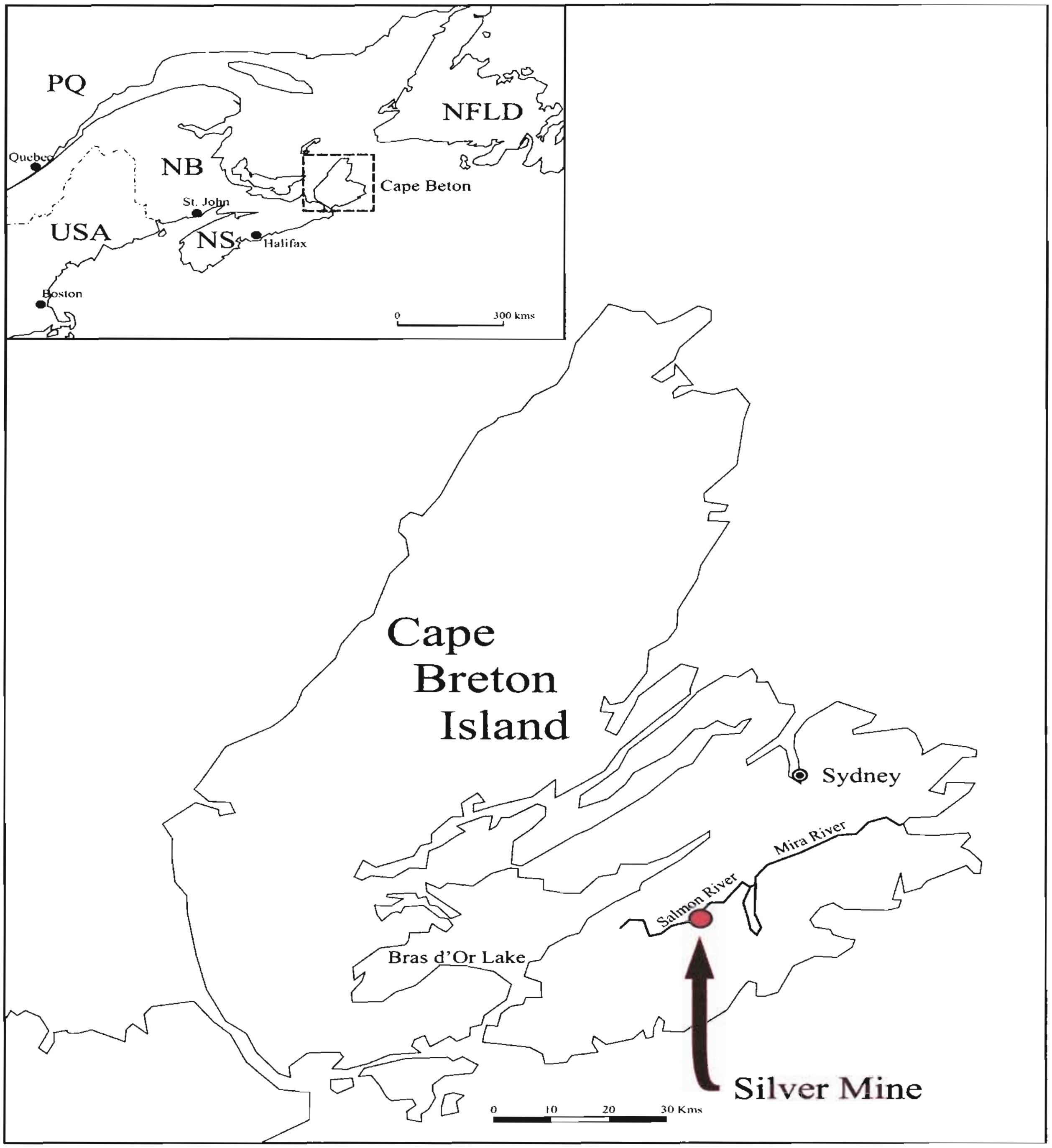

Figure 1 - The location of Silver Mine, Cape Breton County, Nova Scotia.

3. LICENCE

The licence numbers, tracts and claims held by YTI as of April 30th, 2006 are included in Table 1.

Licence	NTS Map	Tracts	Claims	Date of Issue
SL/103	11F/16B	88 ✓	ABCFGHJKLOPQ	16 July 2003
SL/103	11F/16B	89 ✓	EFKLMNOPQ	16 July 2003
SL/103	11F/16B	103 ✓	DEFGJKLMNOPQ	16 July 2003
SL/103	11F/16B	104 ✓	ABCDEFGHJKLMNOPQ	16 July 2003
SL/103	11F/16B	105 ✓	ABCH	16 July 2003
SL/103	11F/16C	7 ✓	ABCDEFGLM	16 July 2003
SL/103	11F/16C	8 ✓	ABCGHJ	16 July 2003

Table 1 – The YTI property licence numbers, tracts and claims as of April 30th, 2006 at Silver Mine, Cape Breton County, Nova Scotia.

4. HISTORY

The following history of development at the Yava deposit is extracted largely from a report by Geisler et al. (1993).

Small but rich showings of argentiferous galena in limestone outcrops have been reported near Silver Mine since the 1880s. During World War I, one tonne of 'galena-bearing granite' is said to have been excavated from a small exposure on a creek about 550 meters (1,800 feet) southeast of the road at Silver Mine and it is possible that the settlement derived its name from that time.

These showings and the presence of the Mindamar lead zinc mine at Stirling, 14 km to the south, led to geochemical investigations in the area during the late 1950s. Despite a number of significant anomalies and follow-up drilling, no important mineralization was found.

In 1961, the Yava deposit was discovered during an exploration program conducted by R.A. Geisler for Gunnar Gold Mines Ltd., Lehman Brothers, a New York investment bank, and E.F. Carr. The deposit was acquired and named Yava by Phelps Dodge Corporation of Canada Ltd. After an extensive drilling program, the property was sold to Barymin Exploration Limited in 1976. Barymin mined about 388,000 metric tonnes containing 4.75% Pb from the west zone during the 1978-1981 period (Vaillancourt and Sangster, 1984) and then abandoned the deposit due to poor economic conditions. The property then reverted to the crown.

The prevailing opinion was that although the deposit was of considerable size, it was too low grade to be mined economically. Sangster and Vaillancourt (1990, p. 214) reported that the estimated ore reserves (prior to Barymin's mining) were 3.5 million tonnes at 4.1% Pb at a 3% Pb cutoff. The property was held under closure by the Nova Scotia Department of Natural Resources, until 1989, when it was obtained by 656593 Ontario Limited for re-evaluation. In 1996, 656593 Ontario Limited changed its corporate name to Yava Technologies Inc (YTI), and YTI is currently conducting further feasibility studies to investigate the potential of ISLM for the deposit.

5. GEOLOGY

The geology of the area has been described in detail by Vaillancourt and Sangster (1984, 1986), and Sangster and Vaillancourt (1990). Other studies of the property have been reported by Patterson (1979, 1987), Boneham (1982), and John A. Leslie & Associates Limited (1992).

The Yava sandstone-lead deposit lies along the southeastern margin of the Salmon River Basin, which trends northeast-southwest. The basement at the Yava site consists of the Devonian-age Salmon River rhyolite porphyry (often referred to simply as rhyolite in the mine-related literature), which has intruded an older undivided series of Hadrynian to Ordovician-age rocks of volcanic and sedimentary origin (Boehner and Prime, 1985). According to Sangster and Vaillancourt (1990, p. 207), "the major rock units above the Salmon River porphyry in the Yava area are marine carbonates and siliciclastics of the Uist Formation, the shale-dominated transitional assemblage of the MacKeigan Lake Formation, and disconformably overlying continental sandstones of the Silver Mine Formation". The Uist Formation is part of the Windsor Group, while the

MacKeigan Lake Formation is part of the Canso Group (Boehner and Prime, 1985).

At Yava, the Pennsylvanian-age sandstone of the Silver Mine Formation is separated from the porphyry by the shales of the MacKeigan Lake Formation, however, the sandstone does rest directly on the rhyolite porphyry where the porphyry forms paleo-topographic ridges. The mineralization in the sandstone is enriched within small paleo-topographic valleys that reflect the rhyolite surface and has thus been subdivided into three zones, referred to as the east, central, and west zones. The galena mineralization forms part of the cement (along with carbonates) within the pore spaces of the sandstone and is considered to be post-depositional in nature.

6. RESEARCH AND DEVELOPMENT

In order to design a working well-field for ISLM it is necessary to understand the geology and hydrogeology of the host rock formation, including the ore zone. The interaction of the aquifer within the host rock formation must be modeled to account for the permeability of the host sandstones with respect to the interconnectivity of fracture and pore space in the ore body. Based on the geology (2004) and hydrogeology (2005) reported on in past years, a preliminary hydraulic model of the host sandstone was developed leading to the preliminary design of a well-field. To further define the model, work was continued in the vicinity of Silver Mine to further detail the geochemistry and hydrogeology of the ore zone, including the confining layers.

6.1 Geology and hydrogeology

To fulfill requirements with respect to the ***"Environmental registration for the proposed Silver Mine Project"*** being completed by Jacques Whitford Environmental Ltd., a summary of the geology and hydrogeology of the YTI property was completed based on information compiled to date, and is included as Appendix A to this report.

6.2 Modeling to design a well-field

The groundwater flow properties in the host sandstones at the YTI test site were modeled to develop parameters useful in designing a preliminary well-field for ISLM. Based on the hydraulic heads in the host sandstone, and the results of pumping, packer and tracer testing of past years, the YTI test site was modeled using the University of Waterloo Hydrogeologic Inc. software Visual ModFlow Pro. 4.0. The calibrated and verified model was used to design a preliminary 5-spot well-field containing four corner injection wells around a central recovery well. The corner wells were optimized to be 25 m apart, and this design could be used alone or in multiples in an operational well-field. The complete modeling and design report, including guard wells and monitoring wells, is included as Appendix B to this report.

6.3 Ore zone development

The ore zone at the YTI property was outlined using the available past exploration and drilling information in order to show the relationship of the ore zone to the host sandstone. The included maps and cross-sections are included as Appendix C to this report.

6.4 Assay and permeability values from DDH 03-05

The CETI report, for year end 2004, listed assay data for DDH 03-05 to the bottom of the ore zone at 100 m in depth. In order to determine the confining properties of the Windsor mudstones below this level the next 145 m of core was assayed by SGS Canada Inc. of Toronto Ontario for all minerals, and the results of this analysis are listed in Appendix D to this report. A brief description of the analyzed core is listed in Table 2.

To check the Windsor mudstones for properties useful as a barrier or aquiclude, a representative sample from 139 m in depth was tested by H. Q. Golder and Associates for permeability. Permeability results indicate that the hydraulic conductivity in the horizontal direction is 1.1×10^{-9} cm/s, and the hydraulic conductivity in the vertical direction is 5.7×10^{-10} cm/s, indicating that this mudstone will act as an aquiclude. The H. Q. Golder and Associate results are also included in Appendix D to this report.

Table 2 - Assay sample numbers and lithology from below the ore zone in DDH 03-05.

Assay Sample Numbers	Depth (meters)	Lithology
65751	101 - 102.6	red mudstone
65752	102.6 - 104.1	gray-green shale
65753	104.1 - 105.2	red mudstone
65754	105.2 - 106.5	red mudstone
65755	106.5 - 108.2	red mudstone
65756	108.2 - 109	green mudstone
65757	109 - 110.2	green/red mudstone
65758	110.2 - 111	red mudstone
65759	111 - 111.7	rhyolite conglomerate
65760	111.7 - 113	red/green mudstone
65761	113 - 114.5	green mudstone
65762	114.5 - 116	red mudstone
65763	116 - 117.6	red/green mudstone
65764	117.6 - 119	rhyolite conglomerate
65765	119 - 119.9	rhyolite conglomerate
65766	119.9 - 121	red mudstone
65767	121 - 121.8	red/green mudstone
65768	121.8 - 122.4	red/green mudstone
65769	122.4 - 123	red/green mudstone
65770	123 - 124.5	gray shale
65771	124.5 - 126	gray shale
65772	126 - 127.5	gray shale
65773	127.5 - 129	gray shale
65774	129 - 130.5	gray shale
65775	130.5 - 132	gray shale
65776	132 - 133.5	gray shale
65777	133.5 - 135	gray shale
65778	135 - 136.5	gray shale
65779	136.5 - 138	gray shale
65780	138 - 139.5	gray shale
65781	139.5 - 141	red mudstone
65782	141 - 142.5	red mudstone
65783	142.5 - 144	red mudstone
65784	144 - 145.6	red mudstone

6.5 Windsor Formation mudstones

During October 2005, exposed carbonate mudstones that outcrop along the Salmon River Road west of Silver Mine, were sampled for analysis to determine the mineralogy of the Windsor formation in the vicinity of the YTI property. The outcrop locations and mineral assay results of this project are included as a map and tables in Appendix E to this report.

7. CONCLUSIONS

The work conducted during the current program confirmed conclusions reached in previous studies and indicated the feasibility of ISLM at the YTI property. The aquifer model of the host sandstone, and initial well-field design, are a good first step in forming a practical working plan to carry out ISLM. To further refine the model, investigation of the hydraulic and chemical properties of the host sandstone must be undertaken to account for changes over the extent of the ore zone. This will enable the advanced design and operation of an efficient well-field for ISLM.

8. RECOMMENDATIONS

In order to develop an efficient ISLM operation, the well-field designed from the model, included as Appendix B to this report, should be developed and put into operation on the YTI property. Data obtained from this operation will provide detailed local aquifer and ore zone characteristics, enabling full control of well-field design and efficient removal of the target minerals. This ongoing process should be combined with hydrogeologic and geologic exploration of the extents of the aquifer and ore zone contained within the YTI claim.

9. REFERENCES

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Boneham, O.J.H. 1982. The Yava Lead Deposit, Salmon River, Cape Breton, Exploration (1962 -1981) and Mining Activities (1979 - 1981). Nova Scotia Department of Mines and Energy. January 15, 1982.

Canadian Council of Ministers on the Environment (CCME). 2003. Summary of Existing Canadian Environmental Quality Guidelines. Internet publication. http://www.ccme.ca/assets/pdf/el_062.pdf

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Freeze, R. A. And Cherry, J. A. 1979. Groundwater. Prentice-Hall Inc. New Jersey.

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Vaillancourt, P.D. and Sangster, D.F. 1986. Isotope and hydrocarbon studies of the Yava sandstone-lead deposit, Cape Breton Island, Nova Scotia: a progress report. in: Current Research, Part A, Geological Survey of Canada, Paper 86-1A, p. 133-140.

CURRICULUM VITAE

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PERSONAL DATA

Profile	I am a retired Canadian Naval Officer who is an Environmental Hydrogeologist employed by Yava Technologies Inc.

Research

2005 – Present	Modeling the geological and hydrogeological relationships between ore zones and host sandstone aquifers in order to design and develop operational well-fields for in-situ leach mining.
2004 – 2005	Analysis and development of the hydrogeologic properties of the sandstone aquifer within the YTI claim at Silver Mine, Cape Breton County, Nova Scotia.
1997 – 2004	Investigation of the sedimentary, hydro-geochemical and hydrogeologic nature of lacustrine environments, using micropaleontological, paleecological and paleolimnological indicators as analytic tools to indicate paleoclimate and its affect on the modern environment.

EDUCATION

2004 - 2005	**Post Doctorate**; Earth Sciences, Carleton University, Ottawa, Canada and Yava Technologies Inc., Toronto, Ontario **Research and Development**: Analyzed data and developed the hydrogeology of the Yava Technologies Inc. Claim site in Cape Breton County, Nova Scotia in order to develop a plan for in-situ leach mining of lead. ***Supervisor:*** Dr. Frederick A. Michel
2004	**Doctor of Philosophy**; Earth Sciences, Carleton University, Ottawa, Canada ***Dissertation***: Holocene Paleoenvironmental Reconstruction of James and Granite Lakes in the Temagami Region of Northeastern Ontario: from the retreat of the Laurentide Ice Sheet to the present ***Supervisors***: Dr. R. Timothy Patterson Dr. Frederick A. Michel

1999 **Master of Science**; Earth Sciences, Carleton University, Ottawa, Canada
Thesis: Foraminifera and Arcellaceans from non-marine environments in northern Lake Winnipegosis, Manitoba.
Supervisor: Dr. R. Timothy Patterson

1997 **Bachelor of Science**; Geology, Department of Earth Sciences, Carleton University

EMPLOYMENT

2005 – Present	Chief Geologist for Yava Technologies Inc.
1993 – 2005	Student and Teaching Assistant at Carleton University
1982 – 1992	Naval Officer in the Canadian Navy

PROFESSIONAL ORGANIZATIONS

2005 – Present	National Groundwater Association
2003 – Present	International Association of Hydrogeologists
2002 – 2004	American Geological Institute
1999 – Present	Cushman Foundation for Foraminiferal Research
1997 – Present	The Paleontological Society
1996 – 1999	Society for Sedimentary Geology

ACADEMIC RESEARCH

2004 – 2005 Acted as a co-advisor for undergraduate students with respect to their research projects

2004 – 2005
- Modeling hydrogeologic environments using Visual MODFLOW to determine the feasibility of mining metals using pH shift in a sandstone aquifer, incorporating data from pump, packer and leach tests
- Investigation of lake levels of glacial proximal lakes in the Temagami region of northeastern Ontario by coring
- Investigation of taphonomic change in limnological testate amoeba populations

1999 – 2004 *Research Assistant for Dr. Frederich A. Michel, Carleton University*
- Collecting lacustrine sediment core (Livingstone Corer) from northeastern Ontario
- Sampling chemical properties of groundwater (pH, temperature, electrical conductance, oxygen concentration, pore water sampling for chemical analysis)
- Using a Russian Auger to sample marsh sediment in northeastern Ontario
- Temperature profiling (Wheatstone Bridge) abandoned mine shafts in Timmins, Ontario to perform a feasibility study on the possibilities of CTES (Cavern Transfer Energy System) for heating and cooling
- Drilling and development of potable water wells in glaciofluvial sediment
- Quantitatively and qualitatively analyzed chemical and physical data from sediment and pore-water samples collected in northeastern Ontario
- Assembled and described information relating to the stratigraphy of sub-bottom lacustrine sediment in northeastern Ontario

1998 – 2004 *Research Assistant for Dr. R. Timothy Patterson, Carleton University*
- Coring Pink Lake PQ to determine the Holocene history of the Ottawa Region from the retreat of the Laurentide ice sheet to the present
- Collecting lacustrine sediment core (Livingstone Corer) from Southeastern New Brunswick and northeastern Ontario

- Vibracoring peat and marsh sediment from Chance Harbour, New Brunswick
- Using an Eckman Box Corer to collect sediment samples from the sediment-water interface
- analyzed sediment-water interface and core samples using a variety of quantitative and other micropaleontological techniques
- sub-bottom profiling of lakes and waterways using a Knudsen 320 B/P dual frequency echo sounder in northeastern Ontario and southeastern New Brunswick
- assisted with paleontological research at Joggins Nova Scotia

2002	*Field Assistant for Dr. J. Allan Donaldson, Carleton University* – balloon assisted aerial photography and mapping of an Ordovician stromatolite field, Gatineau, Quebec; Site protected, and work featured in local newspaper, radio, television news report and on a plaque at the site
1999	*Field Assistant for Dr. W. Brian McKillop, Manitoba Museum* - sampled groundwater and sediment for chemical and micropaleontological analysis in the vicinity of salt springs in Lake Winnipegosis, Manitoba
1999	*Independent Research* - investigated the feasibility of Pareto analysis to weigh variable importance using - quantitative micropaleontological data with Dr. James J. Cheetham (Biochemistry) and Dr. George Carmody (Biology)

AWARDS AND SCHOLARHIPS

1999 – 2004	Carleton University Graduate Student Scholarship
2000	Cushman Foundation for Foraminiferal Research; Award for Student Research

ACADEMIC TEACHING EXPERIENCE

2005 -	Instructor Instructed Environmental Geology to 36 students (ENSC 2001; Earth Resources and Natural Hazards: Environmental Impacts)
2004 – 2005	Tested two undergraduate students and one graduate student, as a member of their examining boards for their thesis defenses
2004 – 2005	Judged research posters of fourth year honors students (2 years)
2003 - 2004	*Instructor* Duties include teaching coring (Livingstone, Eckman and Russian corers), and sampling techniques for water and sediment analysis in lacustrine environments - 67.285 (currently GEOL 2805): Environmental Field Geology
2002 – 2003	*Laboratory Coordinator, Introduction to Geosciences (67.105), Carleton University* Responsible for managing up to eight laboratory sections with 16 teaching assistants and 120 students per academic term; duties include instructing teaching assistants, creating assignments and tests, acting as curator for teaching specimens, and assisting the professor.
2001 – 2003	*Instructor, Enrichment Mini-Course, Carleton University and Ottawa-Carleton District School Board* Co-developed and instructed a one week course (first week in May, each year): "Exploring the Earth Through Fossils"; on basic geology and paleontology for 20 high

school students aged 13 – 17; course consisted of lessons, invited speakers, lab exercises and field excursions

1996 – 2004 *Teaching Assistant, Department of Earth Sciences, Carleton University*
Duties include teaching labs, grading assignments and tests, leading field trips and holding regular office hours to assist the students

- 67.105: Introduction to Geosciences (no longer offered)
- 67.106 (Geol 1006): Exploring Planet Earth
- 67.285 (Geol 2805): Environmental Field Geology
- 67.224 (Geol 2204): Engineering Geology
- 67.325 (Geol 3205): Hydrogeology
- 67.487 (Geol 4807): Advanced Field Studies

COMMUNITY AND DEPARTMENTAL ACTIVITIES

Winter 2005 Investigated the feasibility of restoring a polluted pond for the Environmental Committee of Ottawa South; while co-supervising two undergraduate students

2000 – 2002 President, Geology Graduate Student Association, Department of Earth Sciences, Carleton University

1998 – 2000 External Director to the Council of the Graduate Student Association, Carleton University

PUBLICATIONS

Peer-Reviewed Papers

Boudreau, R. E. A., Galloway, J., Patterson, R. T. , Kumar, A. and Michel, F. A., 2005, A Paleolimnological record of Holocene climate and environmental change in the Temagami region, northeastern Ontario. Journal of Paleolimnology, v. 33, p. 445-461.

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Report Examination:

The report contains work conducted under the supervision of the author and by the author. The author was responsible for decisions concerning the test programs and collection of geologic samples. The author was on-site during October 2005, surveying a possible processing plant site and prospecting and sampling the Windsor Formation mudstones.

The author has not received any interest, direct or indirect, in the property of Yava Technologies Inc., nor does he own, directly or indirectly, any securities of YTI or any affiliate.

(Signature)

15 July 2006
(Date)

Appendix A

Appendix A

To

GEOLOGIC AND HYDROGEOLOGIC INVESTIGATIONS
AT THE
SILVER MINE PROPERTY
CAPE BRETON COUNTY, NOVA SCOTIA

For the year ending 30 April 2006

YAVA TECHNOLOGIES INC.

Geology and Hydrogeology of the Silver Mine Project Cape Breton County, Nova Scotia

Prepared by

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(22 February 2006)
Revisied April 2006

Prepared for
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EXECUTIVE SUMMARY

This report summarizes the current status of geologic and hydrogeologic investigations by Yava Technologies Inc.(YTI) on its property containing the Yava lead deposit at Silver Mine Nova Scotia. The mineralization of interest is within Pennsylvanian sandstones and Mississippian mudstones in the Salmon River Basin. The basin is a syncline plunging to the southwest, sitting unconformably on the granitoid basement, and bound by faults to the southwest and northwest. The hydrogeologic properties of the sandstone make it ideal for in-situ leach mining (ISLM) because it is confined above and below by mudstone, the whole layer dipping at 15° towards the northwest. Regional fluid flow within the porous sandstone is parallel to the axis of the syncline, towards the southwest.

A description of the proposed well-field is included, along with prevention and mitigation methods to provide environmental protection.

1 INTRODUCTION

1a. Purpose

The ultimate objective of YTI's investigations is to recover extensive deposits of several minerals that have been indicated on the property. To this end the report also describes the mineralogy and chemistry of the YTI deposit in order to commence ISLM of lead and other metals within the sandstones and carbonates. The geology summarizes previously available information [4][1],[5],[16], [21], [22], [26], [27],[30], but the hydrogeology was developed from current research [6], [15], [19], [20], The mineralogy combines previously available data with current research [18], [23],[24],[25],[28],[31],[32], as does the chemistry of the groundwater and overburden [2],[20].

1b. History of the Yava Silver Mine Project

The YTI deposit, a monomineralic occurrence of galena, was discovered by Robert A. Geisler in 1961[14]. Phelps Dodge Corporation of Canada Ltd. acquired the property from Talisman Mines Ltd. in 1962, then conducted an extensive exploration program until 1969 [5]. Phelps Dodge sold the property to Barymin Exploration Ltd. in 1976, and Barymin mined about 388,000 mt containing 4.75% Pb from the deposit between 1978 and 1981[5], [14]. Due to the low grade of Pb in this immense deposit, Barymin abandoned the property, and it reverted to the crown in 1981. The Nova Scotia Department of Natural Resources held the property under closure until 1989 when it was obtained by 656593 Ontario Limited for re-evaluation (corporate name now YTI) [14].

2. LOCATION

The YTI property straddles the deserted settlement of Silver Mine on the Salmon River road located about 42 km southwest of Sydney. It is adjacent to the Salmon River on the island of Cape Breton, Nova Scotia (Figure 1). The geographic coordinates of the property are 45° 51.6′ North latitude and 60° 24.3′ West longitude.

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[1] Numbers in square parentheses refer to items in the list of references at the end of this report

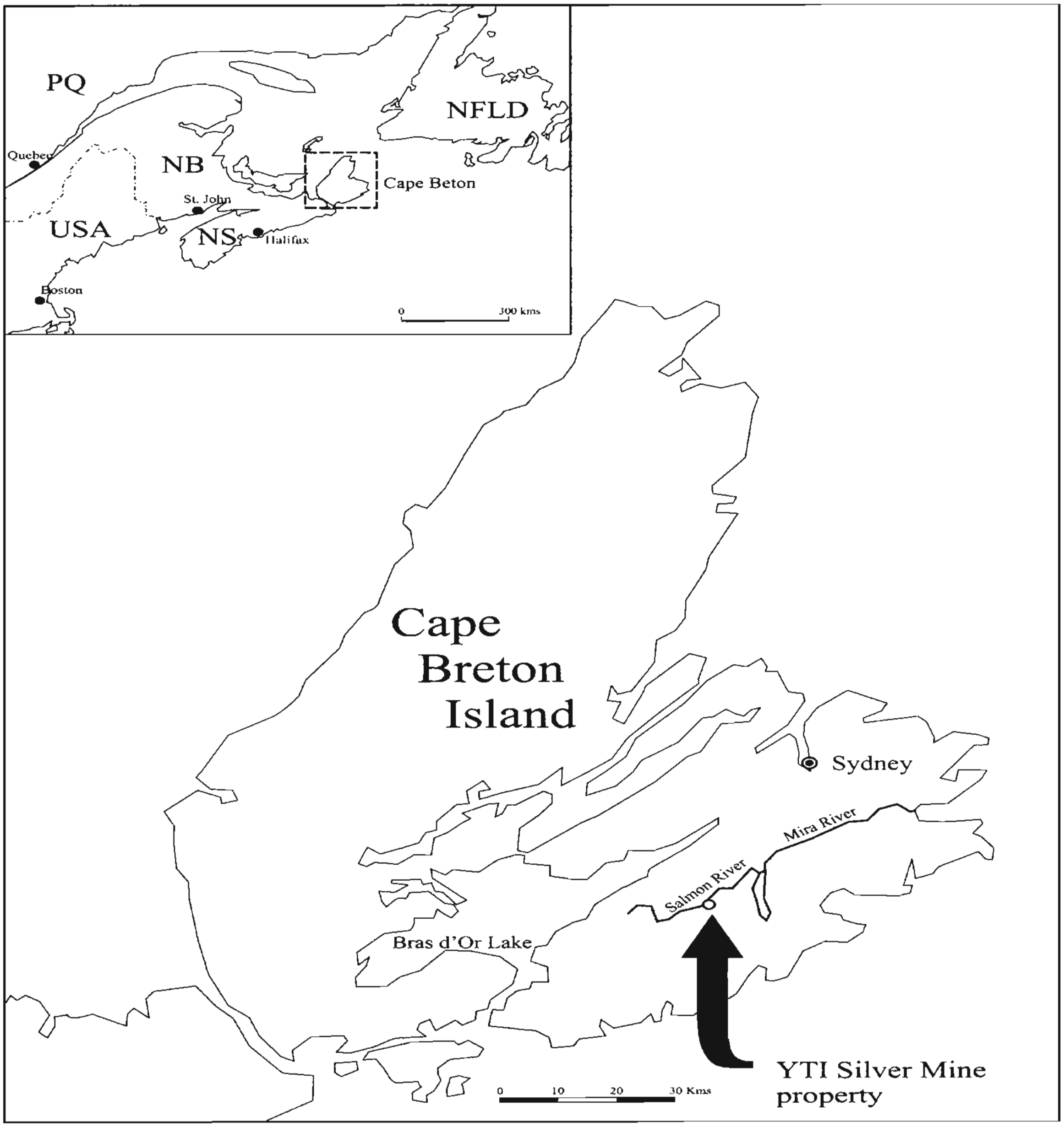
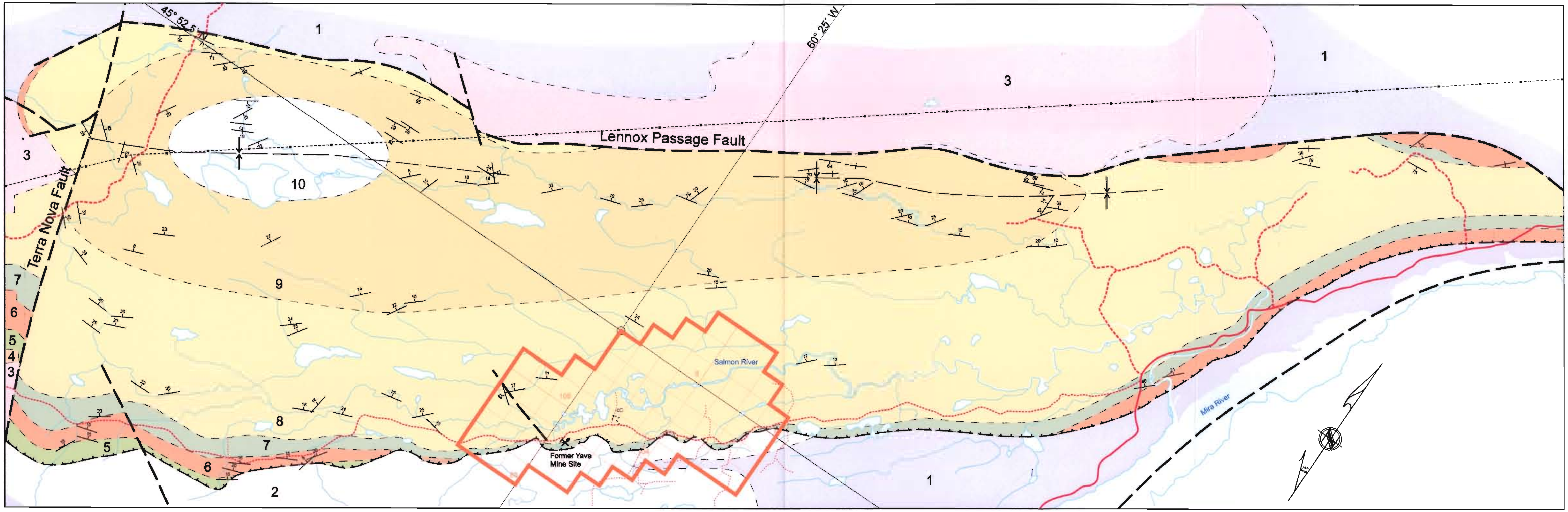


Figure 1 – Location map for the YTI Silver Mine property.

3. REGIONAL GEOLOGY

The YTI property lies within a northeasterly trending, triangular shaped syncline, which makes up the Carboniferous age Glengarry Half-Graben[26], or Salmon River Basin (Figure 2). This half-graben describes the Mississippian age Windsor Group consisting of conglomerate, sandstone, shale and limestone which lie unconformably above the Salmon River Porphyry rhyolites of Devonian age[18]. The Windsor Group is overlain by Pennsylvanian age Riversdale Group consisting of conglomerate, sandstone and shale.

Figure 2 – Map showing geology of the Salmon River basin.

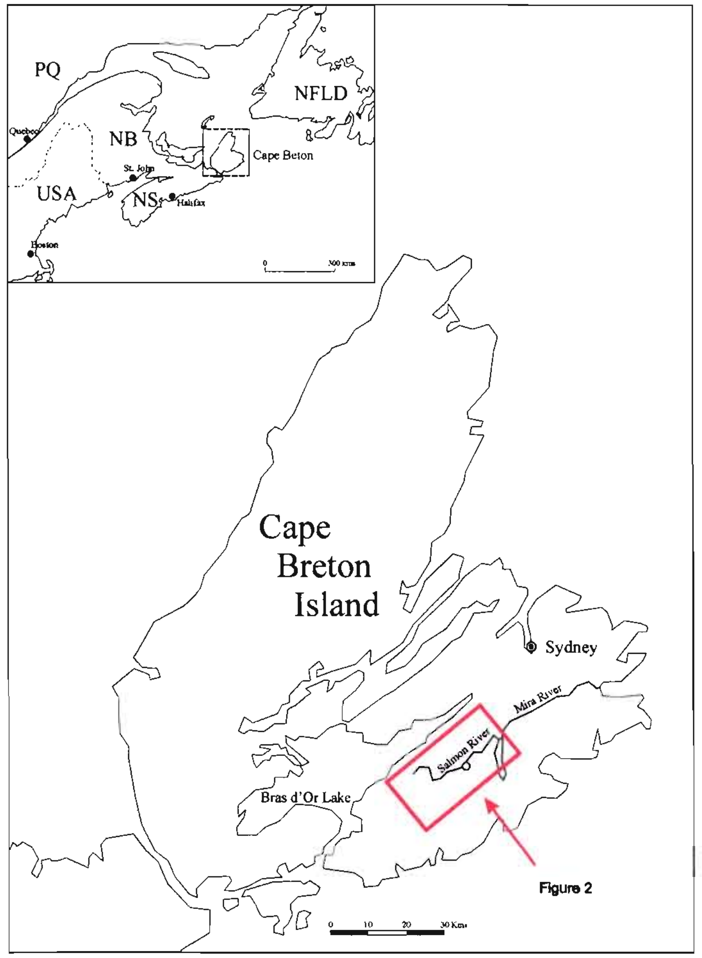


LEGEND

PENNSYLVANIAN	10	**GLENGARRY VALLEY FORMATION:** gray-green/red sandstone/siltstone/shale.
	9	**BIG BARREN FORMATION:** red conglomerate/sandstone and gray-green mudstone/sandstone.
	8	**SILVER MINE FORMATION:** gray-green/gray sandstone with coal; gray-green/gray mudstone.
MISSISSIPPIAN	7	**MacKEIGAN LAKE FORMATION:** gray/red shale, siltstone, gypsum/anhydrite.
	6	**UIST FORMATION:** Red siltstone, minor conglomerate intercalated with limestone/dolostone; minor gypsum/anhydrite; dolostone.
	5	**LOCH LOMOND FORMATION:** gypsum, anhydrite and limestone/dolostone; red siltstone and sandstone/conglomerates.
	4	**UNDIVIDED BASAL CONGLOMERATE**
DEVONIAN	3	**LOCH LOMOND, IRISH COVE and HUNTINGTON MOUNTAIN PLUTONS:** Granitoid rocks
Pre-SILURIAN	2	**SALMON RIVER RHYOLITE PORPHYRY:** Granitoid rocks; rhyolite intruding coarse grained sandstone/conglomerate.
	1	**HADRYNIAN to ORDOVICIAN ROCKS:** Volcanic and sedimantary rocks, variably deformed and metamorphosed.

SYMBOLS

Syncline
Bedding (inclined)
Geologic boundary (assumed)
Unconformity, nonconformity (approximate)
Fault (approximate)
Mine
Road
River, Lake
Power Lines
Yava Technologies Inc. claim boundary



Map 2006 - 1

YAVA TECHNOLOGIES INC.

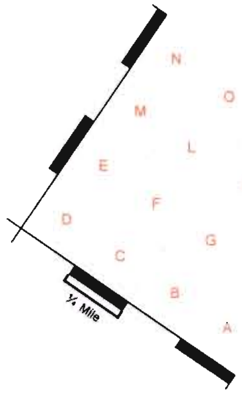
GEOLOGY OF THE GLENGARRY HALF GRABEN

SILVER MINE PROPERTY, CAPE BRETON COUNTY
NOVA SCOTIA

R. E. A. BOUDREAU

Scale 1:28,800
Miles 1 0 1 2 Miles
Kilometres 1 0 1 2 3 4 Kilometres

2006



Plan of all claims in tracts 7, 8, 9, 88, 89, 103, 104, 105 from Nova Scotia Claim Reference Maps 11F16B Victoria Bridge and 11F16C Mira, Cape Breton County showing the locations of the Yava Technologies Inc. claims (as of 03 May 2005).

Modified from Boehner and Prime, 1985, Loch Lomond Basin and Glengarry Half-Graben, Nova Scotia Department of Mines and Energy Map 85-2

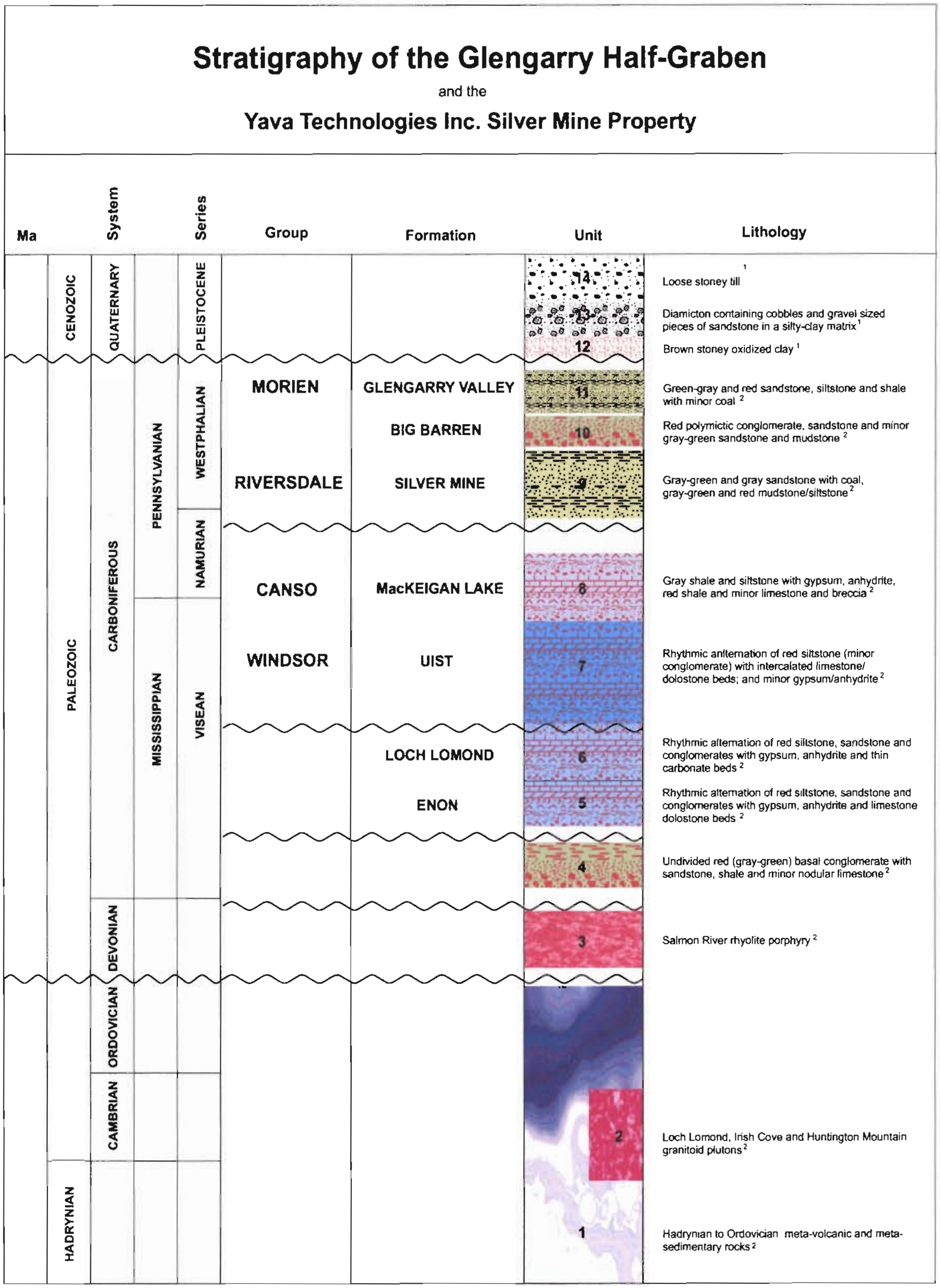
The sedimentary rocks of the Salmon River Basin are bounded on the northwest by the Lennox Passage fault, to the southwest by the Terra Nova fault and on the southeast by an angular unconformity with the Pre-Carboniferous basement [4],[18],[26]. These sedimentary rocks are asymmetrically folded into a northeasterly trending syncline whose southern limb dips 10 to 15 degrees to the northwest [18].

The lowermost rocks of the Windsor Group belong to the Uist Formation, a sequence of siliciclastics and marine carbonates (Figure 3). Lying conformably above the Uist Formation are the shales of the MacKeigan Lake Formation, part of the Canso Group [4],[18]. The Pennsylvanian aged sandstones of the Silver Mine Formation, part of the Riversdale Group lie disconformably above the Canso Group rocks. The Silver Mine Formation is transitional with the overlying conglomerates of the Big Barren Formation, part of the Morien Group [4],[18].

4. LOCAL GEOLOGY

The YTI property is located within the Fundy Basin system [30] that was formed as a result of regional subsidence occurring after the Acadian Orogeny, about 400 million years ago (ma), when a zone of steeply dipping northeasterly trending faults developed in the region [16],[30]. This system, consisting of a series of connected troughs and intermountain basins, has been interpreted as a rift valley with high angle faults on each side [3],[30]. The Fundy Basin is bounded to the south by the Meguma Platform, to the northwest by the New Brunswick Platform and to the northeast by the Newfoundland Platform [30] (Figure 4).

A transgressive marine incursion occurred during the Visean depositing a cyclic sequence of marine carbonates, evaporites and red siltstone. In the early Pennsylvanian, a regressive cycle occurred during which calcretes and a soil horizon was formed on the weathering Windsor shales. After this period of erosion, the upper part of the Fundy Basin platform was deposited, forming a continental molasses facies containing the



Stratigraphy of the Glengarry Half-Graben

and the

Yava Technologies Inc. Silver Mine Property

Ma		System		Series	Group	Formation	Unit	Lithology
	CENOZOIC	QUATERNARY		PLEISTOCENE			14	Loose stoney till [1]
							13	Diamicton containing cobbles and gravel sized pieces of sandstone in a silty-clay matrix [1]
							12	Brown stoney oxidized clay [1]
	PALEOZOIC	CARBONIFEROUS	PENNSYLVANIAN	WESTPHALIAN	MORIEN	GLENGARRY VALLEY	11	Green-gray and red sandstone, siltstone and shale with minor coal [2]
						BIG BARREN	10	Red polymictic conglomerate, sandstone and minor gray-green sandstone and mudstone [2]
					RIVERSDALE	SILVER MINE	9	Gray-green and gray sandstone with coal, gray-green and red mudstone/siltstone [2]
				NAMURIAN	CANSO	MacKEIGAN LAKE	8	Gray shale and siltstone with gypsum, anhydrite, red shale and minor limestone and breccia [2]
			MISSISSIPPIAN	VISEAN	WINDSOR	UIST	7	Rhythmic anltemation of red siltstone (minor conglomerate) with intercalated limestone/ dolostone beds; and minor gypsum/anhydrite [2]
						LOCH LOMOND	6	Rhythmic alternation of red siltstone, sandstone and conglomerates with gypsum, anhydrite and thin carbonate beds [2]
						ENON	5	Rhythmic alternation of red siltstone, sandstone and conglomerates with gypsum, anhydrite and limestone dolostone beds [2]
							4	Undivided red (gray-green) basal conglomerate with sandstone, shale and minor nodular limestone [2]
		DEVONIAN					3	Salmon River rhyolite porphyry [2]
		ORDOVICIAN						
		CAMBRIAN					2	Loch Lomond, Irish Cove and Huntington Mountain granitoid plutons [2]
	HADRYNIAN						1	Hadrynian to Ordovician meta-volcanic and meta-sedimentary rocks [2]

1. Michel FA. 2004. Geological and hydrogeological Investigations at the Silver Mine Property, Cape Breton County, Nova Scotia for Yava Technologies Inc.
2. Boehner, R. C. and Prime, G., 1985. Geology Loch Lomond Basin and Glengarry Half-Graben, Nova Scotia, Map 85-2, Nova Scotia Department of Mines and Energy

Figure 3 – Stratigraphic column of the Salmon River basin.

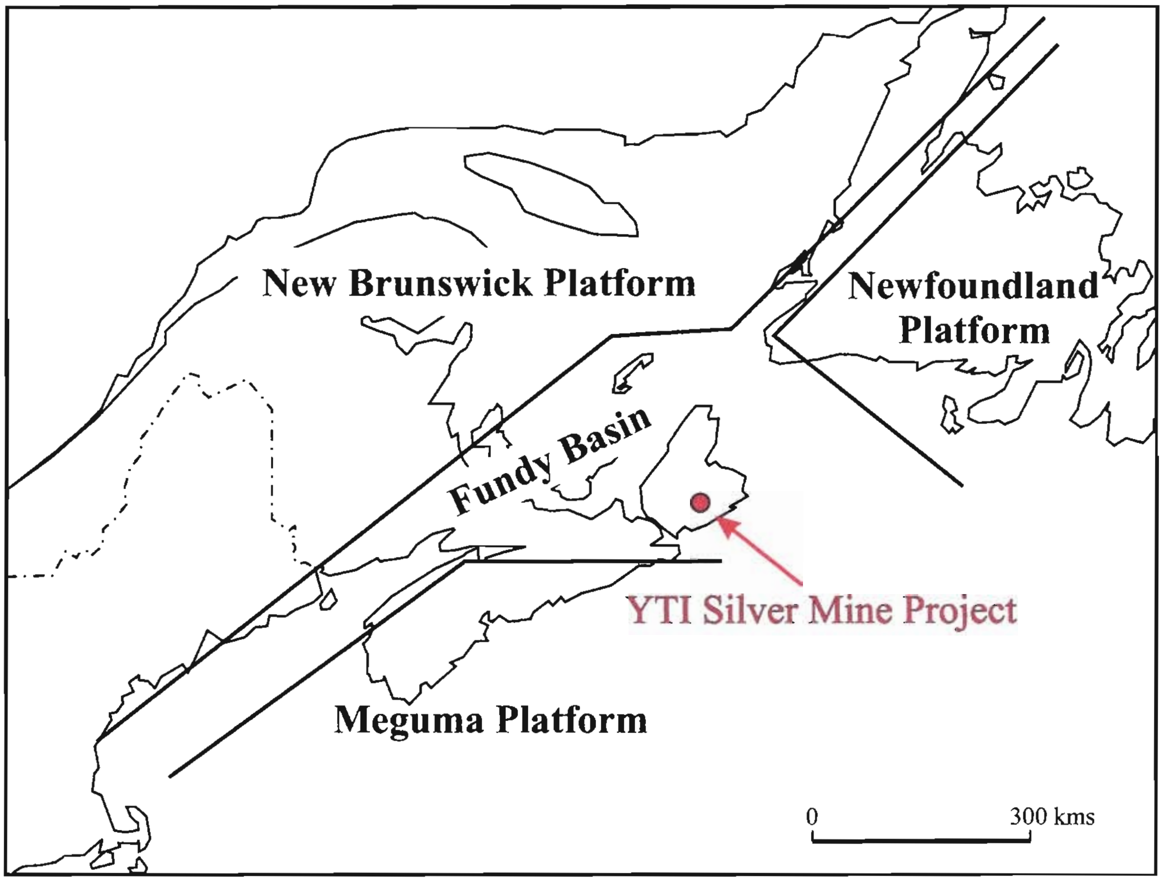


Figure 4 – Map showing the Fundy basin.

extensive coal deposits in the Pennsylvanian sandstones [5],[30]. These sandstones were deposited in a fluvial environment filling paleo-depressions within braided streams, eventually overstepping the Windsor Group with an accumulation of over 1500 meters of sediment [5].

5. GEOLOGY OF THE SILVER MINE PROPERTY

The reddish colored rhyolites of the Salmon River rhyolite porphyry make up the Devonian aged basement as indicated in figure 5. The rhyolites are composed of a fine-grained matrix of microcrystalline quartz and potassium feldspar, with phenocrysts of quartz, feldspar and minor muscovite [18],[27],[28].

The Mississippian Windsor Group lies unconformably above the Salmon River rhyolite porphyry within depressions in the rhyolite topography [18],[21],[33]. The Uist

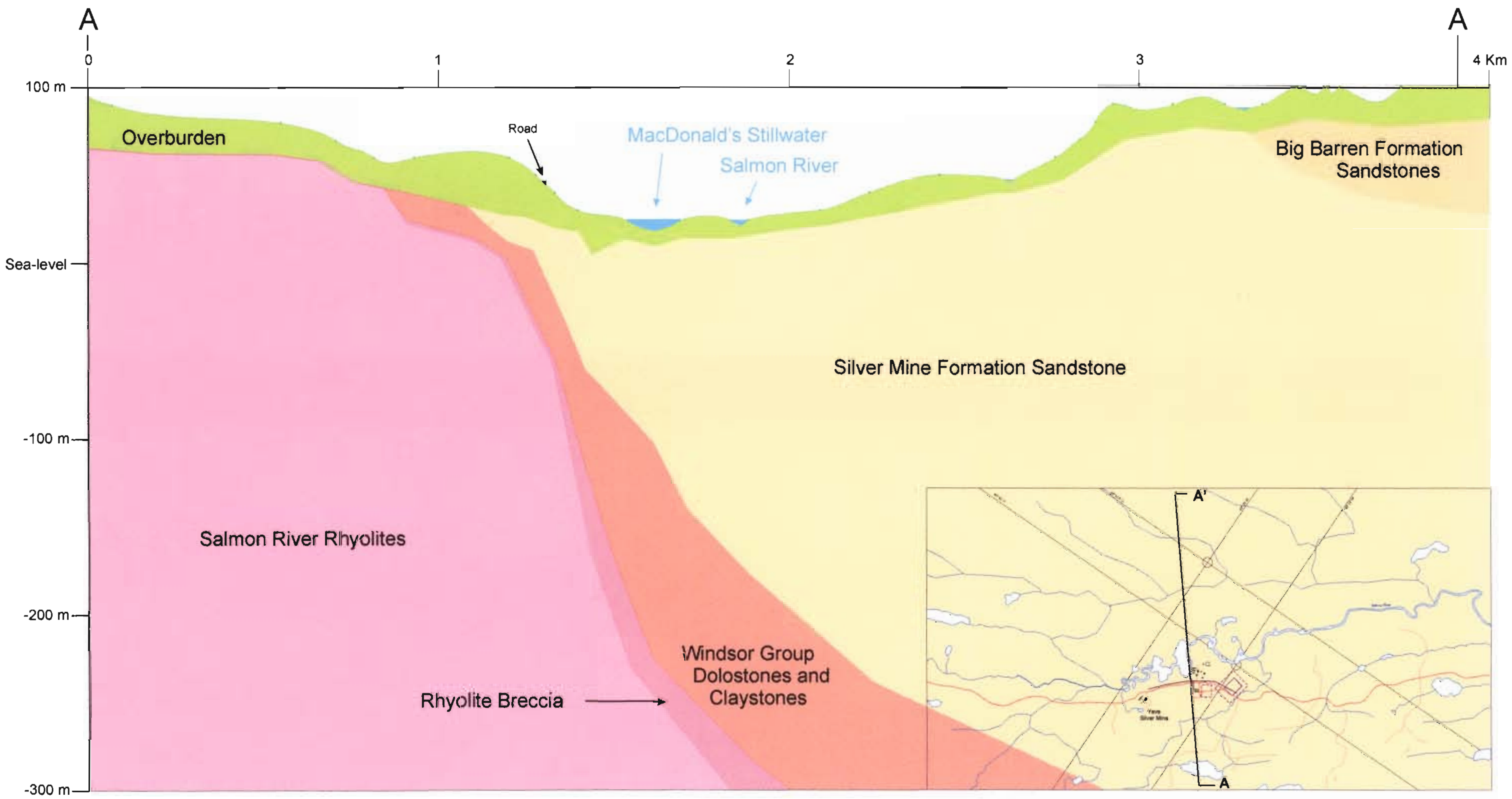


Figure 5 – Generalized cross-section through the Yava Technologies Inc. Silver Mine Property.

Formation consists of a lower unit of conglomerate and breccia, with rhyolitic clasts, occurring within a calcareous claystone matrix with local interbeds of shale and sandstone. The Big Glen Member, the upper unit of the Uist Formation is a slightly fossiliferous finely crystalline, massive dolostone [18].

The MacKeigan Lake Formation of the Canso Group lies conformably above the Uist Formation. The MacKeigan Lake Formation contains three sub-units, a basal red or green siltstone overlain by interbedded gray shale, limestone and gypsum. The middle unit, a transitional fine-grained red sandstone and mudstone with decreasing gypsum; and at the top of the formation is a caliche soil representing an unconformity [18](Leslie, 1992).

Above the Mississippian age Canso Group disconformably lies the Pennsylvanian age Riversdale Group. The lowermost Silver Mine Formation of the Riversdale Group consists of three units[18],[26]. Infilling paleovalleys and covering the girdling highs is a sequence of basal, medium to coarse-grained gray sandstone which grades upward into a green sandstone. This sandstone contains subhedral to anhedral grains of quartz mixed with minor amounts of microcline in a matrix of clay minerals and microcrystalline quartz.[18],[30]

The middle unit of the Silver Mine Formation is transitional from the lower medium-grained sandstone to a fine-grained, red to gray-green sandstone with increasing mudstone interbeds and minor coal [18].

At the YTI Silver Mine Property, the major rock suites are marine carbonates of the Uist Formation, overlain by the shale-dominated assemblages of the MacKeigan Lake Formation and the sandstones of the Silver Mine Formation [30].

7. MINERALIZATION

The YTI mineral deposits within the Salmon River Basin occur at or near the unconformable pinchouts that are present at several stratigraphic levels in contact with the Salmon River rhyolite porphyry. The lead mineralization occurs as disseminated galena in sandstone forming a zone about 2miles long by 1mile wide dipping 15° to the northwest varying from one ft to 100 feet thick. The zone is estimated to contain 52.6 million mt grading 1.84%Pb, 0.097% Zn, 0.18% Mn, 2.03% Ca, and 0.065 troy oz of Ag per mt. The mineralization forms three lobes with Pb grades locally reaching 10%, and between which, the mineralization is continuous albeit of lower grade. It overlies paleotopographic depressions in the underlying rhyolite basement and is separated from adjacent zones by intervening basement highs [30].

7. HYDROGEOLOGY

7a. Hydraulic Conductivity

Fluid flow through the Silver Mine Formation sandstone is controlled by the permeability of the sandstone and the properties of the fluid. Golder and Associates [15] determined that the hydraulic conductivity (K) parallel to the sandstone bedding ranged from 1.8 x 10^{-4} cm/s to 3.6 x 10^{-4} cm/s, and attributed the lower values to flow through the sandstone, while attributing the higher values to flow through fractures or joints. The sandstone has been noted to contain a number of fining upward sequences in grain size ranges from coarse sand to fine sand with occasional pebble conglomerates [27], and the pebble conglomerates could be the reason for higher hydraulic conductivity values. The hydraulic conductivity perpendicular to the sandstone bedding was found to range in value from 5.2 x 10^{-6} cm/s to 1.7 x 10^{-5} cm/s.

Packer testing was conducted during 2004 calculate hydraulic conductivity values for the area of the Yava Silver Mine test well field. Using the method outlined in Golder and Associates [15], values of 1 x 10^{-6} cm/s were found in fine-grained unfractured sandstone to 4 x 10^{-4} cm/s in coarse-grained sandstone [20]. Hydraulic conductivity in fracture zones was found to be as high as 3 x 10^{-2} cm/s.

In order to describe the subsurface lithology and identify fractures in the test well field, [20] used a borehole TV camera system. This examination revealed that the fracture density varied between wells within fining upward sequences; and that variations within the hydraulic conductivities were likely due to a combination of open fractures and coarse grained permeable sandstone horizons.

7b. Water Levels

Static water levels vary from 9 to 12.5 meters below the tops of the well casings and represent the hydraulic head within the sandstone aquifer. It was determined that annual variation in any of the wells tested was about 0.40 meters [20].

7c. Hydraulic Parameters of the sandstone

The hydraulic parameters of the sandstone aquifer, transmissivity (T) and storativity (S) were calculated using methods outlined by Theis [29] and Jacob [17] for confined aquifers from data collected in 2003 and 2004 from pumping tests performed at the Yava Silver Mine test well site [19],[20]. The values obtained reflect large-scale averages for the sandstone aquifer. Values ranging from 12 to 32 m^2/day were obtained for transmissivity, which indicates the amount of water that can be transmitted horizontally through a unit width of the saturated thickness of the aquifer with a hydraulic gradient of one [7].

Storativity which indicates the volume of water that a permeable unit will absorb or expel from storage per unit surface area per unit change in hydraulic head was found to range from 5×10^{-3} to 5×10^{-5} (storativity is a dimensionless quantity) [20] typifying a fully confined aquifer. The sandstone aquifer is confined above by a clay unit in the overburden and below by the mudstones of the Windsor Group.

7d. Drawdown

Drawdown in monitoring wells is a response to flow through the homogeneous porous sandstone and is controlled by the distance between the pumping and monitoring wells, and by the hydrogeological parameters of the aquifer. Contoured drawdown diagrams were plotted to determine the orientation of flow during pumping tests, and it was determined from the resulting ellipsoid that the preferential flow was from the northeast to the southwest. Also, because the ellipse is not completely flattened, it indicates that small-scale fracturing or jointing is providing the secondary permeability of the sandstone [20].

7e. Travel time through the sandstone

Travel time of fluid from injection wells to pumping wells was determined by conducting tracer testing in 1992 [13] and 2004 [19]. Transmission speeds from 0.1 cm/s to 1.4 cm/s were reported indicating continuous permeability across the well site.

7f. Hydrogeologic Conclusions

The sandstone unit containing the mineralization is confined above by the clay rich overburden and below by the mudstone and rhyolite. These units, along with the several thin mudstone layers within the sandstone restrict the flow of water to the bedding plane. It was found that the flow through the sandstone was through areas of coarse-grained sandstone and zones of small scale fractures and that this flow approached 1 cm/s in velocity from the northeast to the southwest, parallel to the axis of the Salmon River Basin. This indicates that the method of ISLM is viable and that the well-field can be restricted to the confined sandstone unit. There is no obvious fluid transfer between the overburden and the sandstone units which act as separate aquifers.

8. WATER AND SEDIMENT CHEMISTRY

Surface and groundwater samples collected during 2003 by Jacques Whitford Environment Limited and F. A. Michel [2],[20] indicated levels of ammonia, aluminum, cadmium, copper, iron, lead and zinc exceeded the CCME Water Quality Guidelines for the Protection of Freshwater Aquatic Life [(CCME), 2003 #5]. The levels of iron and copper exceeded the above guidelines in the Salmon River. Lead concentrations as high as 27 μg/L were found in the surface water, and concentrations of lead as high as 18 μg/L were found in the groundwater. Manganese concentrations of 380 μg/L exceeded the groundwater aesthetic guideline of 50 μg/L.

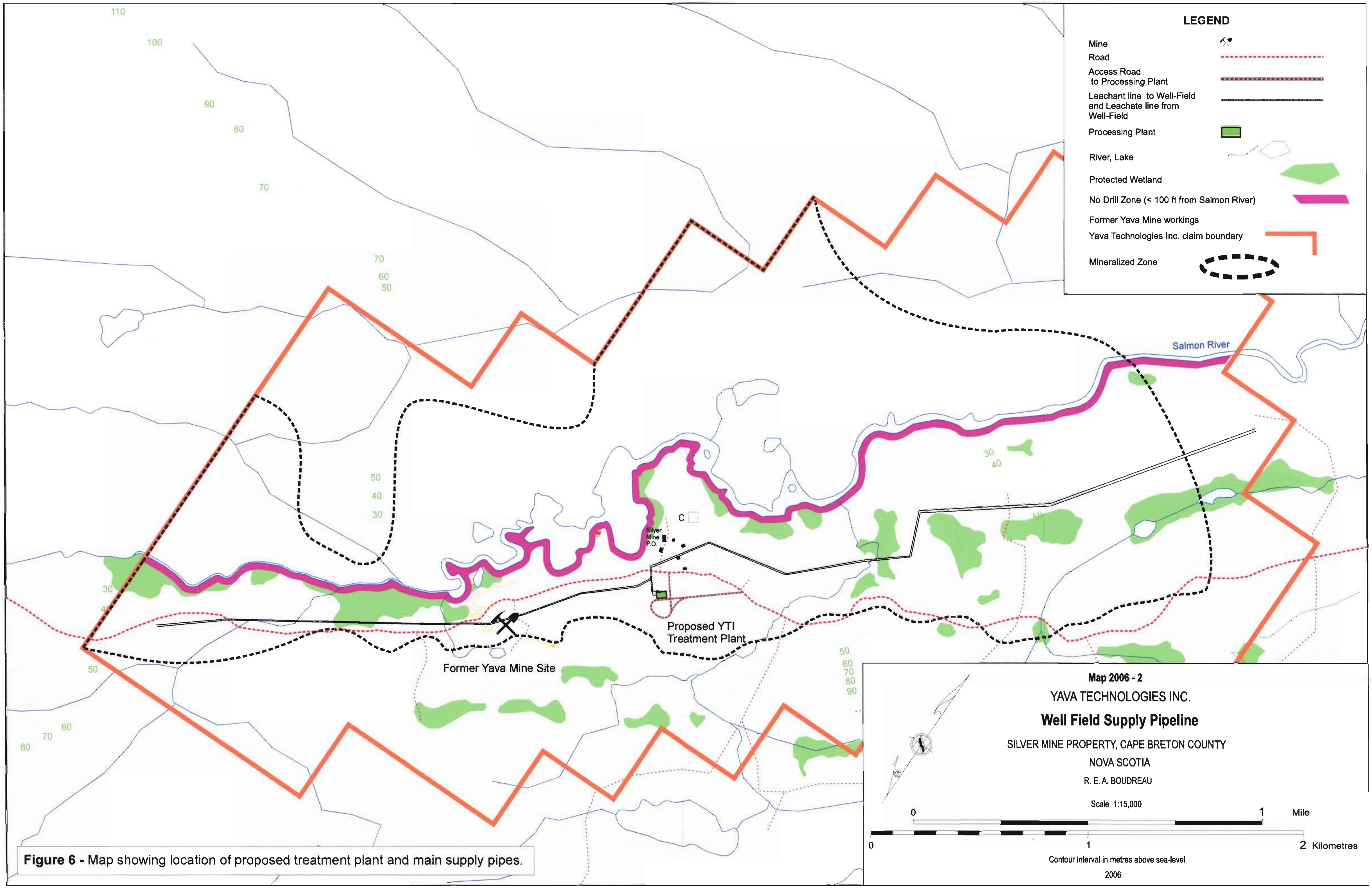


Figure 6 - Map showing location of proposed treatment plant and main supply pipes.

Geochemical surveys conducted on the soil in the vicinity of the Yava Technologies Inc. Silver Mine property by R. A. Geisler during 1962 indicated high anomalies of lead were present with values over 10,000 ppm (parts per million) on the property just south of the Salmon River Road. [8],[9],[10],[11],[12].

9. THE WELLFIELD

The wellfield extracts the ore by means of ISLM. The method consists of a series of wells to inject a suitable leachant into the ore deposit to dissolve the target minerals into a leachate which is pumped through a separate pipe to the treatment plant for recovery as value added products ready for industrial use.

The wellfield will generally consists of six adjoining five spot well systems forming a rectangular block surrounded by air injection wells forming an air curtain to help prevent excursion which in turn are surrounded by an outer tier of wells with sensors to detect such excursion. Each five spot well system consists of a 25m square with an injection well at each corner and a recovery well in the center. Wells are six inches in diameter, vertical to the bottom of the zone to be leached.

The injection and recovery wells are serviced from two main supply pipes respectively conducting leachant from or leachate to the plant shown on Figure 6. They are of identical design consisting of an inner 8in diameter pipe to conduct solution and an outer 12in pipe to contain possible leakage from the inner pipe. Both are constructed of high density polyethylene. The inner pipe contains a series of valves that may be closed on command to sequester fluid in the pipe in the event of rupture. The outer pipe has a sensor along its length to detect any sign of leakage from the inner pipe and thence close the valves to terminate fluid flow. A cross section of the pipe is depicted in Figure 7

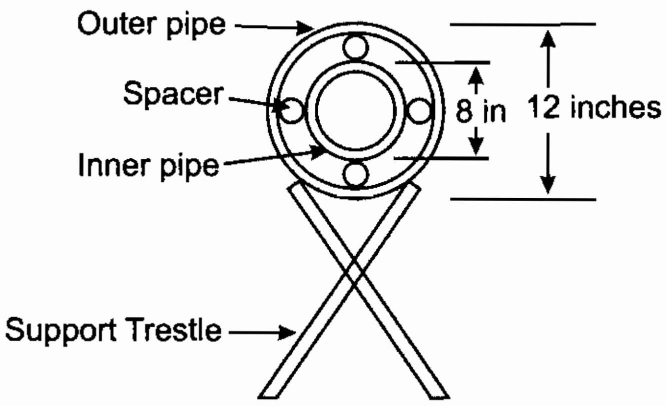


Figure 7 – Cross-section of main supply pipe.

Both pipes are supported across the property on wooden trestles about two ft high and their paths marked by coloured flagging. The pipes cross beneath the Salmon River road in a 4ft diameter reinforced concrete tunnel depicted in Figure 8. The leachate recovery pipe is wound with a layer of polyethylene sheet to capture discharge in the highly unlikely event of inner and outer pipes rupturing simultaneously. Openings along the sheet direct such discharge to purpose built ground sumps for bioremediation cleanup.

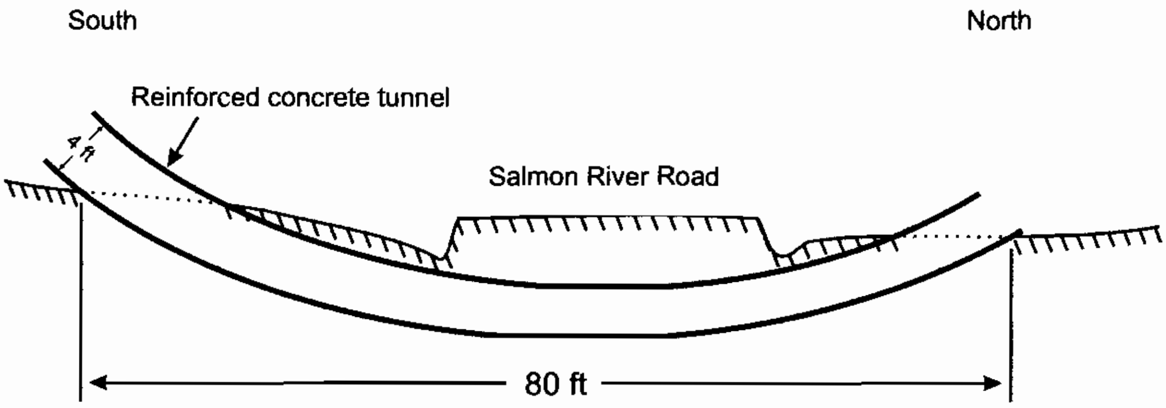


Figure 8 – Cross-section of Salmon River Road showing pipe transfer tunnel.

Wellfields are serviced from the main supply pipes by means of temporary branches. When the area to be mined is depleted the wellfields are moved to adjoining mineralized sectors and re-connected by new branches to the main supply pipes which can be extended if necessary. The entire wellfield activity ie solution flow, sensor information, valve control leachate principal mineral content plus wellfield / pipeline camera surveillance and plant operation are continuously depicted in real time to the process operators on 24hr duty continuously at the control center in the plant.

10. ENVIRONMENTAL IMPACT

10a. Measures for Prevention and Mitigation of Potential Environmental Impact

YTI believes that avoidance is the most desirable way of mitigating environmental impact because if impact is avoided so are the mitigation expenses. This could be achieved through modifications of plants, designs or schedules.

Consideration will be given to means of mitigating and avoiding undesirable impacts. In addition, a monitoring system will be specified to ensure that the measures work. It may require changes in planning, design and engineering and project management. Monitoring and follow-up are also important to verify the accuracy of impact predictions, establish what impacts actually occur and to modify the mitigation measures to improve their effectiveness.

10b. Spills and Accidental Releases

Plant design and construction will concentrate on spillage control by flow cut-off and isolation procedures as well as equipment. A quality assurance/quality control system will be applied, involving pre-operational testing of equipment, periodical testing and regular inspection of equipment, especially pipelines, and interlock monitoring on line flows and pressures with automatic shutdown in response to flow, pressure and conductivity changes. Consequently, spills, if any, will be extremely small. Bulk chemical, fuel and lubricant holding areas would be designed and constructed in accordance with current regulations, including requirements for dike and lining.

10c. Liquid Leak Detectors

Liquid leak detectors sense when a liquid is discharging from a pipe, tank, or other receptacle area. With most liquid leak detectors or fluid monitoring systems, an alarm or indicator alerts the user based upon the accumulation of liquid, the pressure difference, or change in flow. These sensors can detect the spillage as low as three drops of a chemical, reducing the extent and magnitude of the spillage to virtually non-existent level.

Electrical conductivity or resistance is generated when the liquid contacting both probes completes a conductive circuit in most cases the tank or vessel is included in this circuit. Liquid leak detectors use a low-voltage power source applied across separate electrodes. An electrochemical sensor or a selective chemical sensor may measure the presence or concentration of liquid chemicals. The output is presented as an amperometric, conductometric or potentiometric electrical signal. The signal is often relayed to a process controller. This signal may be relayed for automatic shut off.

10d. Excursion

During ISLM operations, the volume of extracted solutions is arranged to be 0.5 – 1.0% higher than that injected, in order to create a hydraulic curtain by providing a net inflow of clean groundwater from the aquifer to the well field. thereby helping to minimize the likelihood of well field excursions by leachate.

10e. Piping

Two concentric double-walled pipes will be used to insure that accidental discharge will never come in contact with the surface soil if the inner pipe breaks. To accommodate catastrophic disaster e.g., the entire double walled pipe breaks, a collector plastic membrane will be wound around the entire length of the double-walled pipe.. The membrane will feed discharge to purpose built sumps specially constructed to sequester solution from soil and enable appropriate remediation treatment..

It is important to note that YTI's proprietary lixiviant is a mixture of hydrogen peroxide and acetic acid. The concentration of acetic acid used by YTI is equivalent to vinegar consumed as food. Moreover, acetic acid is biodegradable and the hydrogen peroxide is a light sensitive chemical. On exposure to the surroundings, it immediately converts into water. The preventative measures employed in conjunction with the environmentally friendly YTI leachant will have virtually no known negative impact on the environment.

10f. Socio-Economic Impact

No permanent residents live within a 5 km radius of the YTI mine site. Since there is no substantial ground disturbance and no waste dumps associated with YTI's ISLM process, there is no possibility of disruption to the current local lifestyles. On the other hand, a tremendous positive impacts will be enjoyed by the community due to the increased in employment opportunities in an otherwise depressed area. YTI will establish a relationship with the community to create a consultative process and community involvement. This process will provide a forum for discussing the community's needs, feelings and attitudes and will be utilized to optimize social and economic benefits to both the community and YTI. A map indicating the watershed in the area of the Silver Mine Site is included as figure 9.

11. MICROBIOLOGY

Water samples collected during the summer of 2004 for bacteriological analysis indicate that sulfate reducing bacteria (SRB) occur naturally within the sandstones of the Silver Mine Formation. Dr. W. Douglas Gould of CANMET who conducted the analysis indicated that if acetate were present, such as would occur in an accidental spill, then the number of bacteria would rapidly increase. In the presence of sulfate (which is present in the soil), the SRB would consume the acetate, reducing the sulfate ions to create H_2S and thereby re-precipitate lead as lead sulfide [20].

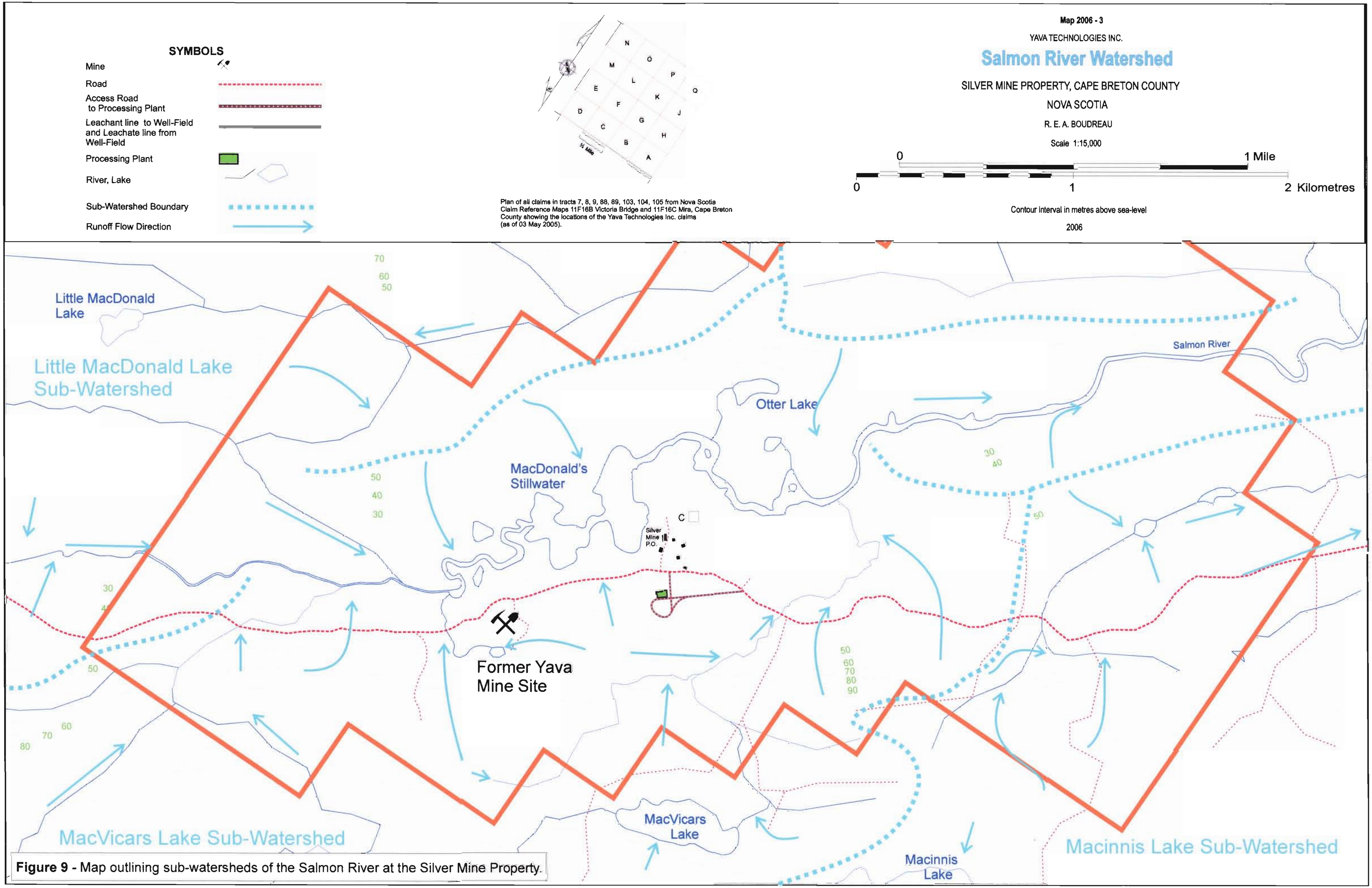


Figure 9 - Map outlining sub-watersheds of the Salmon River at the Silver Mine Property.

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Appendix B

Appendix B

To

GEOLOGIC AND HYDROGEOLOGIC INVESTIGATIONS
AT THE
SILVER MINE PROPERTY
CAPE BRETON COUNTY, NOVA SCOTIA

For the year ending 30 April 2006

MODELING AND DESIGN OF A WELL-FIELD FOR IN-SITU LEACH MINING OF THE SILVER MINE FORMATION SANDSTONES

Silver Mine Property
Cape Breton, Nova Scotia

Prepared by
Robert E. A. Boudreau, Ph.D., CD.
Environmental Site Investigations

30 March 2005

Prepared for
Robert A. Geisler
Yava Technologies Inc.

ABSTRACT

This report outlines the modeling conducted using the Waterloo Hydrogeologic Inc. software Visual ModFlow Pro v.4.0 to design a well field for in-situ leach mining at the Yava Silver Mine Property in Cape Breton, Nova Scotia. The model was based on pumping tests, packer tests and tracer tests that were recorded during the summer and fall of 2004. The hydrogeologic characteristics of the Silver Mine Formation sandstones were modeled and this model was calibrated, then verified to seven days using drawdown data from the 2004 pumping tests. The radius of influence of the pumping wells in conjunction with the verified model determined that 25 meters was the optimal distance to space wells apart in this sandstone to conduct in-situ leaching of metals. This well field contains a base cell consisting of four injection wells placed, each on the corner of a square, 25 meters apart. This cell is centered by a pumping well. This cell can be used alone, or in multiples, but the operating well field must be ringed with containment wells to prevent escape of the leachate. This design also includes monitoring wells which must be placed outside the containment wells; and also within the overburden and the underlying strata.

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LIST OF ABBREVIATIONS

%	percent
1/m	per meter
3-D	three dimensional
CHB	Constant Head Boundary
cm/s	centimeter per second
GHB	General Head Boundary
gpm	gallons per minute
K	hydraulic conductivity
L/min	liters per minute
m	meters
m^2/s	square meters per second
mm/yr	millimeters per year
S_s	specific storage
S_y	specific yield
VMP4	Visual ModFlow Pro v. 4.0
WB	Wall Boundary
WHI	Waterloo Hydrogeologic Inc.

EXECUTIVE SUMMARY

Modeling the hydrogeologic characteristics of the Silver Mine Formation sandstone was undertaken to design a well field that will be used to form a practical working plan to carry out in-situ leaching at the Yava Silver Mine site. The objectives were met and a working design of a well field was produced.

The model was calibrated in Visual ModFlow Pro v.4.0 (Waterloo Hydrogeologic, Inc.) and verified by running the model using pumping test data from June 2004 at the Yava Test Site. Maximum drawdown in monitoring wells and the zones of influence of the pumping wells were used to design the optimal distance between wells. The distance between wells was set at 90% of the monitored maximum drawdown to ensure complete coverage of the ore zone. The northeast trending hydraulic conductivity zones that resulted from the initial model were then used to test potential well field cells.

Three different well separation distances were investigated; 20m, 25m and 30m. The optimal distance between wells was determined to be 25 meters. This was investigated using a six 5-spots in a well-field where one 5-spot consisted of one pumping well centered between four injection wells, forming a square, each of which were 25 meters apart. An outline of a proposed well field is indicated in Figure 1EX. This well field design must include containment wells surrounding the injection and pumping cells within the ore zone; and monitoring wells both surrounding the containment wells in the ore zone, and above and below the ore zone.

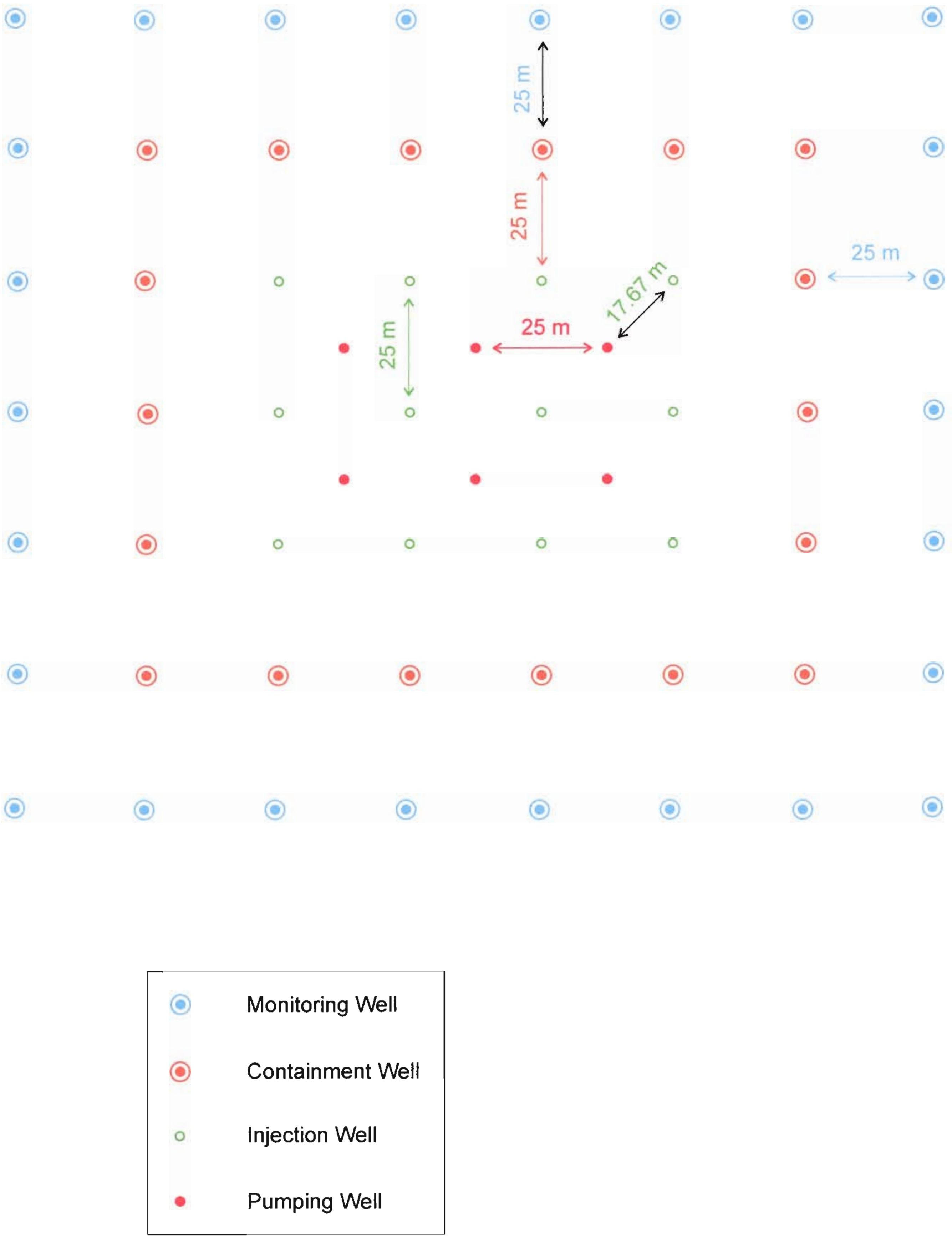


Figure 1EX - Well-field design as tested using Visual ModFlow Pro v. 4.0.

1. INTRODUCTION

This report examines the results of modeling performed using data collected from pumping, tracer and packer tests that were conducted during the year 2004 in the Silver Mine Formation sandstones at the Yava Technologies Inc. (YTI) Silver Mine Property (Fig. 1).

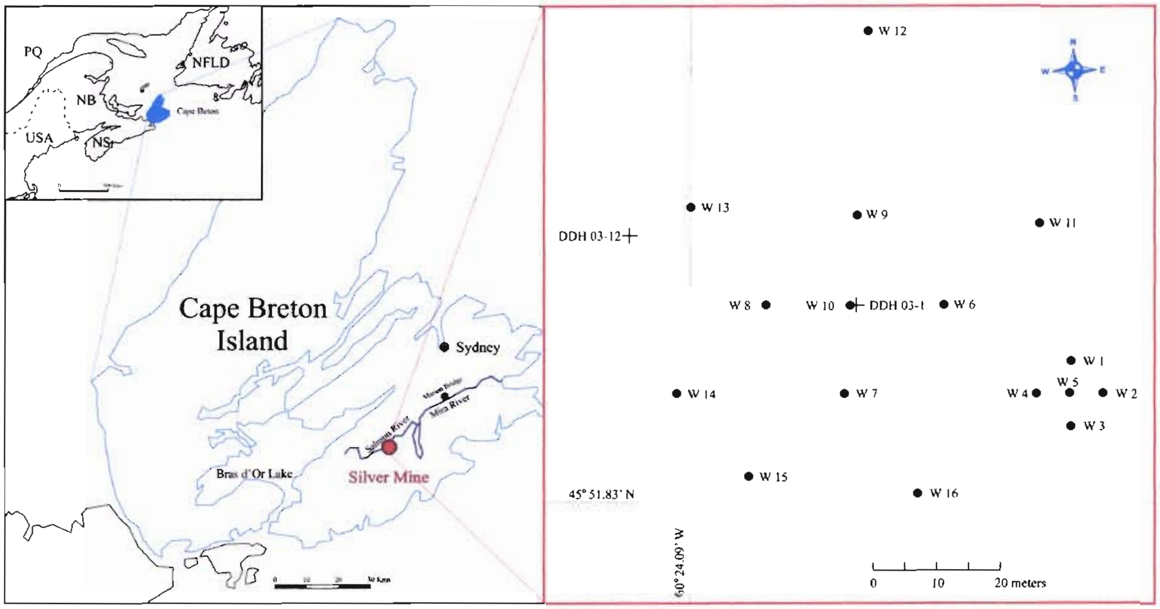


Figure 1 - Map showing the location of the Yava Technologies Inc. test well site at Silver Mine, Cape Breton County, Nova Scotia.

Pumping tests were performed during June 2004 to quantify drawdown characteristics of the aquifer in the vicinity of the test wells. This data was analyzed using Waterloo Hydrogeologic Inc. (WHI) software AquiferTest v. 4.0 to obtain transmissivity and storativity values for each of several pumping wells, and their associated monitoring wells (Michel 2004; Michel 2005).

Tracer testing was performed during September and November 2004 in order to determine the flow characteristics of the aquifer. The resulting data was analyzed using Microsoft Office EXCEL 2003 to develop breakthrough curves that indicated aquifer flow times between wells at the test site.

Packer tests were performed from June to November 2004 to determine hydraulic conductivity in the Silver Mine sandstone, and to outline fracture zones within the sandstone. This data was analyzed using the method outline in (*Golder, 1963*)

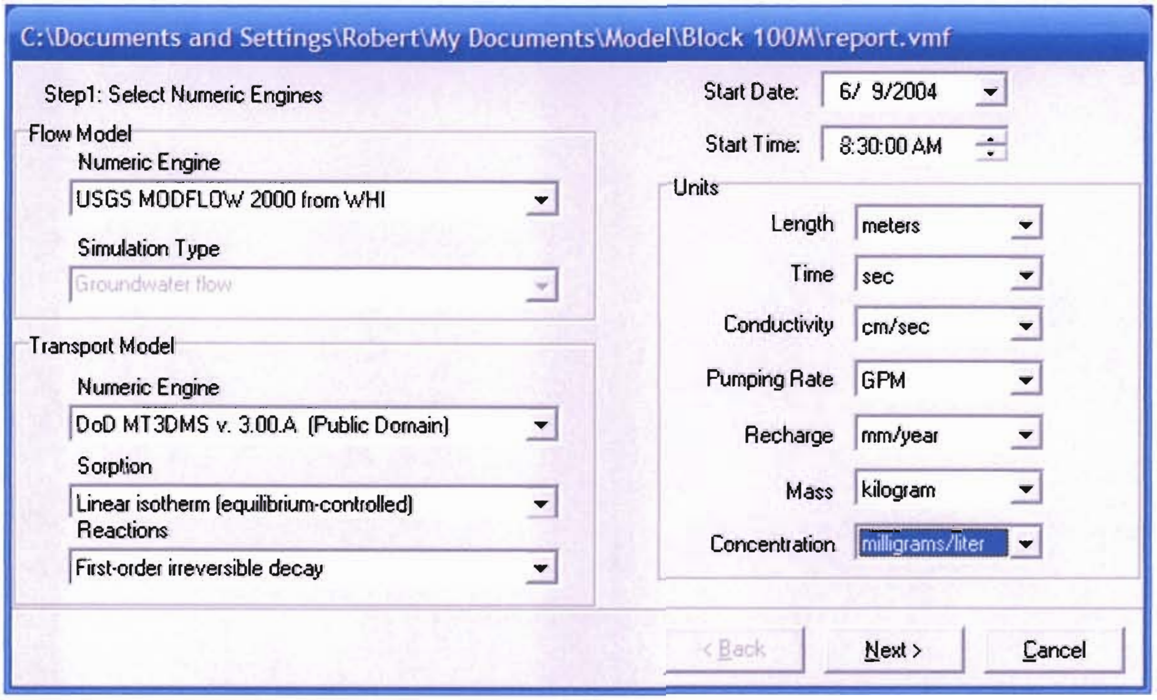
This information was then used to set up a model with the Waterloo Hydrogeologic Inc. software Visual MODFLOW Pro v. 4.0 (VMP4) so that permeability trends within the ore zone in the sandstones of the Silver Mine Formation at the Yava Silver Mine Site could be determined. VMP4 operates in four modes: set up and initial input, input, run and output. The output trends from the model runs were then used to design a well field for in situ mining at the Yava Silver Mine Site (WHI 2004).

2. MODELING THE YAVA TEST SITE

2.1 Visual MODFLOW Pro v. 4.0

VMP4 is a modeling environment for 3-D groundwater flow. This software can model groundwater flow in an aquifer, such as the Silver Mine formation sandstone, and then approximate the reaction of a theoretical well field.

Initial input data is required to set up the model (Fig. 2). The numeric engine supplied with the software is the USGS MODFLOW 2000 from WHI which simulates groundwater flow. The transport model used is the DoD MT 3DMS v 3.00A which is in the public domain. This model can be specified with different sorption and reaction types depending upon the chemical environment that is to be modeled. Details on numeric engines available for flow and transport, with available options are outlined in the manual supplied with the software (WHI 2004). The start time and date of the system being modeled (in this case the pumping test at 0830 hrs on the 9th of June 2004) are input along with the model units as indicated in Figure 2.



C:\Documents and Settings\Robert\My Documents\Model\Block 100M\report.vmf

Step1: Select Numeric Engines

Flow Model

Numeric Engine: USGS MODFLOW 2000 from WHI

Simulation Type: Groundwater flow

Transport Model

Numeric Engine: DoD MT3DMS v. 3.00.A (Public Domain)

Sorption: Linear isotherm (equilibrium-controlled)

Reactions: First-order irreversible decay

Start Date: 6/ 9/2004

Start Time: 8:30:00 AM

Units

Length: meters

Time: sec

Conductivity: cm/sec

Pumping Rate: GPM

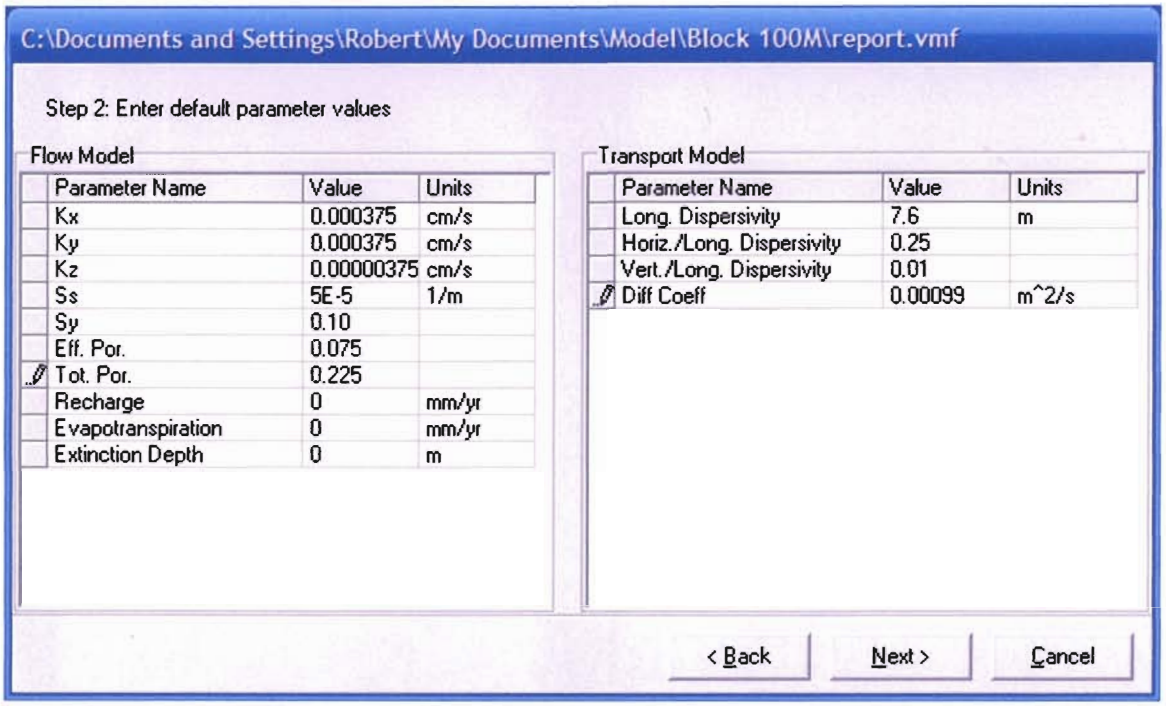
Recharge: mm/year

Mass: kilogram

Concentration: milligrams/liter

Figure 2 – Initial input screen in VMP4.

The default parameters, hydraulic conductivity (K), specific storage (S_s), specific yield (S_y), effective and total porosity, were then entered into the flow model; while, diffusivity estimates and a diffusion coefficient for sandstone were entered into the transport model as indicated in Figure 3.


C:\Documents and Settings\Robert\My Documents\Model\Block 100M\report.vmf

Step 2: Enter default parameter values

Flow Model

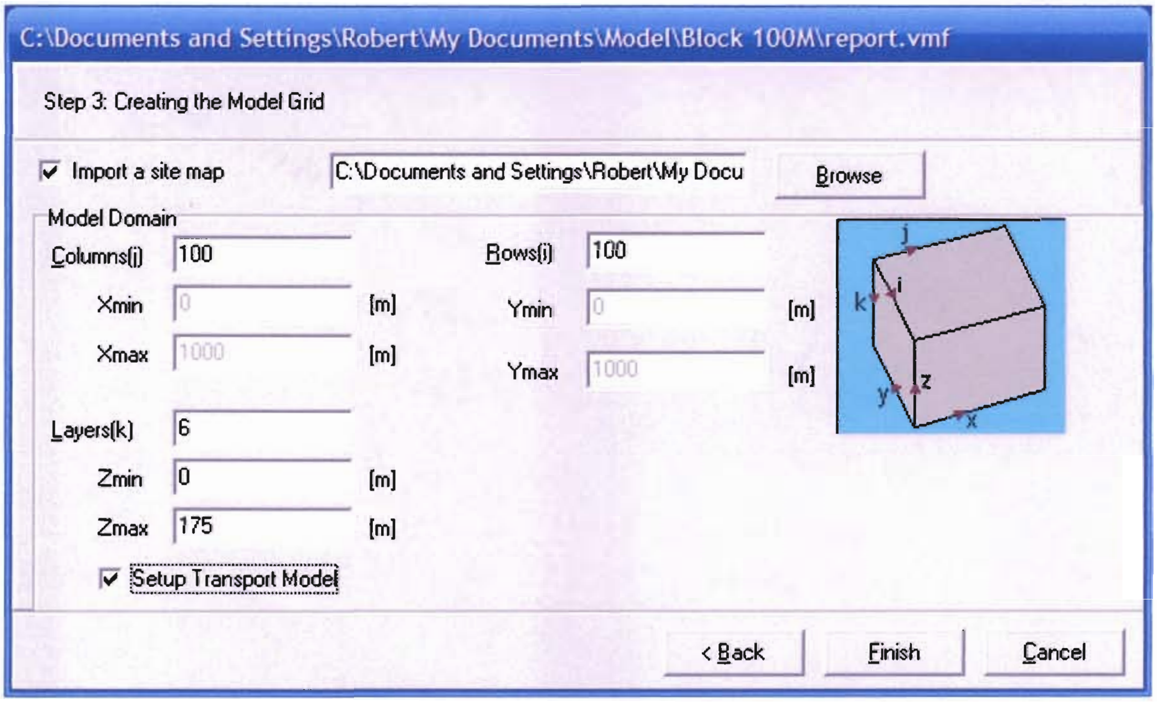
Parameter Name	Value	Units
Kx	0.000375	cm/s
Ky	0.000375	cm/s
Kz	0.00000375	cm/s
Ss	5E-5	1/m
Sy	0.10	
Eff. Por.	0.075	
Tot. Por.	0.225	
Recharge	0	mm/yr
Evapotranspiration	0	mm/yr
Extinction Depth	0	m

Transport Model

Parameter Name	Value	Units
Long. Dispersivity	7.6	m
Horiz./Long. Dispersivity	0.25	
Vert./Long. Dispersivity	0.01	
Diff Coeff	0.00099	m^2/s

Figure 3 – VMP4 default parameter inputs.

The third step is to set up the model grid. In this case, a grid consisting of 100m in the x-direction, 100m in the y-direction, and 175 m in the z-direction (depth) was set up to represent the Yava Silver Mine test-well field. This window also includes input for the site map and initialization of the transport model (Fig. 4).


C:\Documents and Settings\Robert\My Documents\Model\Block 100M\report.vmf

Step 3: Creating the Model Grid

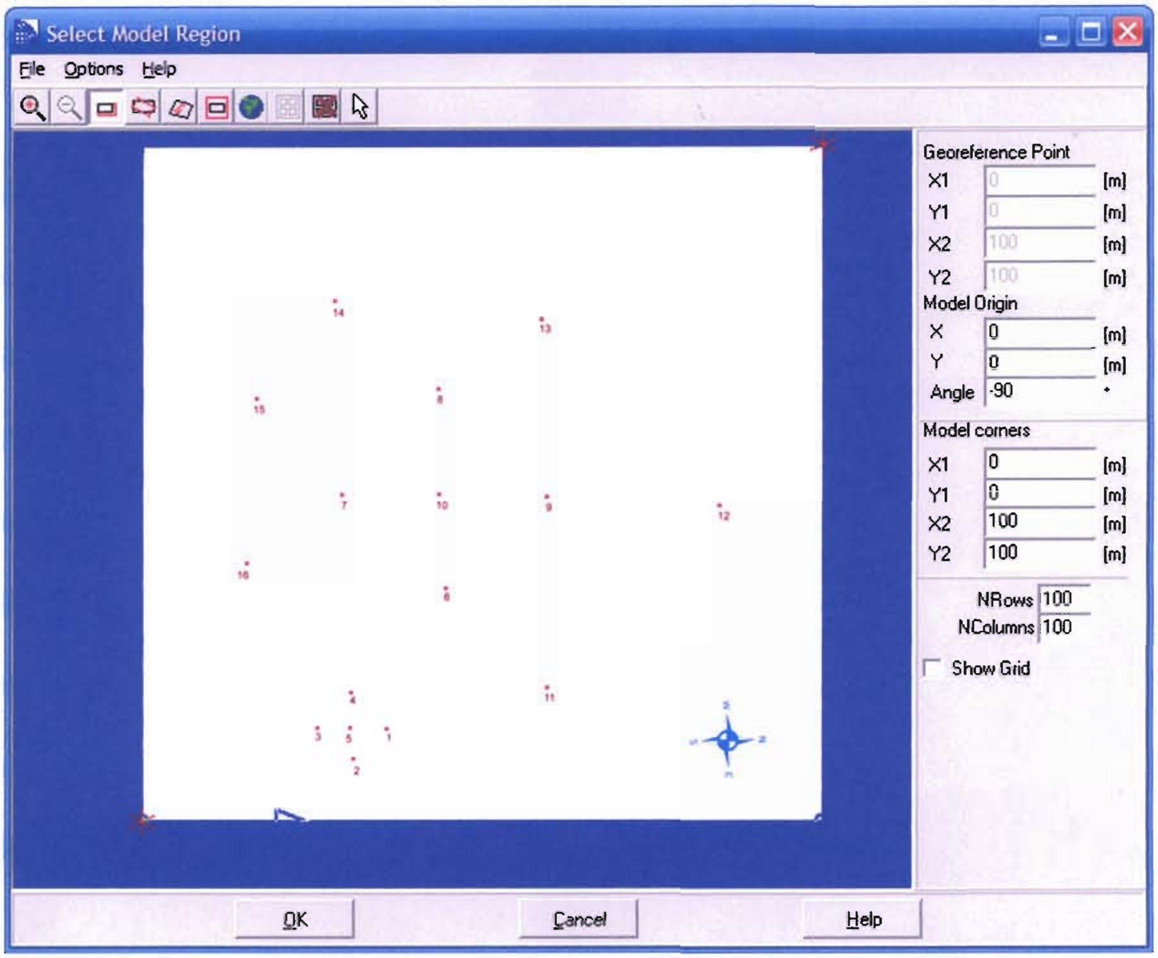
☑ Import a site map: C:\Documents and Settings\Robert\My Docu

Model Domain

Columns(j): 100
Xmin: 0 [m]
Xmax: 1000 [m]
Rows(i): 100
Ymin: 0 [m]
Ymax: 1000 [m]
Layers(k): 6
Zmin: 0 [m]
Zmax: 175 [m]

☑ Setup Transport Model

Figure 4 – VMP4 grid set up window.



Georeference Point
X1 0 [m]
Y1 0 [m]
X2 100 [m]
Y2 100 [m]

Model Origin
X 0 [m]
Y 0 [m]
Angle -90 °

Model corners
X1 0 [m]
Y1 0 [m]
X2 100 [m]
Y2 100 [m]

NRows 100
NColumns 100
☐ Show Grid

Figure 5 – VMP4 georeference setup screen

At this point, **VMP4** allows the user to lock the grid created in the last step to a real world position if desired. The model used in this analysis was not locked to any world coordinate system because the calibrated results of the tests were used to develop trends in the hydraulic conductivity existing in the ore zone in the Yava Silver Mine sandstone.

3.2 Input Data

In addition to the initial input of data as indicated in section 3.1, VMP4 allows the user to tailor layers to mimic the real world. The surface topography, the bottom of the overburden layer, the extents of the sandstone, the Windsor Formation and the top of the rhyolite (Michel 2004) were digitized using Golden Software Inc. Surfer 8.01 which produced EXCEL databases that could be directly imported into the model (Appendix A). VMP4 produces 3D layer maps that indicated thickness as is indicated in Figure 6 (Thickness of the overburden), Figure 7 (thickness of the sandstone) and Figure 8 (thickness of the clay).

The sandstone strata was further divided into nine layers based on the permeability of the

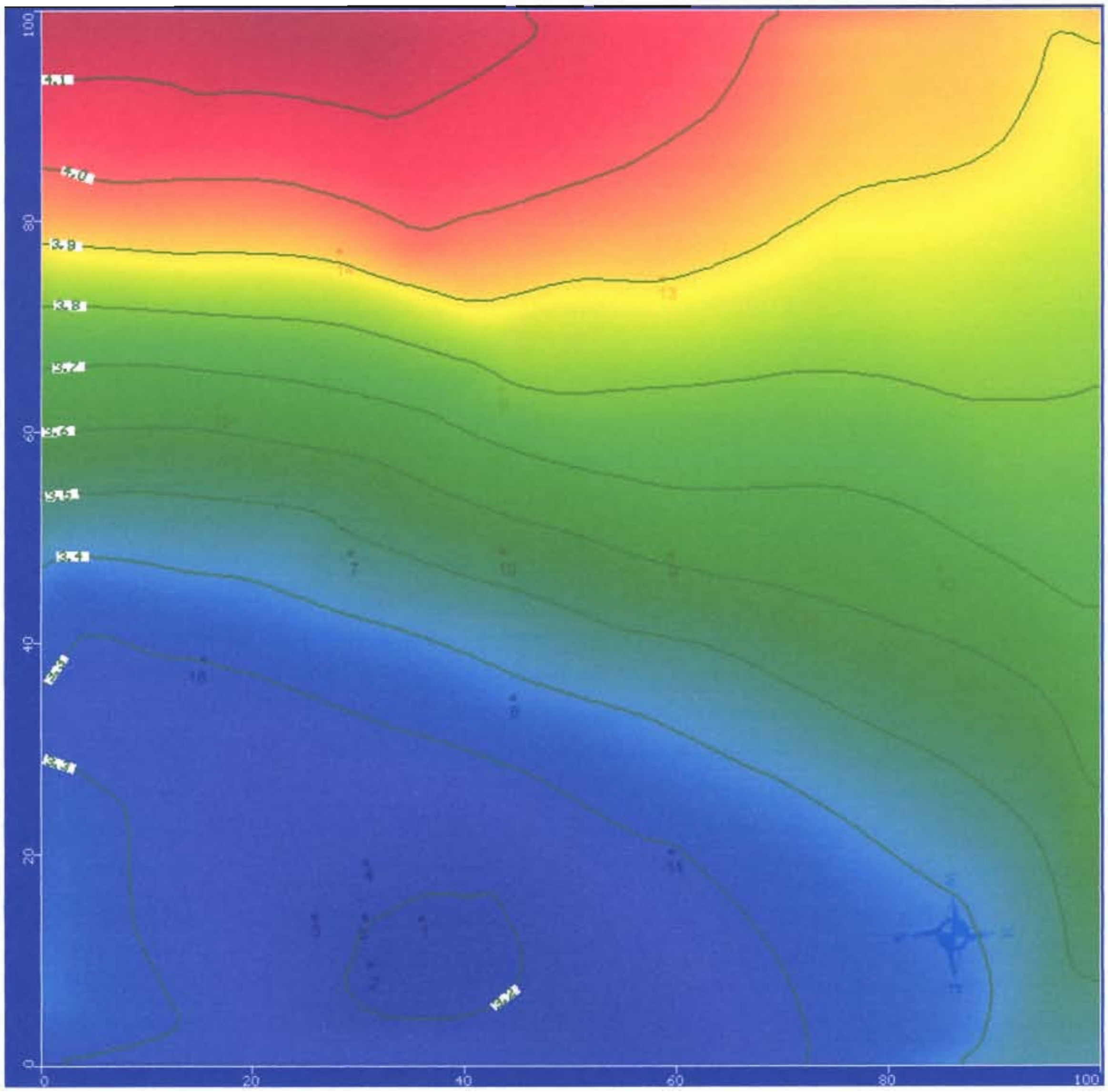


Figure 6 - Thickness of the overburden at the Yava Silver Mine test well field. Thicknesses range from 3.1 to 4.2 meters; distances north and south are indicated in meters; and, the locations of the test wells are indicated as numbered dots.

matrix; and also, whether fractures were present, and how they effect the flow between test wells. A method based on that outlined by Golder and Associates (1963) was used to calculate hydraulic conductivity values for the data acquired during packer testing performed between June and November 2004. To refine the model, the sandstone unit was divided into 9 zones according to the derived hydraulic conductivity values. The hydraulic conductivities for each of the packer tests were calculated as indicated in Appendix B, combined into flow diagrams based upon whether the monitoring wells showed a negative drawdown (Michel, 2004) with the packer at different depths (Appendix C).

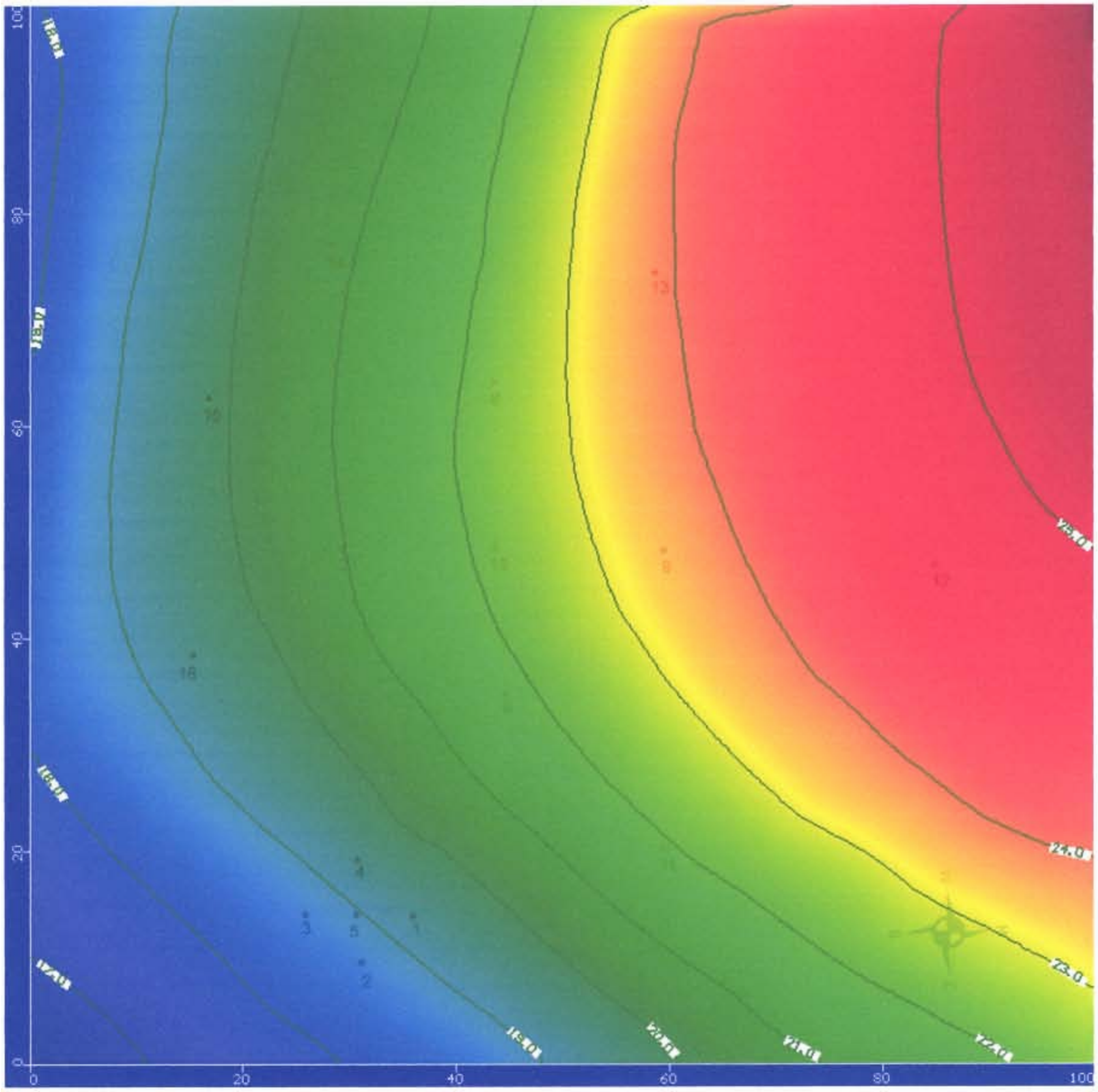


Figure 7 - Thickness of the sandstone (middle layer) at the Yava Silver Mine test well field. Thicknesses range from 16 to 26 meters; distances north and south are indicated in meters; and, the locations of the test wells are indicated as numbered dots.

The flow maps and the Stratigraphy were combined to produce cross-sections of the hydraulic conductivity zones (Appendix D), and these cross-sections were used to produce a 3D - nine layer model of conductivity within the sandstone unit. Once again, EXCEL 2003 was used to produce layer bottom data files that were then input into the model (Appendix E). A cross-section of the resulting diagram from the VMP4 model is shown as Figures 9.

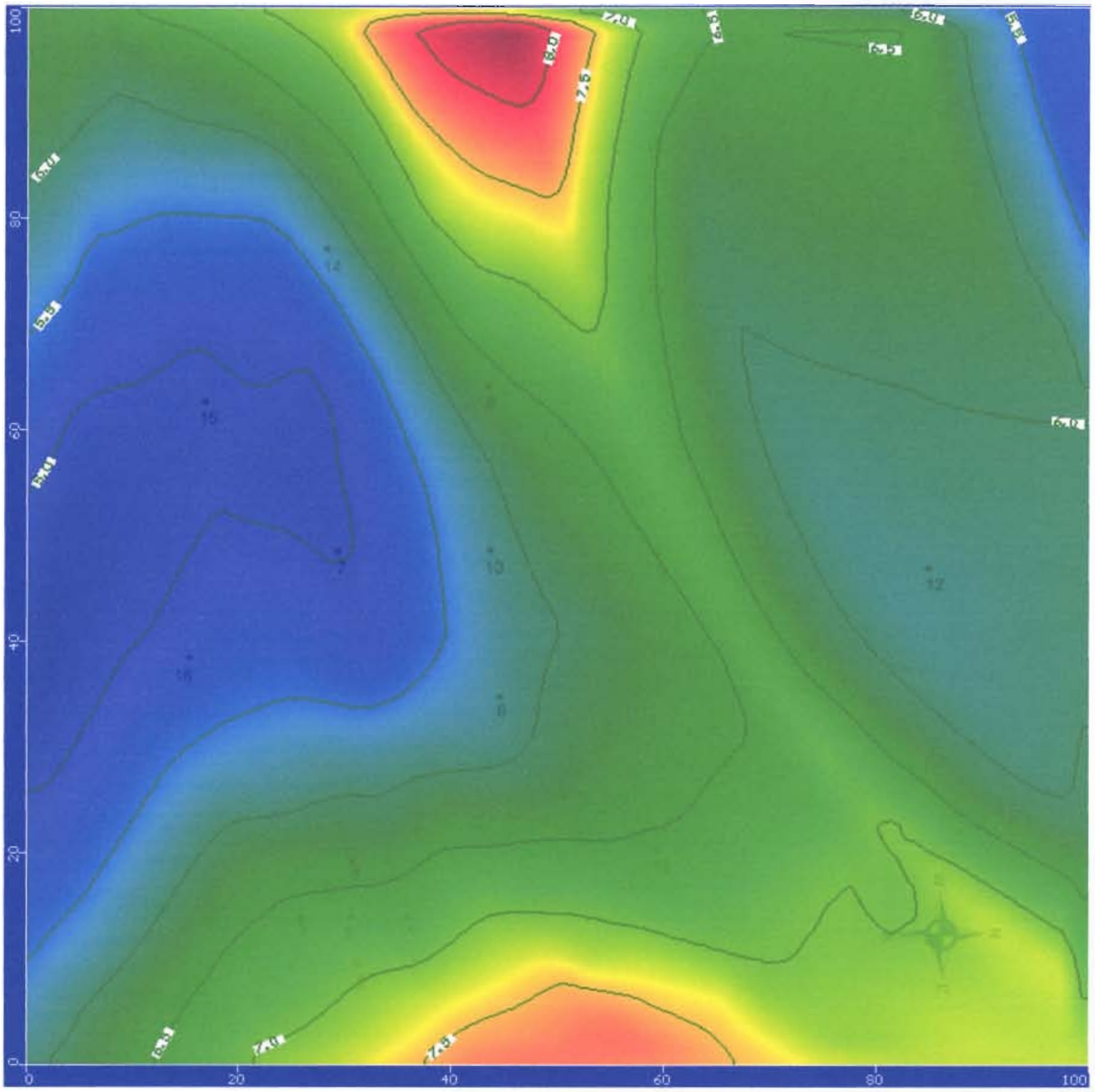


Figure 8 - Thickness of the clay (bottom layer) at the Yava Silver Mine test well field. Thicknesses range from 4.5 to 8.5 meters; distances north and south are indicated in meters; and, the locations of the test wells are indicated as numbered dots.

In addition to the hydraulic conductivity inputs to the model, two other databases were required before the model could be run; hydraulic head and specific storage.

Hydraulic head measurements were calculated by subtracting the static head measurements taken previous to pumping tests from the surface elevations at the test well site (Appendix F). These hydraulic heads were put into an EXCEL 2003 database, then input into the VMF4 model (Figure 10).

Specific storage was estimated by dividing the storativity (Michel, 2004) by the thickness of the sandstone unit, then formed into a database with EXCEL 2003 and input into the VMF4 model (Appendix F; Figure 11).

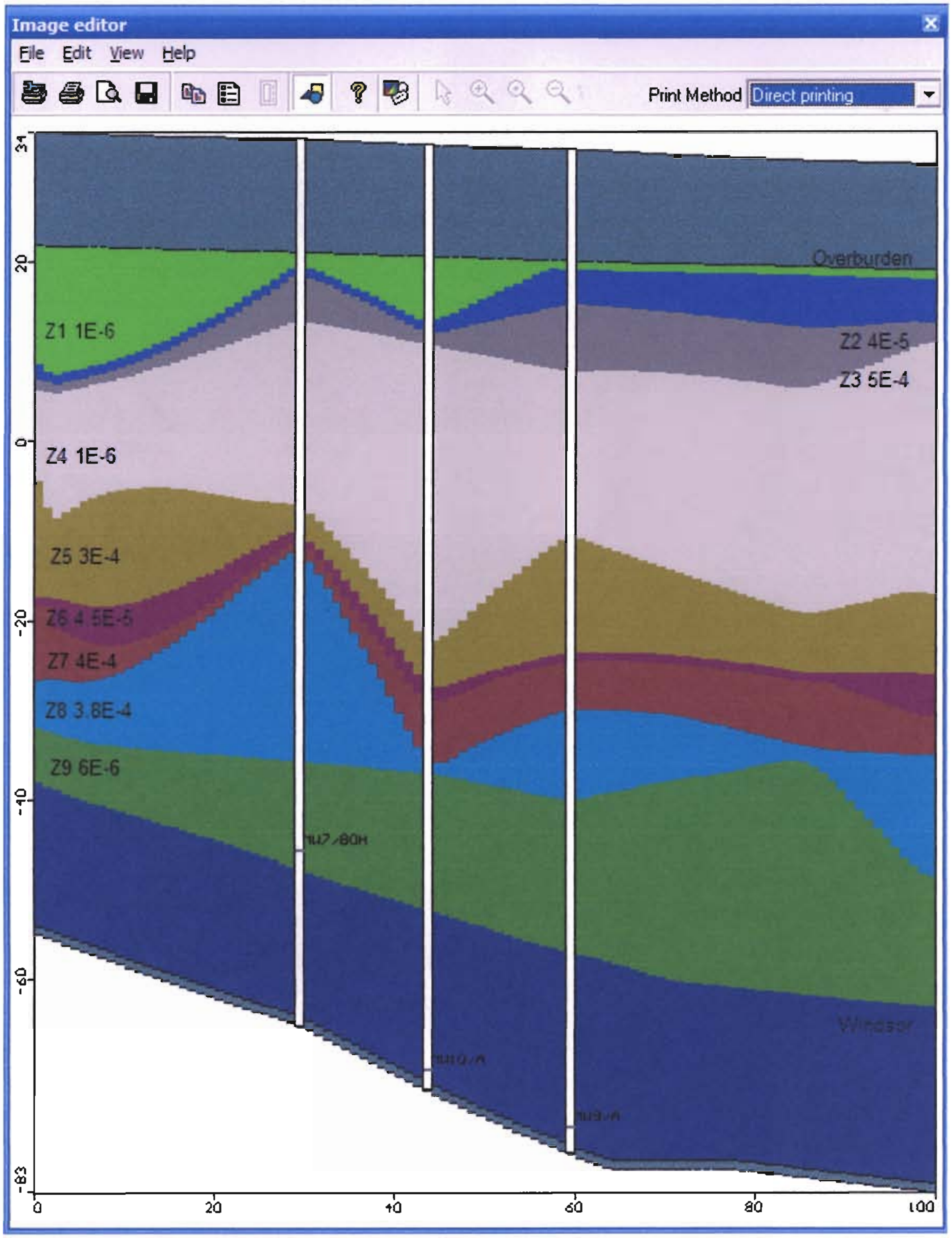


Figure 9 – Cross-section from south to north (left to right) through the model grid. This section indicates the relationship of the hydraulic conductivity zones within the sandstone at the Yava Silver Mine site. Depths are in meters above sea-level and horizontal distance indicated is in meters.

General Head Boundaries (GHB) were input onto the boundaries of the model. The GHB's were set for the time of the model run, and based on the previously input

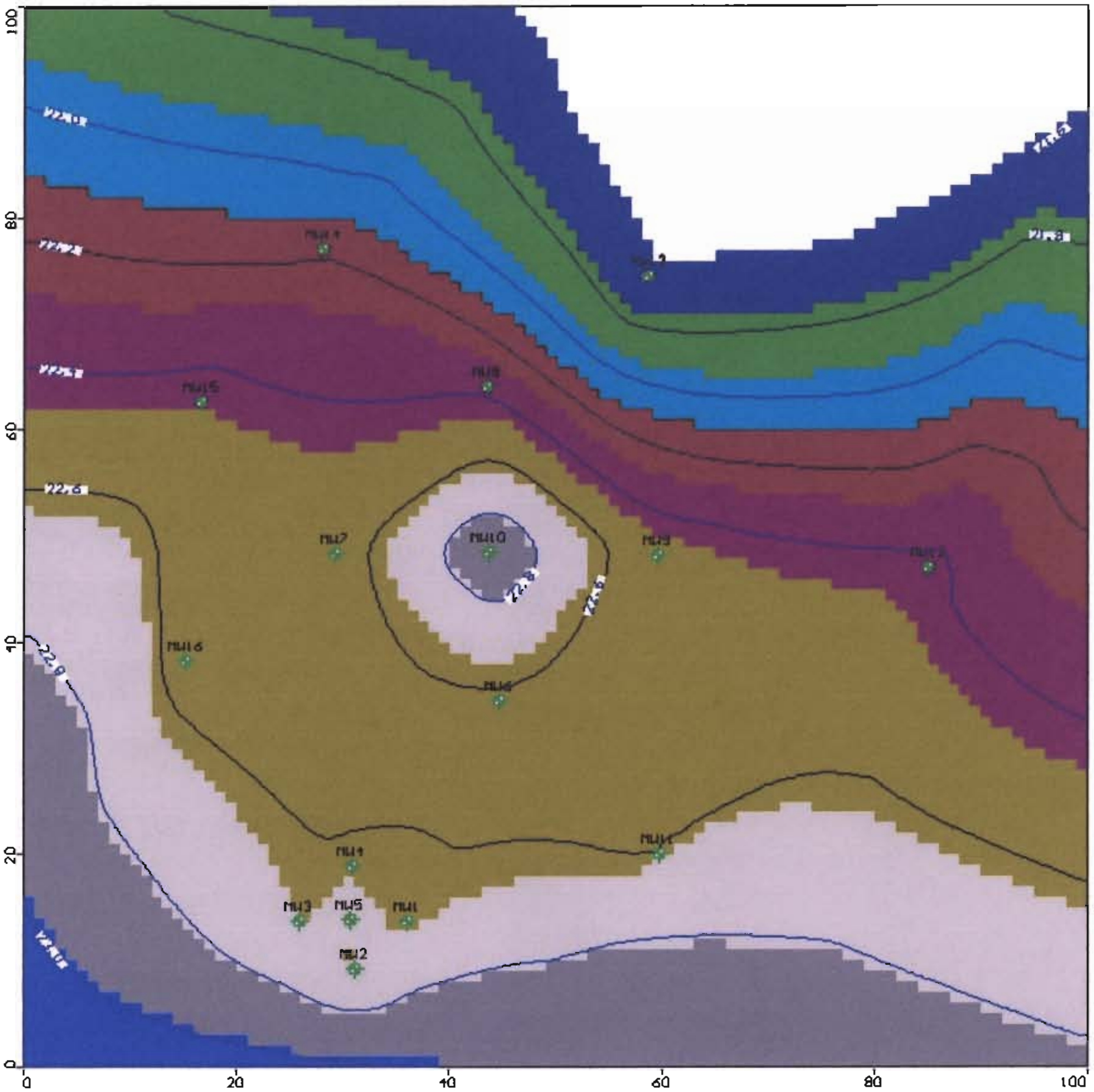


Figure 10 – Initial hydraulic head map for VMP4 model. Horizontal distances are in meters and head data are meters above sea-level.

values of hydraulic conductivity and hydraulic head.

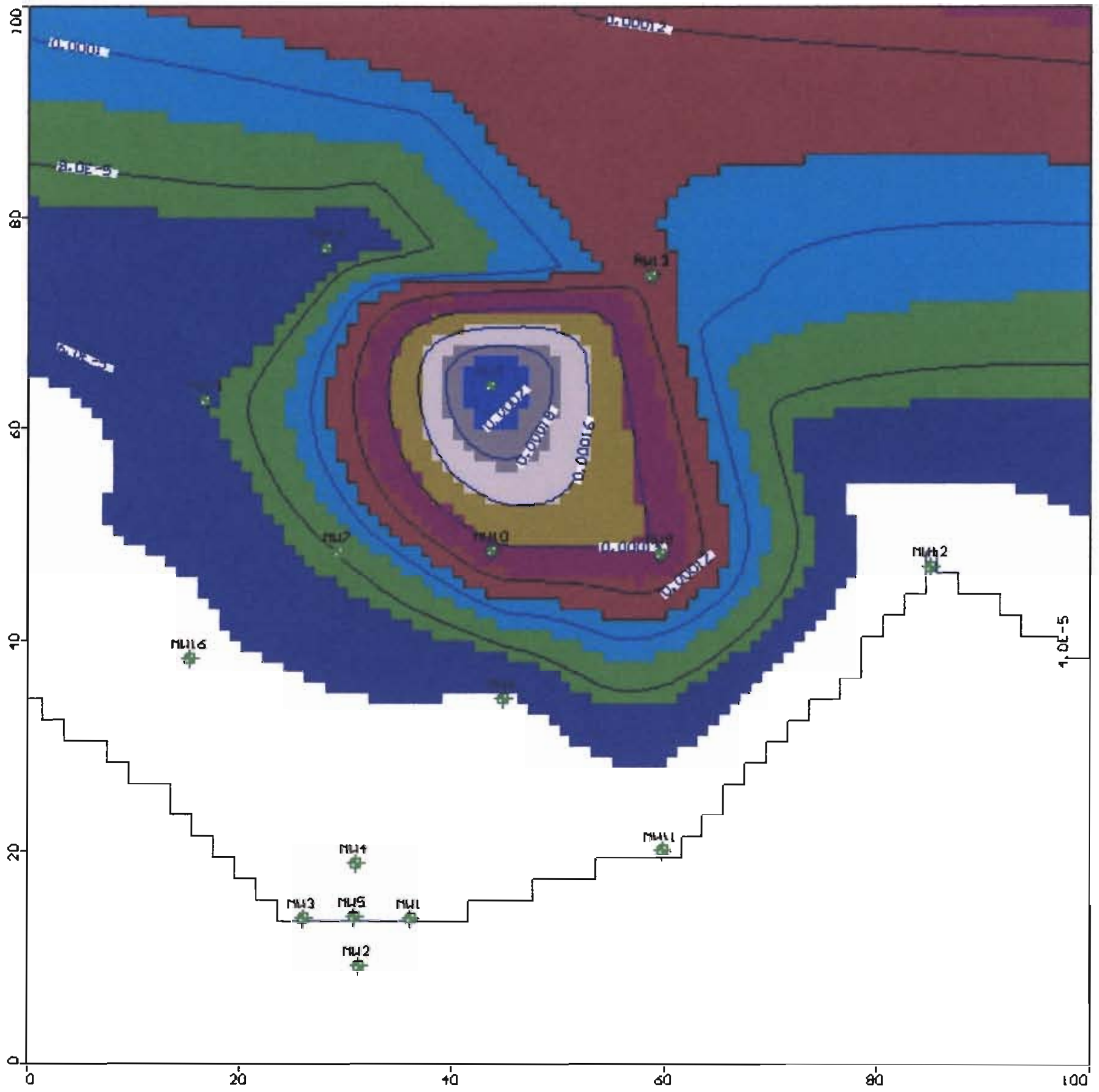


Figure 11 – Initial Specific Storage map input into the VMP4 model. Specific storage values are in inverse meters, and distances are in meters.

3.3 Calibration

At this point, the model was switched to the run phase in order to calibrate input data to the real world. This was done by comparing the drawdown data at the monitoring wells to the results produced by VMP4 on a pumping test run. If the data matched, the model was considered calibrated. However, this usually only happened after numerous runs with converging changes of hydraulic conductivity at monitoring wells. An example of output data from a preliminary run, indicating drawdown is indicated in Figure 12.

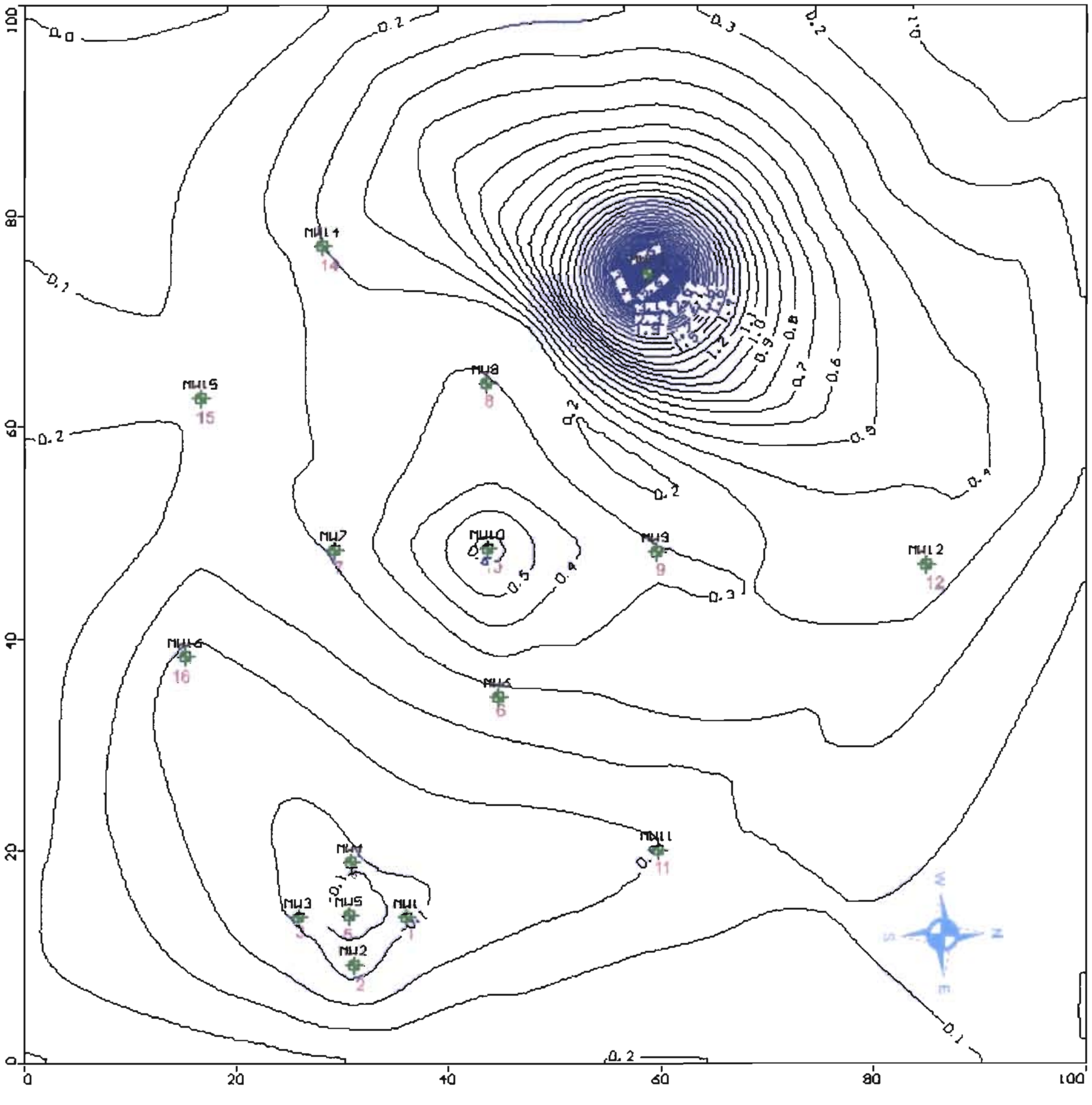


Figure 12 – Preliminary drawdown model data output from a VMP4 calibration run (pumping test for pumping well 13 on 9 June 2004). Drawdown values and grid distances are in meters .

At this stage, it is evident from the above figure, that there is an elliptical, southwest to northeast, trend in the drawdown data. This trend was developed after numerous calibration runs with changes of hydraulic conductivity inputs between runs. The resulting hydraulic conductivity map is indicated in Figure 13.

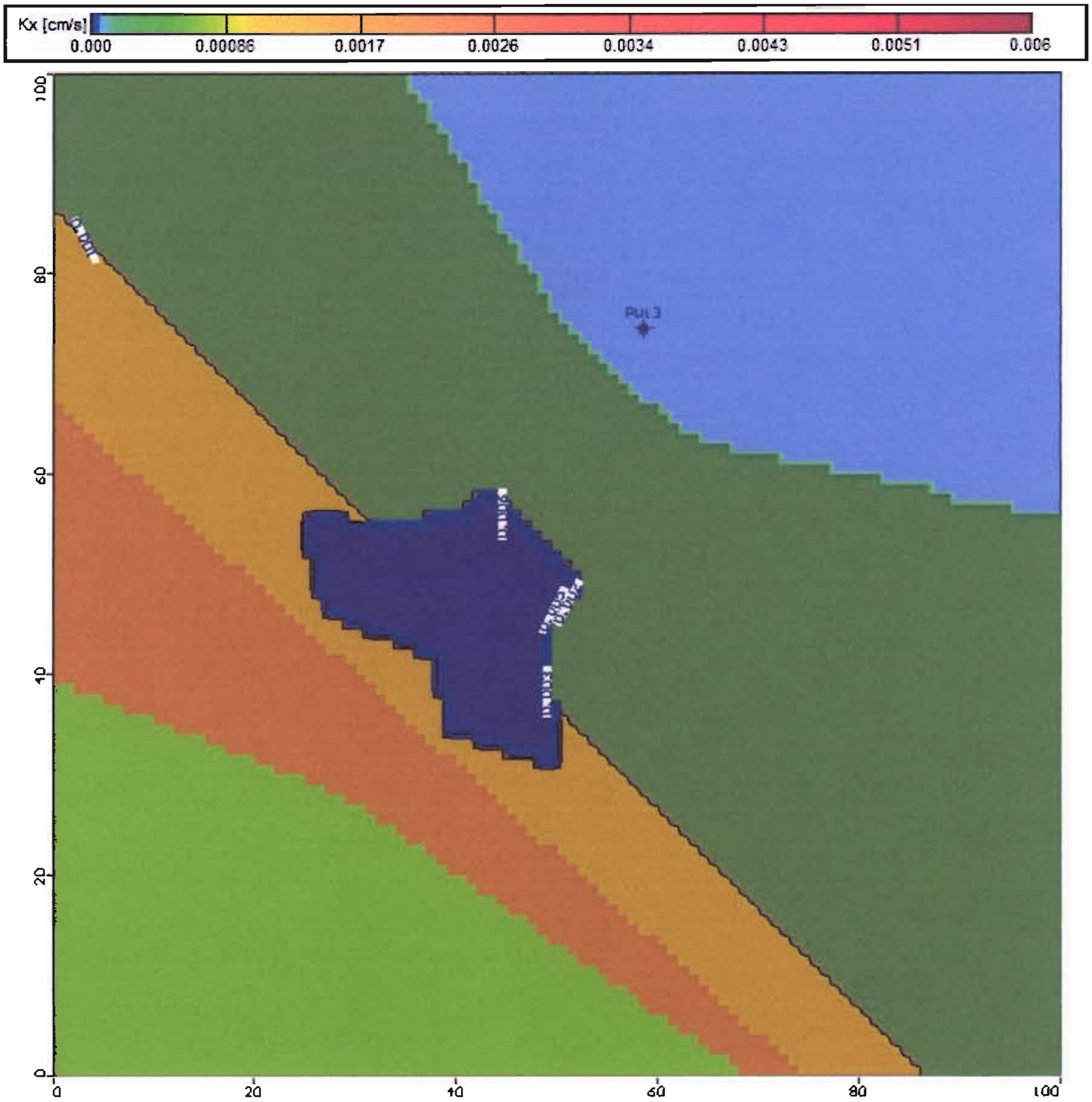


Figure 13 – Results of hydraulic conductivity changes after numerous calibration runs; for Zone 8 in the modeled sandstone layer. Distances are in meters.

In order to refine this model further, Wall Boundaries (WB) were used. These WB's enabled VMF4 to restrict flow between the elliptical trending zones in the drawdown data. The WB's represent infilled fractures, indicating areas of reduced hydraulic conductivity.

Using the above model parameters, and adding data from the remaining pumping tests for June and November 2004, further model runs were attempted. When calibration was attempted on the full model, it was discovered that VMF4 could not handle the required daily changes in hydraulic head, hydraulic conductivity and Specific Storage. As a

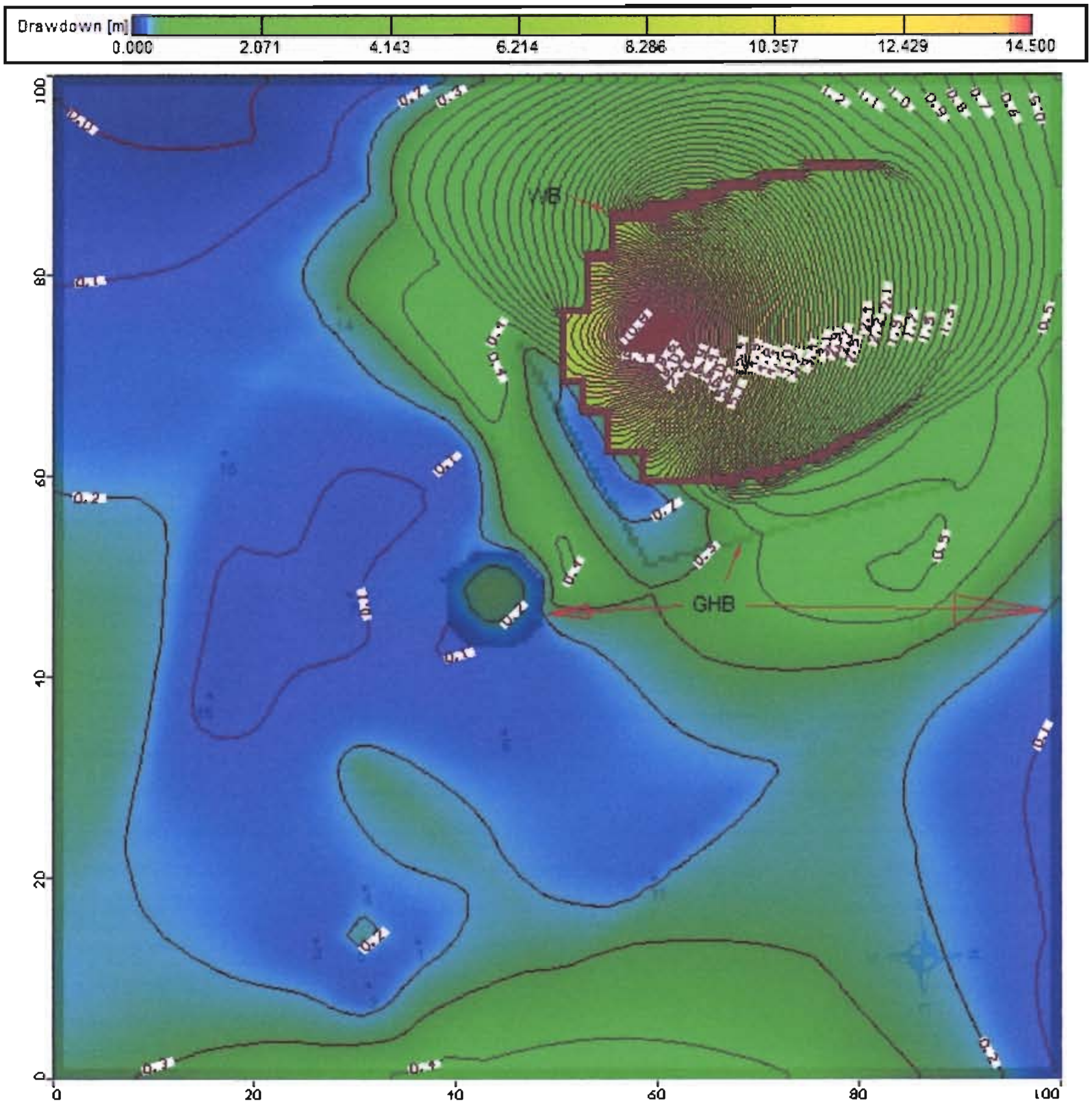


Figure 14 – Results of VMF4 modeling run for pumping test on Well 13 dated 9 June 2004. Shows how General Head Boundaries (GHB) and Wall Boundaries (WB) were used to separate hydraulic conductivity zones. Drawdown results and horizontal distances are in meters.

result, independent model runs were calibrated for each of the pumping tests in order to calibrate drawdown patterns for each pumping well.

3.4 Results

The drawdown patterns for each of the pumping wells gave a range of hydraulic conductivity zones that were modeled within the ore zone of the Silver Mine Formation sandstone. This data gave a range of hydraulic conductivity zones, or blocks, which were separated by zones of lower conductivity (1 x 10^{-6} cm/s), representing infilled fractures; or higher conductivity (5 x 10^{-3} cm/s), representing open fractures. The drawdown data for each of the pumping tests are included as Figures 15 to 19.

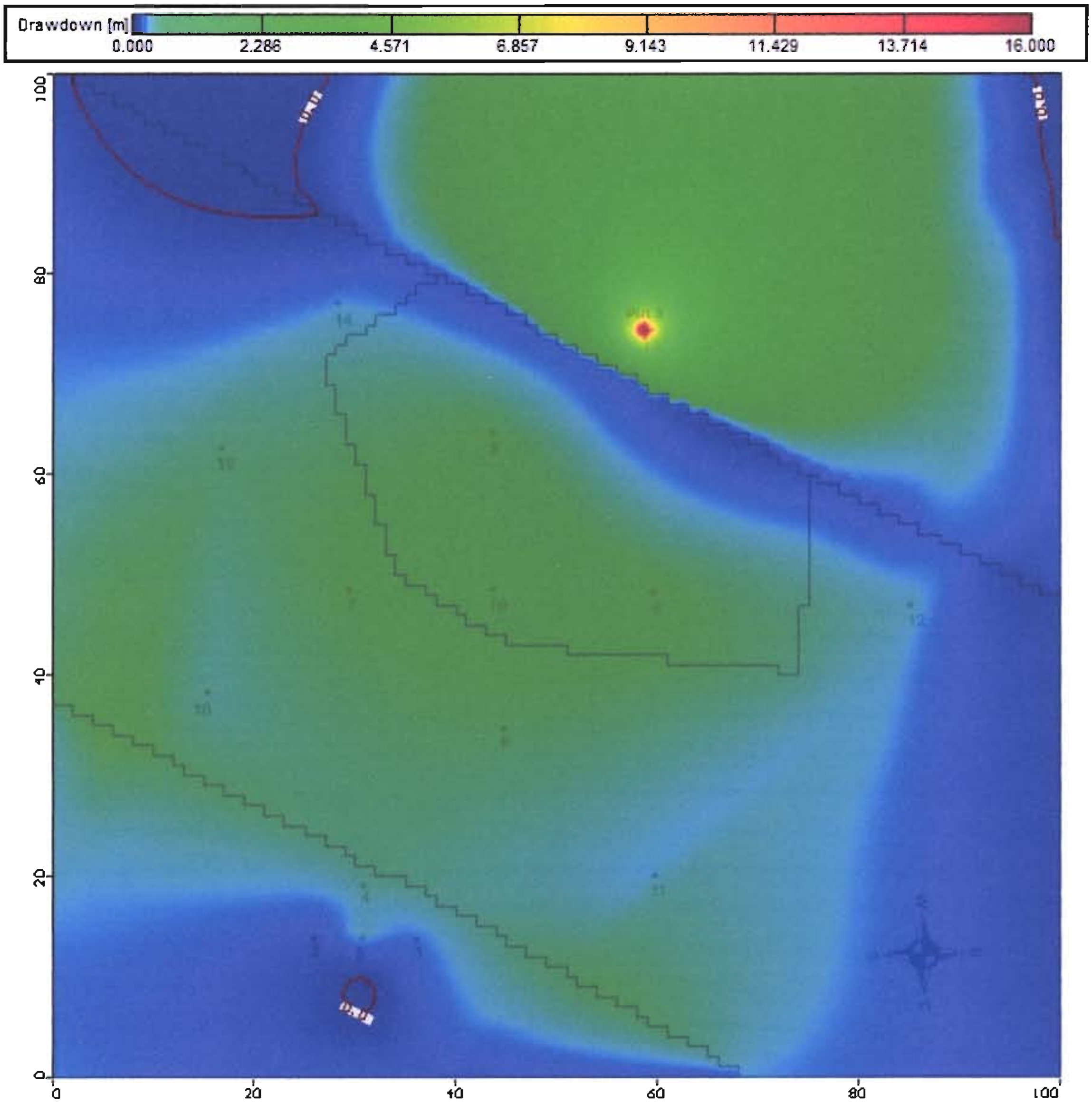


Figure 15 - Drawdown pattern resulting from modeling pumping test in Well 13, 09 June 2004. Horizontal distances are in meters.

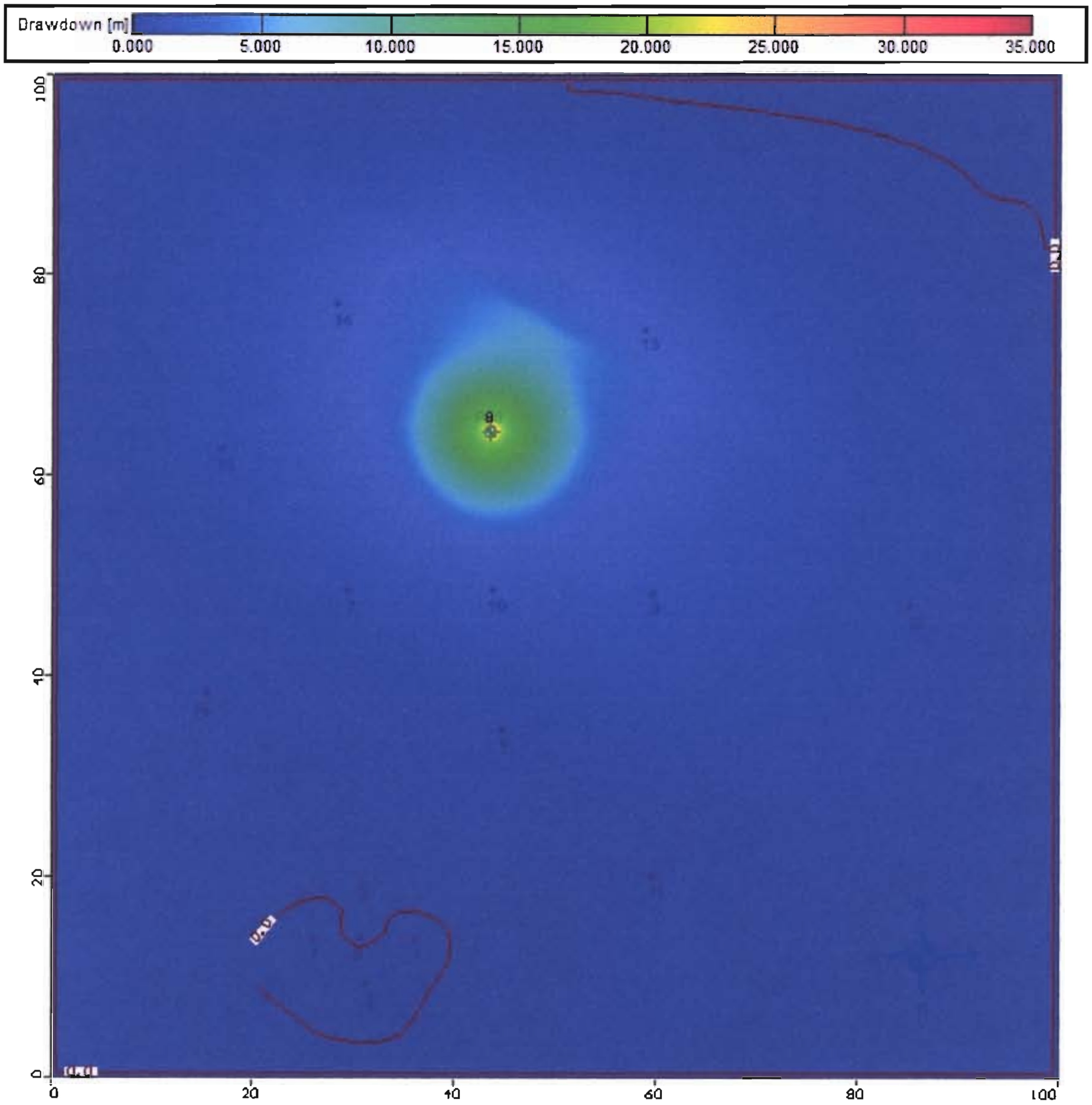


Figure 16 - Drawdown pattern resulting from modeling pumping test in Well 8, 10 June 2004. Horizontal distances are in meters.

Each of the final modeling runs were calibrated using different data for boundary conditions and for hydraulic conductivites. Initial boundary conditions were set for the initial input of hydraulic head data in section 3.2. Since the model was not capable of indicating the correct drawdown patterns based on hydraulic conductivity alone, the boundary conditions and the hydraulic conductivities were varied between pumping test runs.

For the pumping test on 09 June 2004 in Well 13, to achieve calibration drawdown, GHB's were set around the model site to make use of the hydraulic conductivity data, in order to obtain the drawdown data indicated in Appendix F.1. The GHB hydraulic heads were set from 23m to 21.8m in the south; 23m to 21.8m in the east and 22.8m to 21.5m to the north. Variation of the hydraulic conductivities was as represented in Figure 20, giving result indicated in Figure 15.

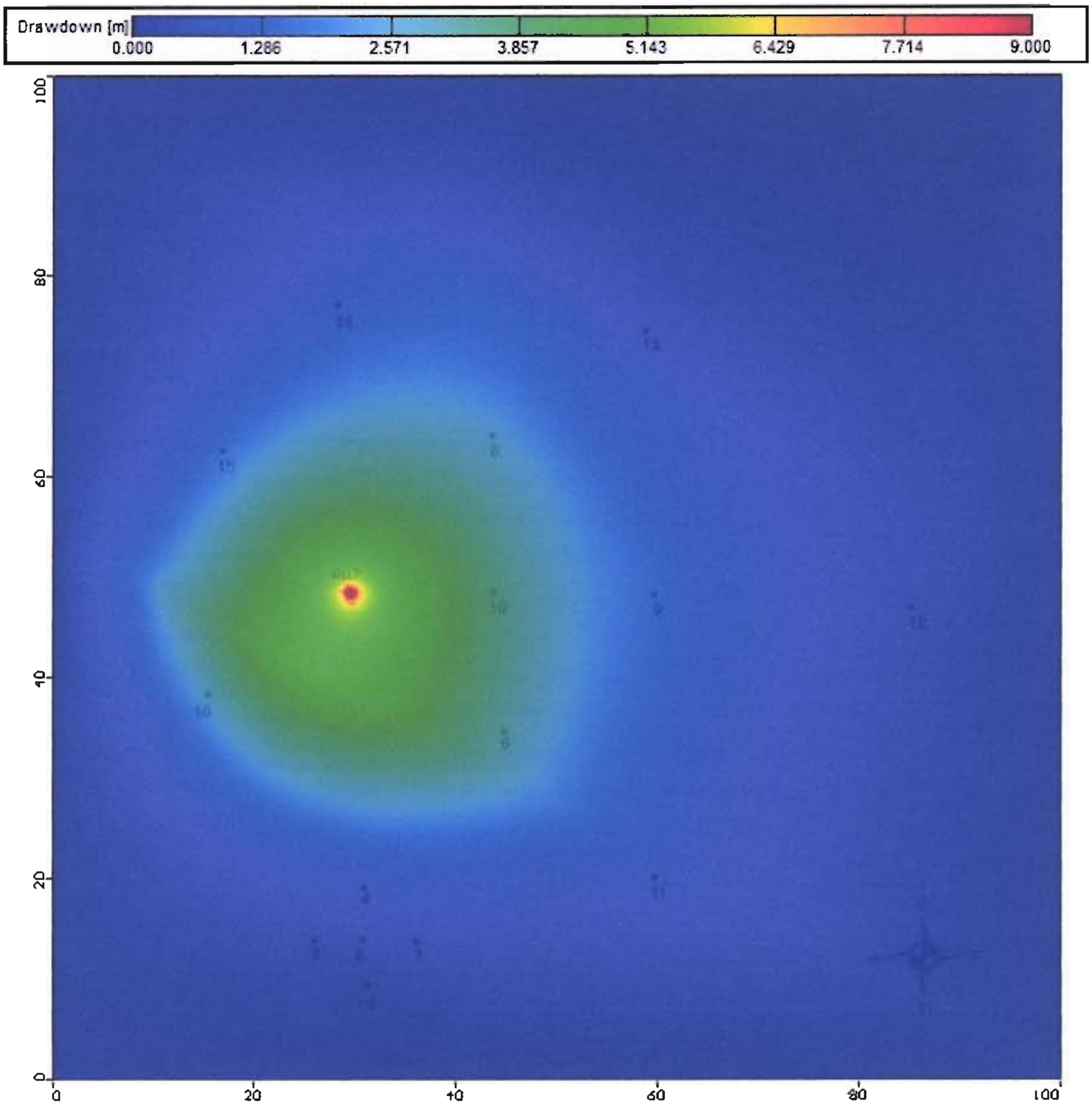


Figure 17 – Drawdown pattern resulting from modeling pumping test in Well 7, 12 June 2004. Horizontal distances are in meters.

To model the pumping test in Well 8 on 10 June 2004 a constant head boundary (CHB) was set along each of the boundaries of the model site; on the south side the CHB was set for a variation in hydraulic head from 23m in the east to 21.8m to the west, on the west side the CHB was set for a variation in hydraulic head from 21.8m in the south to 21.5m to the north, on the north side the CHB was set for a variation in hydraulic head from 22.8m in the east to 21.5m to the west, and on the east side the CHB was set for a variation in hydraulic head from 23m in the south to 22.8m to the north. Additionally, a wall boundary (WB) was set 2m thick facing the northwestern hydraulic conductivity zone which included monitoring well 13 (5×10^{-6} cm/s). The hydraulic conductivity of the WB was set at 1×10^{-7} cm/s.

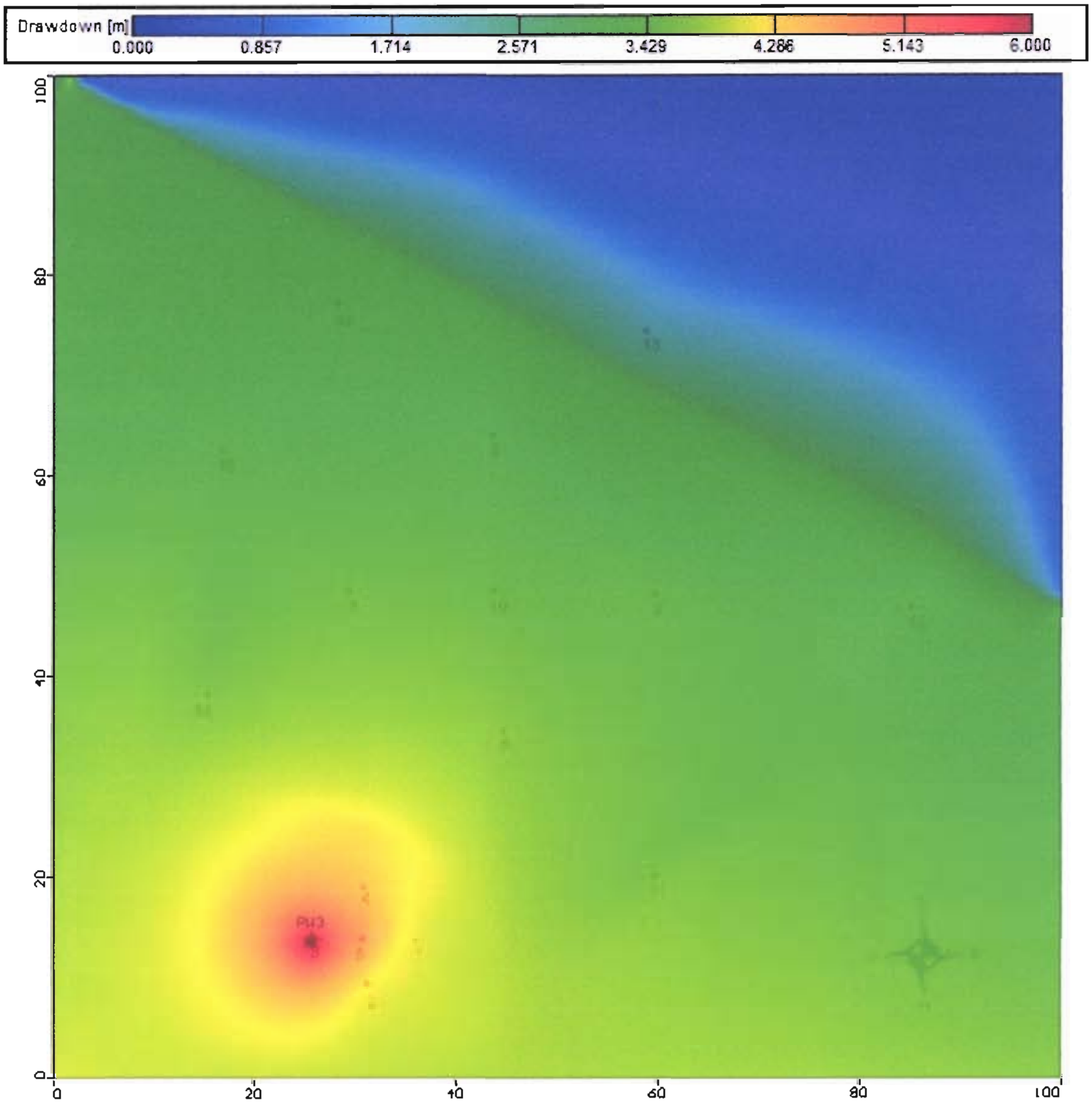


Figure 18 – Drawdown pattern resulting from modeling pumping test in Well 3, 14 June 2004. Horizontal distances are in meters.

To model the pumping test in Well 7 on 12 June 2004, CHB's were set as indicated for the pumping test conducted on 10 June, but hydraulic conductivities were varied as indicated in figure 20. This changed for the pumping test in Well 3 on 14 June 2004. Here GHB's were set around the model site because the hydraulic conductivities indicated in Figure 20 could be incorporated into this type of boundary. The hydraulic heads for this set of boundaries was set as follows: to the south, the head was set at 19m; to the west, the head was set at 21m; to the northwest, the head was set at 21m; to the northeast, the head was set at 19m; and, to the east, the head was set at 19m. A WB was set 5m thick, with a conductivity of 1 x 10^{-8} cm/s, facing the northwestern hydraulic conductivity zone that includes Well 13.

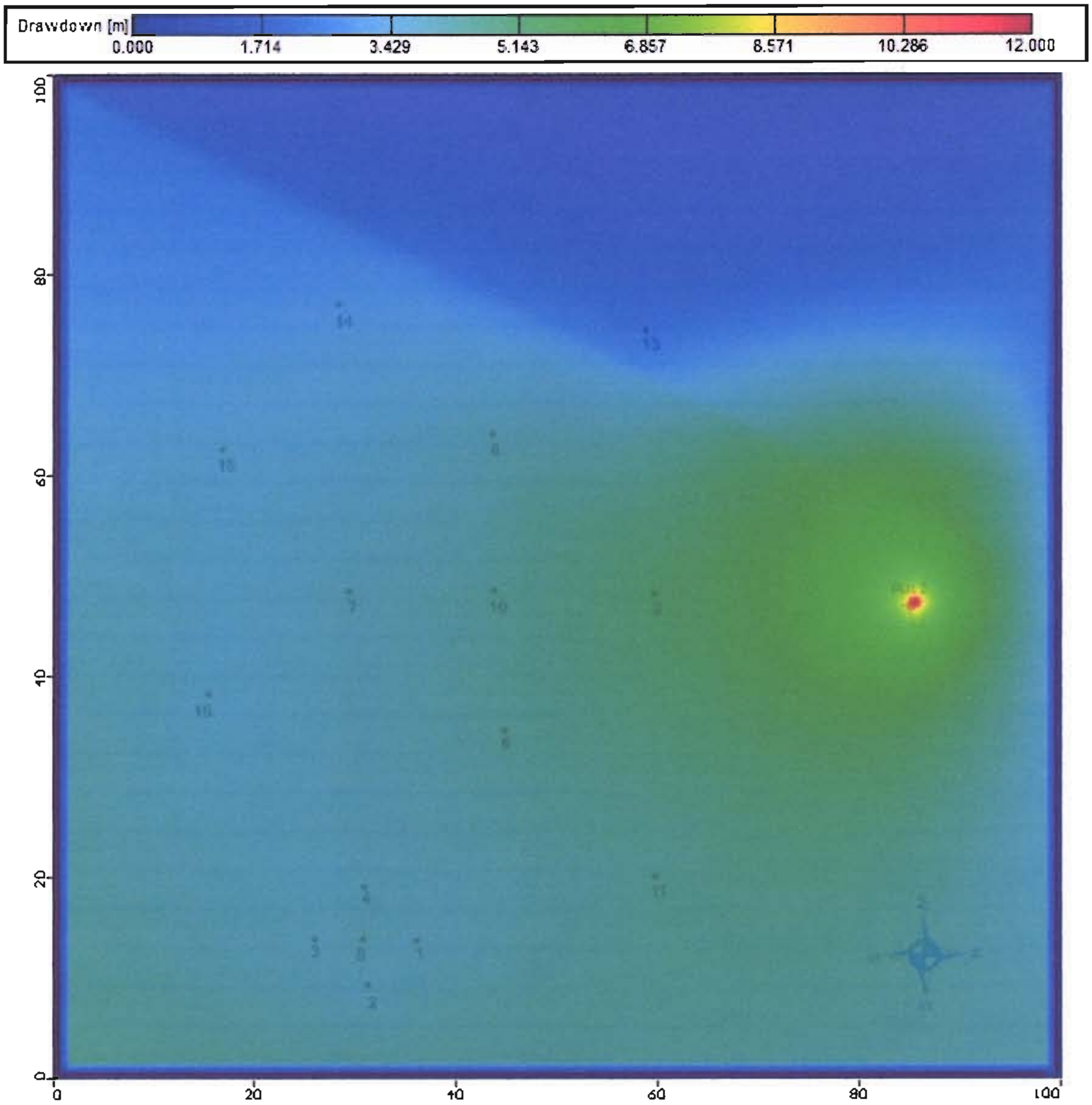
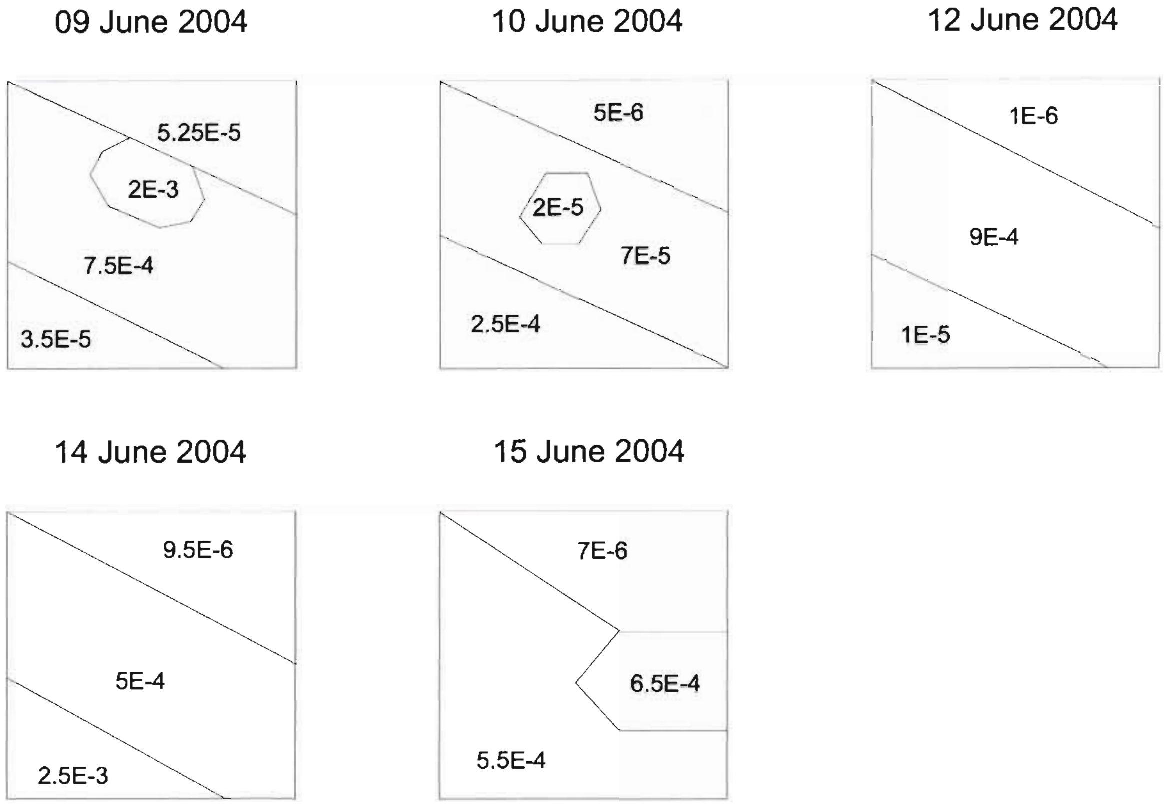


Figure 19 – Drawdown pattern resulting from modeling pumping test in Well 12, 15 June 2004. Horizontal distances are in meters.

To model the pumping test in Well 12 on 15 June 2004, a CHB was set to vary the model's hydraulic head over the time of the model run. The CHB's were set as follows: to the south, head was varied from 19m to 14m; to the west, head was varied from 20m to 15m; to the northwest, head was varied from 19m to 18m; to the northeast, head was varied from 19m to 13m; and, to the east, head was varied from 19m to 14m. The WB was set as indicated for the pumping test on 14 June.



09 June 2004

10 June 2004

12 June 2004

14 June 2004

15 June 2004

Figure 20 – Final hydraulic conductivity maps derived from modeling runs for pumping tests during June 2004. Hydraulic conductivities are in cm/s. North is to the right in all diagrams.

3.5 Verification

The modeled pumping tests for the above five dates were verified by having the model run a continual pumping test for seven days. The results of the verification runs were identical to the initial runs, indicating that steady-state had been achieved, and the model was considered to be verified.

3.6 Permeability Trends

The major difference between the modeled pumping test runs was the change in hydraulic conductivities that is apparent upon inspection of Figure 20. These differences indicated that a greater southwest to northeastern hydraulic conductivity trend exists at the Yava test site. The model indicates that the Silver Mine Formation sandstones are anisotropic in the xy-direction, with K_x (southwest to northeast) greater than K_y (southeast to northwest) from 6 to 33%, increasing towards the northwest. The model indicates an elliptical field based on the conductivities.

4 Well Field Design

To determine the design distance between wells, the maximum drawdown recorded at each of the monitoring wells was plotted against the logarithm of the distance from the pumping well for each of the pumping tests performed during June 2004 (Figure 21a & b).

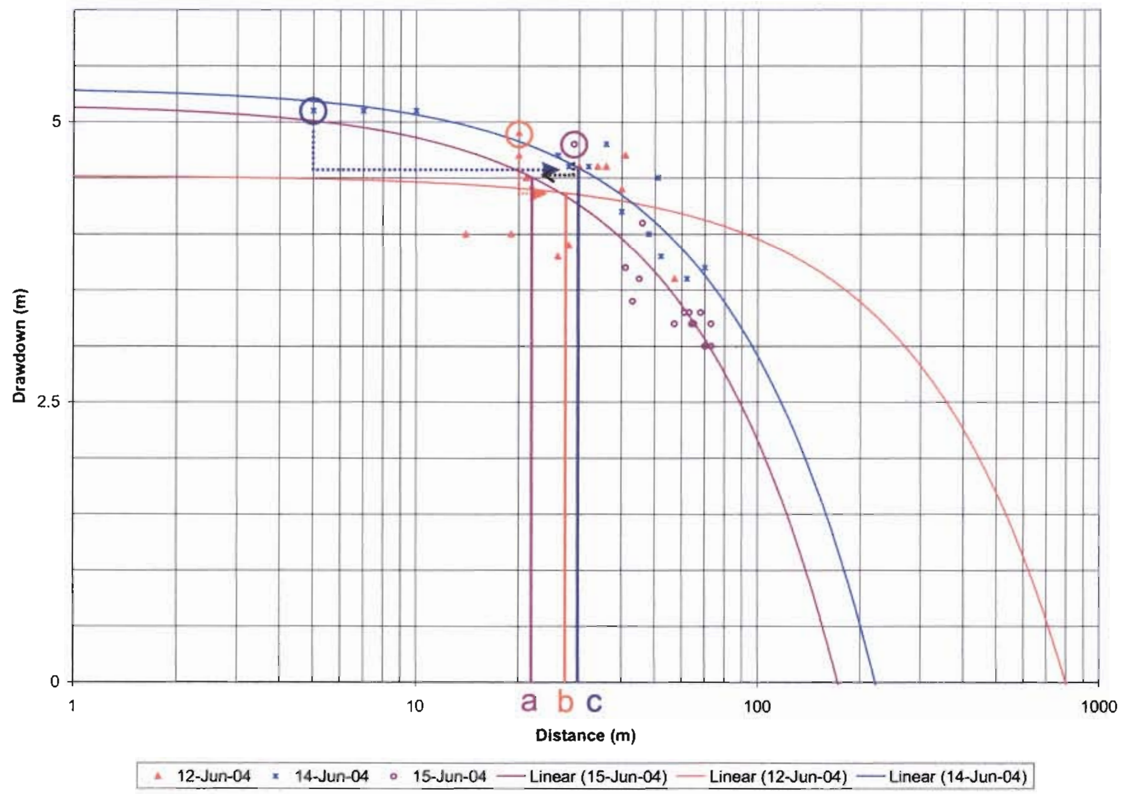


Figure 21a – Semi-log plot of maximum drawdown observed at monitoring wells vs distance from pumping well for pumping tests (Jacob 1940) conducted on 12, 14 and 15 June 2004. Maximum distance is the extents of the pumping well influence. Proposed well design distances are indicated as 22m based on pumping test data from 15 June 2004 (a); as 27m based on pumping test data from 12 June 2004 (b); and as 30m based on pumping test data from 14 June 2004 (c).

In order to ensure complete coverage of the ore zone, 90% of the maximum drawdown indicated by the monitoring wells was used. This gave potential distances between wells as indicated in Figures 21a and 21b. To check these results, well field cells (5-spots) were created consisting of four injection wells, one on each of the corners of a square, with a pumping well in the center. These 5-spots were then tested with baseline distances at 20, 25 and 30 meters apart. For the three tests conducted on the 14 June, the injection wells were set to 25 gpm and the pumping well was set at 150 gpm. The results of the three tests are outlined in figures 22, 23 and 24. For the three tests conducted on the 15 of June, the injection wells were set to 25 gpm for the test at 20 m, and to 75 m for the tests at 25 and 30 m. The pumping wells for all tests on the 15 of June were set to 40 gpm. The Results of the

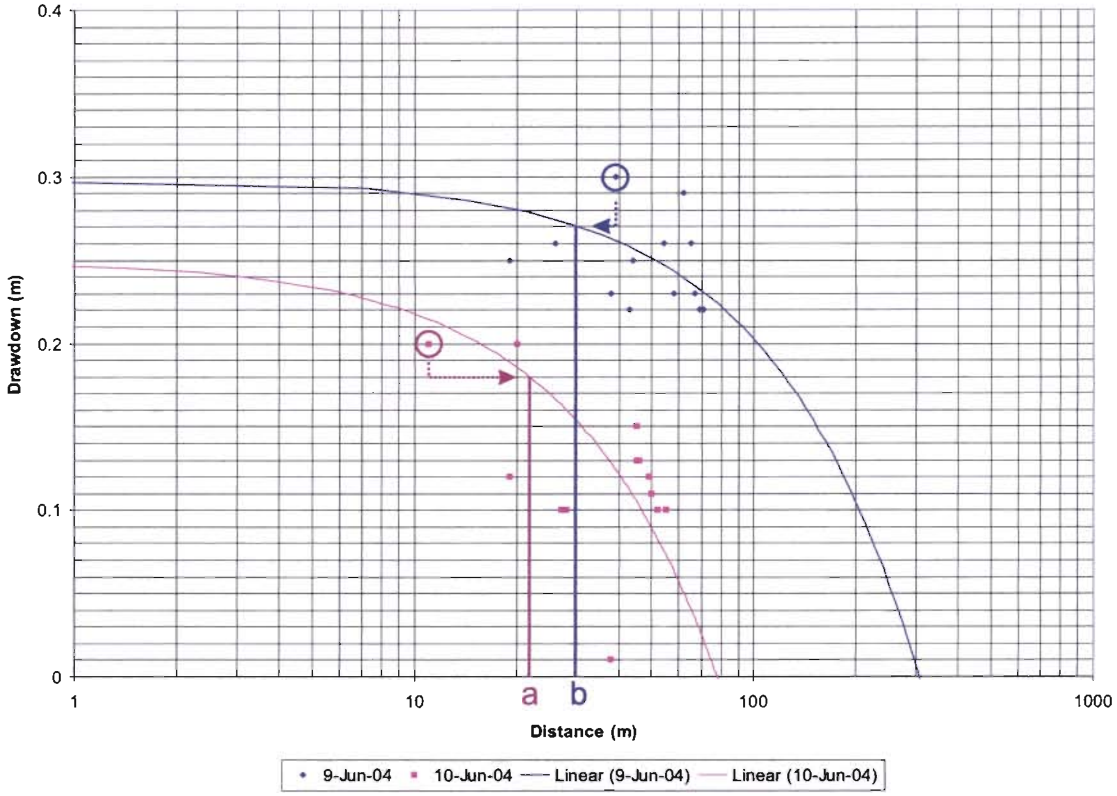


Figure 21b - Semi-log plot of maximum drawdown observed at monitoring wells vs distance from pumping well for pumping tests (Jacob 1940) conducted on 9 and 10 June 2004. Maximum distance is the extents of the pumping well influence. Proposed well design distances are indicated as 22m based on pumping test data from 10 June 2004 (a); and as 25m based on pumping test data from 9 June 2004 (b).

three tests conducted on the 15 June model are outlined in Figures 25, 26 and 27. In all tests, sufficient gradient exists between the injection wells and the pumping well to indicate that in-situ leaching using these distances will work. However, groundwater leakage was detected in tests conducted at the baseline distances of 20, 25 and 30 m indicating that the hydraulic gradient towards the pumping wells was no high enough.

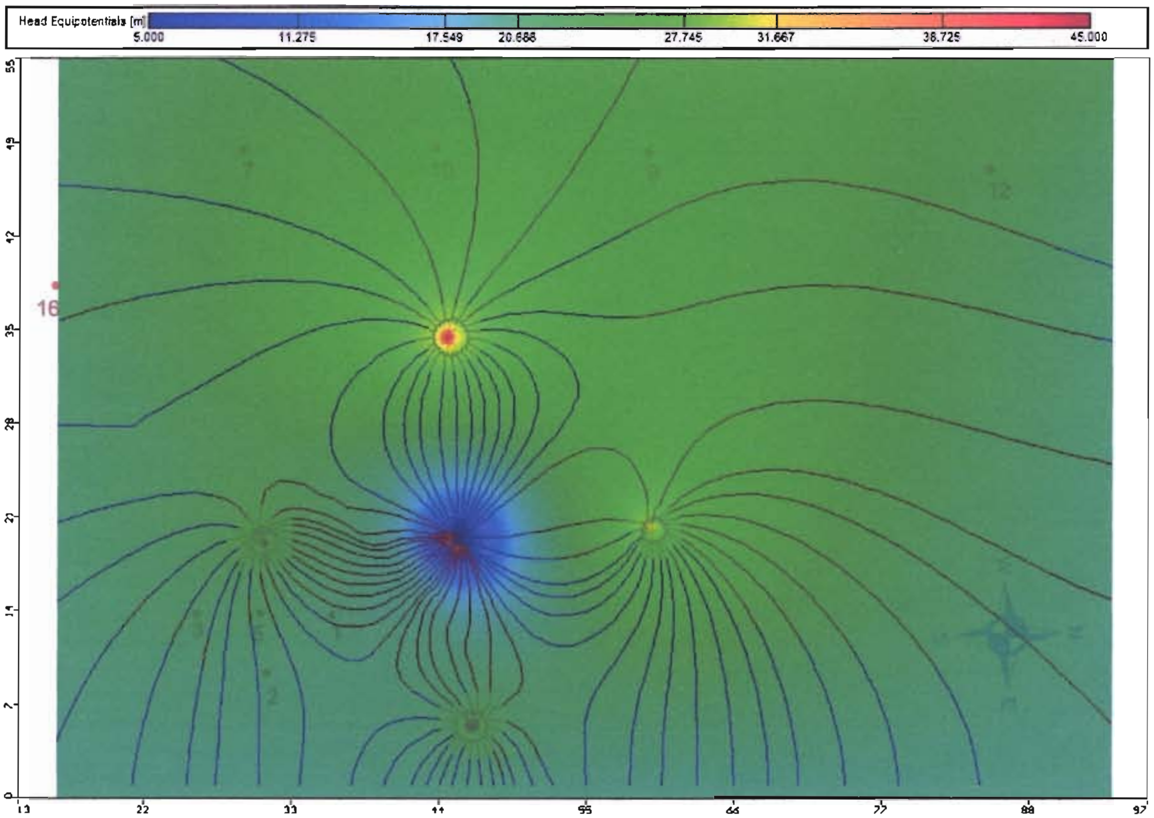


Figure 22 – Hydraulic head results for the 14 June well test cell with injection wells 20m apart.

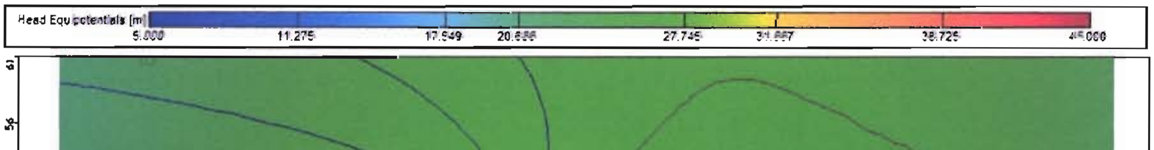


Figure 23 – Hydraulic head results for the 14 June well test cell with injection wells 25m apart.

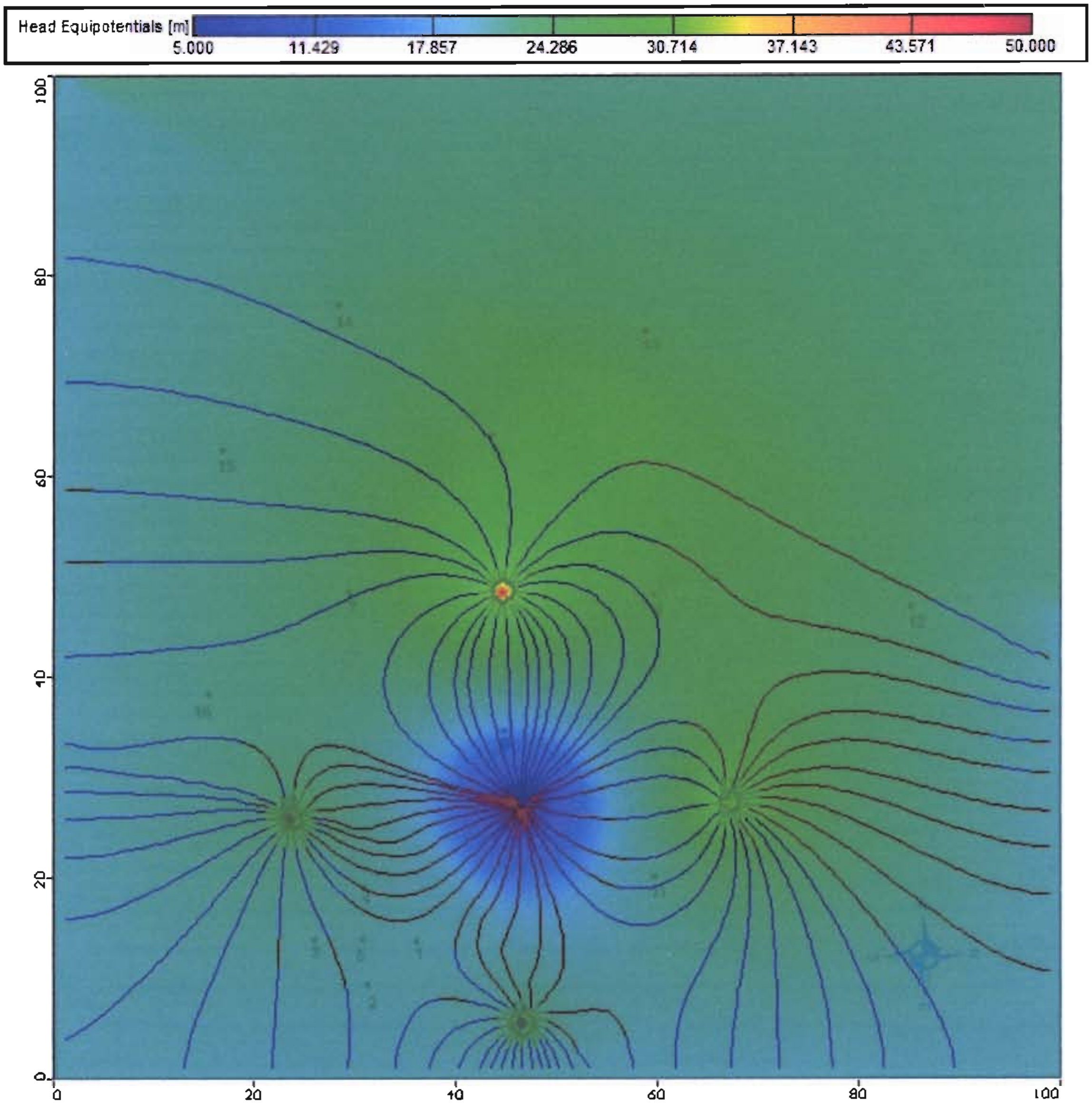


Figure 24 – Hydraulic head results for the 14 June well test cell with injection wells 30m apart.

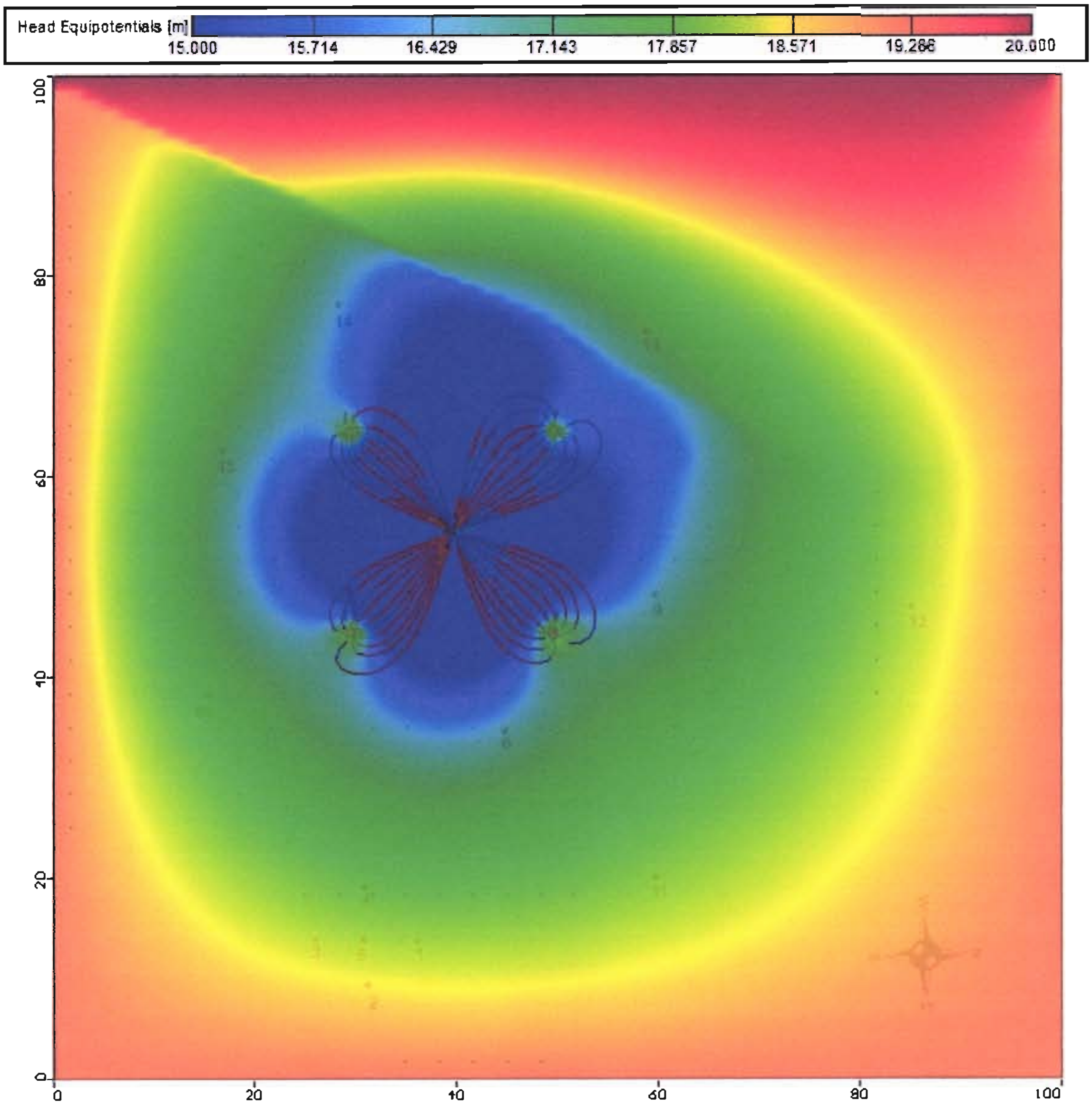


Figure 25 – Hydraulic head results for the 15 June well test cell with injection wells 20m apart.

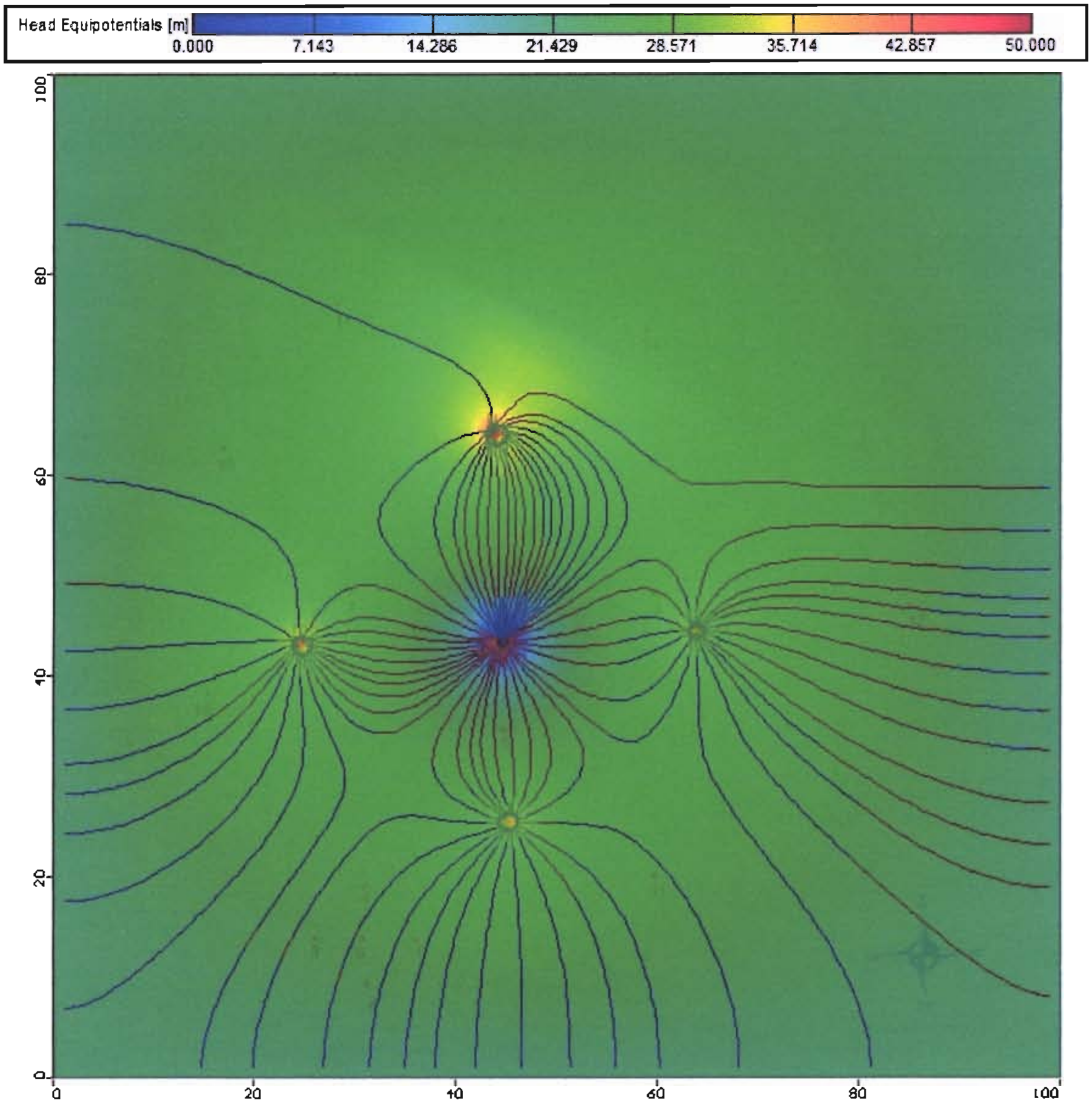


Figure 26 – Hydraulic head results for the 15 June well test cell with injection wells 25m apart.

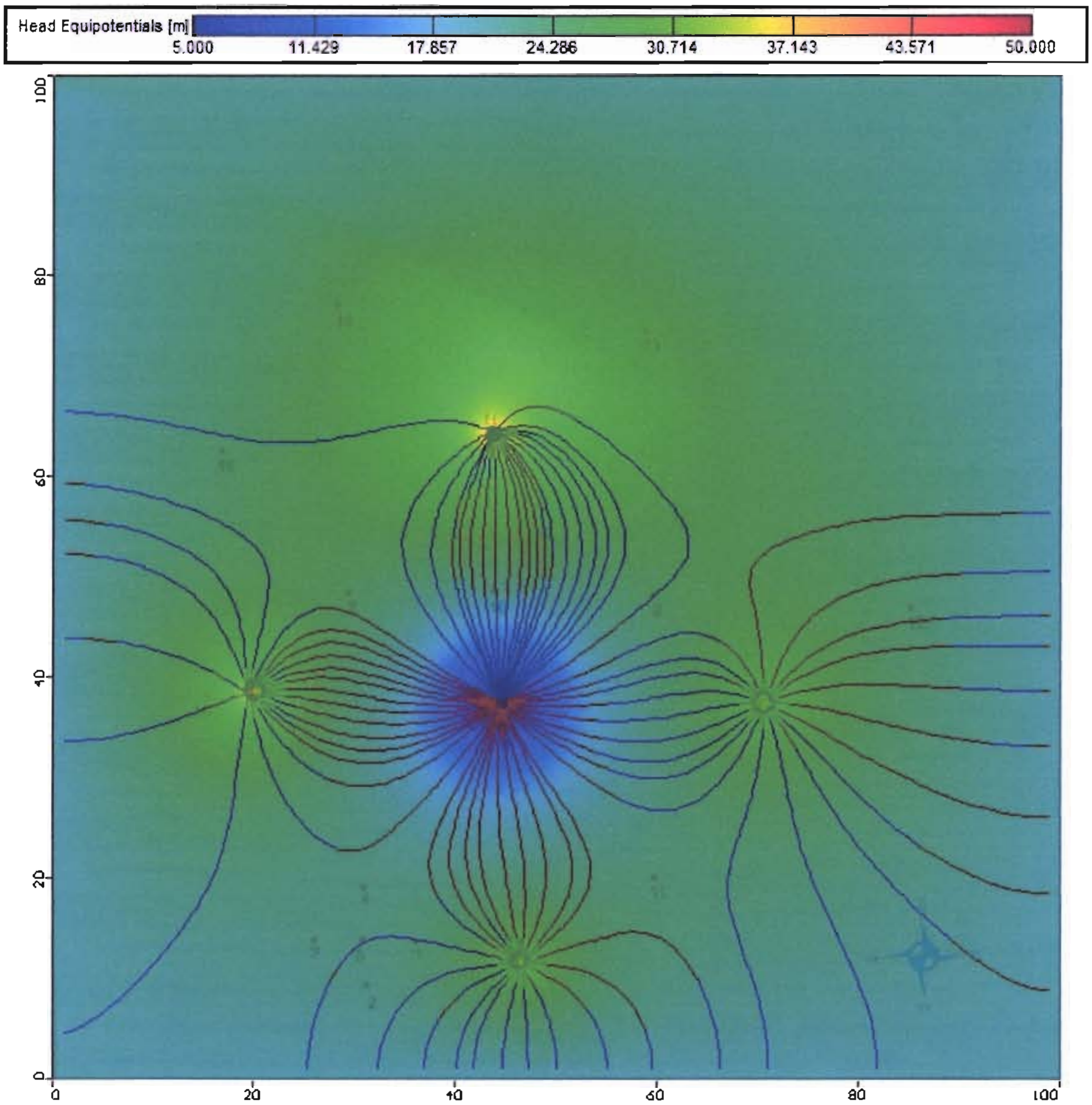


Figure 27 – Hydraulic head results for the 15 June well test cell with injection wells 30m apart.

The last task here is to develop a well field and test it using an expanded model. For this, a 300m square model was built, based on the verified parameters resulting from the 100m model verified with VMP4. Because of the limitations of VMP4, in order for the anisotropy to work in the x and y direction, the well field design model is oriented with the north towards the top right corner of Figures 28 to 31. These figures outline a well field designed with six square injection/pumping cells. Six pumping wells are contained within a matrix of 12 injection wells, which in turn are surrounded by 14 containment wells. These wells are spaced 25m apart in the x and y direction. Around the well field at a distance of 25 m is a ring of monitoring wells. The monitoring wells and the containment wells would in turn become pumping and injection wells when the central part of the well field has been depleted of the required metal. The test was conducted with The total pumping capacity set to 1100 L/min, or to 50 gpm per pumping well. The injection wells were set to 11 gpm except for the corner injection wells which were set to

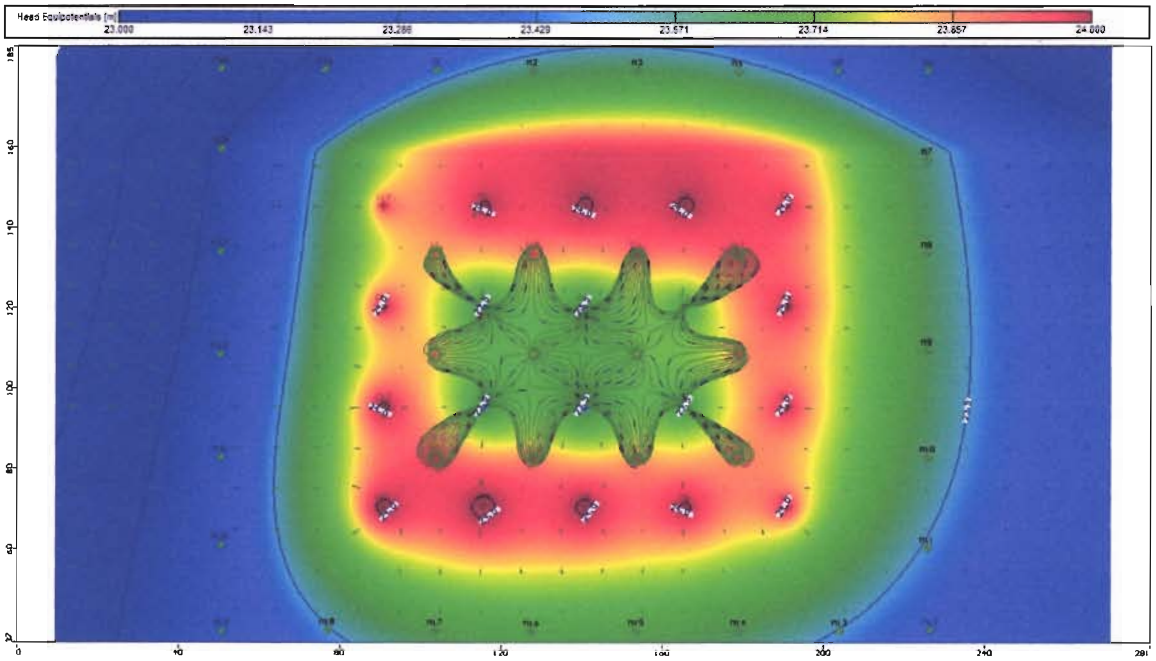


Figure 28 – Results of running the design well field for 24 hours including particle trace and flow from injection well to the pumping wells.

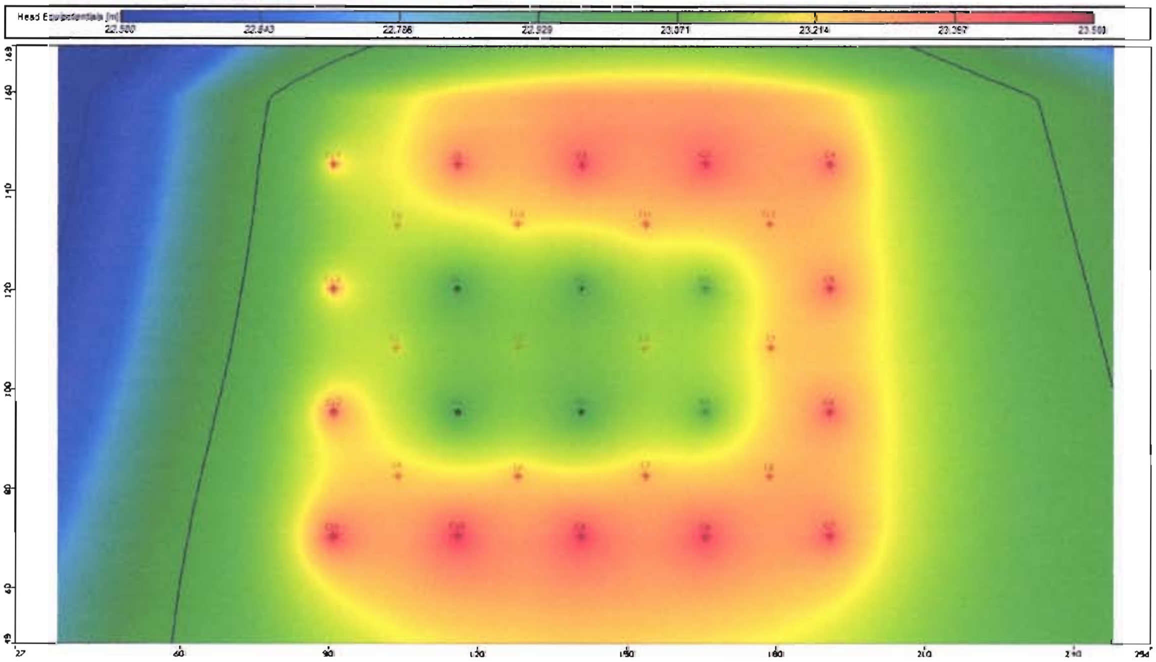


Figure 29 – Hydraulic head results from well design test run indicating the gradient from the containment ring to the pumping wells.

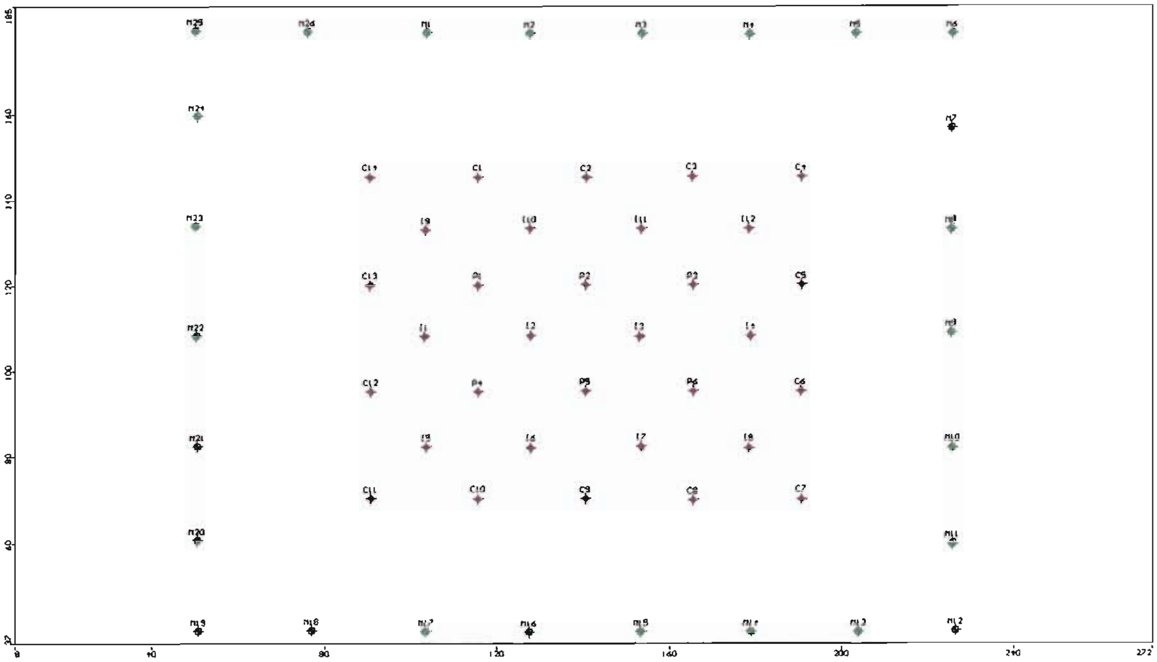


Figure 30 – The design well field indicating containment wells (i.e., C1); the injection wells (i.e., I1); the pumping wells (i.e., P1) and monitoring wells (i.e., M1). Distances are in meters along the x and y-axis.

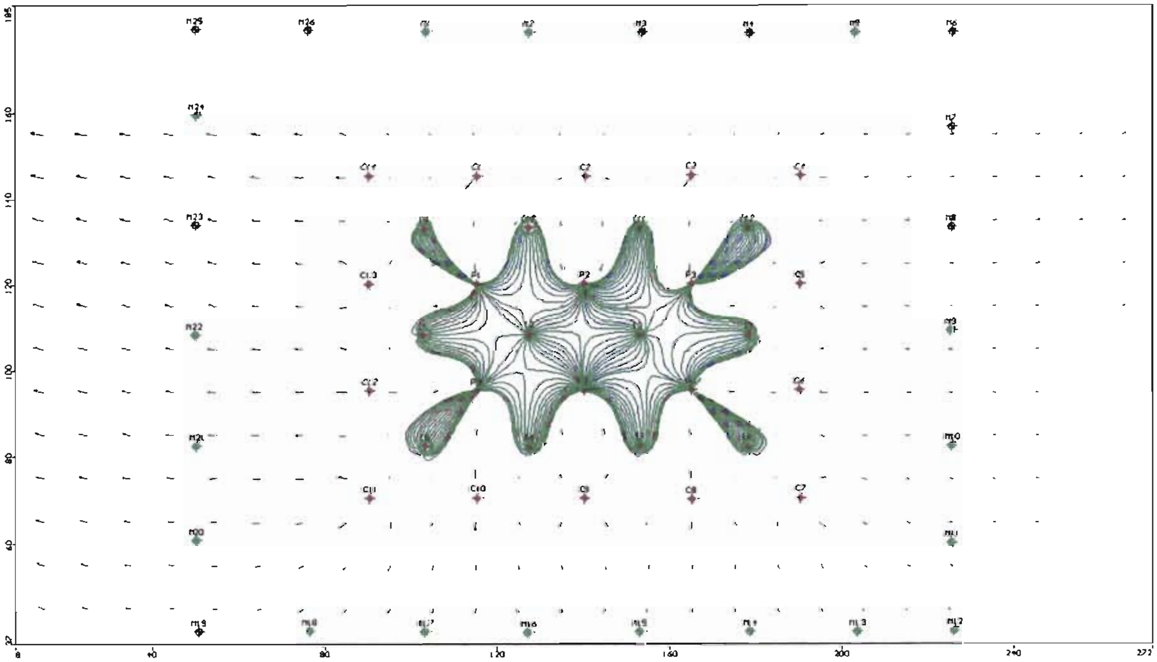


Figure 31 – Coverage of the ore zone with a six-cell well field design with wells at 25m apart in the x and y-directions.

to 2.5 gpm to prevent leakage of the leachate through the line of the containment wells (injection well #9 had to be further reduced to 1 gpm). Because of the anisotropy, the containment wells had to be set to 42 gpm in the x-direction and 40 gpm in the y-direction. As indicated in Figure 31, these settings result in some tunneling at the corners of the well field, but complete coverage occurred within the center of the well field.

Additional monitoring wells must be used in the overburden and within the underlying strata at the extents of the operating well field.

5 Conclusion

A version of the outlined well field design as indicated in this report is a viable option that can be used with the in-situ mining technique proposed for the Yava Silver Mine Site. The well field can be expanded or reduced as required and can be adapted to fit the different permeability zones present at the site.

To be effective the well field must have:

- monitoring wells within the ore zone;
- monitoring wells in the overburden, and within the underlying clay layers;
- containment wells independently capable of injecting air at various pressures, in order to adapt to changing hydrogeologic conditions within the ore zone;
- injection wells capable of independently injecting leachate at variable pressures and injection rates; and,
- high capacity pumps to retrieve the metal laden leachate from the ore zone.

6 References

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Appendix A

to Modeling the Hydrogeologic Properties of the Yava Silver Mine Sandstone

100 m grid topography

x (m)	y (m)	z (masl)	x (m)	y (m)	z (masl)	x (m)	y (m)	z (masl)
35.6123	13.737117	31.7	10.507233	36.346367	32.6	32.676033	15.6457	31.8
30.767467	9.0391	31.67	16.23295	42.952967	32.6	39.869883	21.958667	31.8
25.629	13.883933	31.89	22.692733	50	32.6	48.531867	27.8312	31.8
30.62065	18.728767	31.9	29.44615	55.138467	32.6	56.900217	34.5846	31.8
30.473833	13.737117	31.79	35.6123	60.57055	32.6	62.919567	39.869883	31.8
44.127467	34.731417	32.04	41.484833	65.855833	32.6	67.470783	44.274283	31.8
28.712083	48.678683	32.59	46.476483	70.260233	32.6	72.75605	51.1745	31.8
43.3934	64.094067	32.6	50.734067	74.95825	32.6	76.866833	57.7811	31.8
59.542867	48.38505	32.024	55.285283	79.069033	32.6	80.537167	64.828133	31.8
42.952967	48.678683	32.33	59.102417	84.3543	32.6	83.76705	71.434733	31.8
59.102417	20.0501	31.5	62.625933	90.226833	32.6	86.116067	80.537167	31.8
84.647933	46.6233	31.598	64.681333	95.658933	32.6	87.877817	89.34595	31.8
58.22155	74.224183	32.46	65.855833	99.916517	32.6	88.611883	96.686617	31.8
34.731417	76.426383	33.125	0.3771167	0.0834833	32.2	89.052333	99.7697	31.8
16.379767	62.1855	33.05	3.3133833	5.0751333	32.2	41.1912	0.2303	31.6
14.911633	37.961317	32.5	7.2773333	11.534917	32.2	44.567917	6.5432667	31.6
0.2303	80.243533	34	11.975367	17.8479	32.2	48.8255	13.00305	31.6
3.607	82.739367	34	18.43515	24.01405	32.2	54.844833	18.875583	31.6
8.0114	86.4097	34	24.894933	29.59295	32.2	61.304617	22.83955	31.6
11.82855	89.34595	34	32.969667	35.318667	32.2	69.672983	28.858883	31.6
15.6457	93.016283	34	40.310317	41.044383	32.2	75.3987	34.144167	31.6
19.462833	96.5398	34	46.18285	45.889233	32.2	81.271233	40.897583	31.6
22.98635	100.35695	34	52.2022	52.2022	32.2	88.024633	51.61495	31.6
0.2303	67.470783	33.6	56.459783	57.047033	32.2	90.080017	58.8088	31.6
3.3133833	69.8198	33.6	60.717367	62.1855	32.2	92.135417	67.470783	31.6
7.7177833	72.75605	33.6	64.97495	68.498467	32.2	93.1631	75.545517	31.6
11.82855	76.132767	33.6	68.498467	75.692317	32.2	94.337617	83.1798	31.6
15.352067	79.509467	33.6	70.260233	82.0053	32.2	94.77805	91.9886	31.6
19.756467	82.886167	33.6	71.875183	88.318267	32.2	95.3653	99.916517	31.6
23.720417	86.116067	33.6	72.902867	93.60355	32.2	56.01935	0.3771167	31.4
27.39075	89.492767	33.6	74.077383	99.622883	32.2	58.955617	5.8092	31.4
31.2079	93.60355	33.6	13.149867	0.2303	32	64.681333	12.562617	31.4
33.703733	98.007933	33.6	15.7925	6.1028333	32	69.232533	17.701083	31.4
34.731417	100.06333	33.6	20.1969	11.681733	32	74.664633	21.371417	31.4
0.2303	45.889233	33	24.748117	18.141517	32	80.830783	25.629	31.4
2.5793167	48.238233	33	30.62065	23.133167	32	86.850133	30.327017	31.4
6.9837167	52.2022	33	36.7868	27.978017	32	91.54815	36.19955	31.4
10.65405	55.872533	33	42.072083	32.529217	32	95.218483	43.83385	31.4
15.20525	59.9833	33	49.265933	37.8145	32	98.448383	49.41275	31.4
22.98635	66.00265	33	54.110767	41.77845	32	100.06333	54.844833	31.4
27.537567	69.232533	33	59.249233	47.357367	32	72.75605	0.3771167	31.2
31.79515	72.60925	33	64.3877	52.936267	32	75.105067	4.7815167	31.2
36.346367	75.251883	33	68.938917	60.864183	32	78.7754	10.360417	31.2
39.869883	78.481767	33	72.902867	69.8198	32	85.675633	15.7925	31.2
43.540217	81.711667	33	75.692317	78.18815	32	92.429033	21.665033	31.2
48.38505	87.5842	33	78.18815	86.5565	32	97.127067	25.18855	31.2
51.0277	91.9886	33	79.215833	93.456733	32	99.7697	28.858883	31.2
53.523517	96.099367	33	80.830783	99.7697	32	88.024633	0.0834833	31
54.698017	99.916517	33	23.133167	0.3771167	31.8	90.226833	4.7815167	31
-0.063333	22.252283	32.6	25.335367	5.22195	31.8	94.484417	9.0391	31
4.4878833	28.858883	32.6	28.271633	9.7731667	31.8	98.888817	11.975367	31

Appendix A
to Modeling the Hydrogeologic Properties of the Yava Silver Mine Sandstone
100 m grid top of sandstone

x (m)	y (m)	z (masl)
0.084067	31.60198	22
3.10015	33.11002	22
4.608183	35.07047	22
5.8146	38.08653	22
4.608183	40.801	22
2.798533	43.36465	22
0.084067	45.17428	22
0.234883	23.4586	21.6
3.854167	23.7602	21.6
8.07665	24.21262	21.6
11.39433	24.96663	21.6
15.46602	26.47467	21.6
19.38692	28.58592	21.6
21.79977	32.0544	21.6
20.74413	35.52287	21.6
18.33128	39.74537	21.6
14.8628	44.26947	21.6
10.79112	50	21.6
4.306567	55.57973	21.6
0.536483	58.7466	21.6
0.234883	6.116217	21.2
5.211383	5.211383	21.2
11.09272	4.155767	21.2
19.53772	3.70335	21.2
26.62547	3.70335	21.2
33.26082	4.608183	21.2
37.63412	6.267017	21.2
42.00742	10.3387	21.2
42.76143	16.52165	21.2
40.801	24.21262	21.2
37.48332	32.2052	21.2
33.26082	40.9518	21.2
27.6811	48.94438	21.2
17.87888	57.69097	21.2
9.8863	66.73918	21.2
0.234883	74.88257	21.2
58.44498	0.084067	20.8
59.34982	5.513	20.8
57.99258	13.35477	20.8
55.57973	23.00618	20.8
51.05562	32.2052	20.8
45.9283	41.1026	20.8
38.84055	49.54758	20.8
29.94315	59.0482	20.8
22.25217	67.04078	20.8
14.10878	75.18417	20.8
8.529067	82.57353	20.8
0.536483	92.67737	20.8
71.8665	0.084067	20.4
72.4697	5.6638	20.4
72.92212	12.90237	20.4
72.0173	21.49815	20.4
69.75525	28.28432	20.4
65.08033	34.91967	20.4
55.73053	43.96787	20.4
47.28553	53.61928	20.4
37.78492	62.81828	20.4
29.03833	75.33497	20.4
24.36342	82.42273	20.4
20.89495	90.56612	20.4
19.2361	100.0667	20.4
85.4388	0.234883	20
87.70085	7.62425	20
88.15327	16.52165	20
83.77997	26.47467	20
77.44622	35.82448	20
69.15203	44.42027	20
59.80222	53.01607	20
50	61.76267	20
41.70582	70.96167	20
36.27688	78.35103	20
33.41163	87.55005	20
32.5068	96.14583	20
33.71323	99.61432	20
92.97897	0.084067	19.6
94.93942	6.116217	19.6
96.29665	14.712	19.6
96.14583	25.11743	19.6
93.12977	35.07047	19.6
88.15327	43.21385	19.6
81.0655	51.80963	19.6
72.3189	60.55623	19.6
64.62793	69.60445	19.6
58.5958	78.20023	19.6
55.42892	86.19282	19.6
54.6749	93.73298	19.6
55.27812	99.46352	19.6
99.91593	40.65018	19.2
94.6378	49.54758	19.2
89.6613	59.199	19.2
83.62917	68.69962	19.2
78.65265	78.80345	19.2
76.5414	87.39925	19.2
76.08898	95.09022	19.2
76.5414	99.16192	19.2
100.2175	78.95425	18.8
96.44745	86.04202	18.8
95.24102	94.487	18.8
94.93942	99.91593	18.8

Appendix A
to Modeling the Hydrogeologic Properties of the Yava Silver Mine Sandstone
100 m grid Top of clay - Bottom of sandstone

x (m)	y (m)	z (masl)	x (m)	y (m)	z (masl)	x (m)	y (m)	z (masl)
8.537767	0.18565	-34	14.20527	98.91948	-42	51.04402	18.38132	-48
4.66	4.212567	-34	58.50125	0.782233	-44	47.46453	21.96078	-48
-0.261783	8.090333	-34	54.77263	4.361717	-44	42.9902	26.13683	-48
18.6796	0.036517	-36	50.59658	8.090333	-44	40.3056	29.56717	-48
12.86297	6.300583	-36	46.12223	11.96808	-44	37.17355	34.3398	-48
7.195467	10.92408	-36	41.94618	16.29328	-44	33.59408	38.51585	-48
0.18565	19.57447	-36	38.51585	19.27618	-44	31.20777	43.28848	-48
29.26888	0.036517	-38	33.89237	21.96078	-44	30.16375	48.50855	-48
22.25908	6.0023	-38	31.3569	26.28598	-44	28.97058	54.47435	-48
15.10013	14.3544	-38	28.07572	30.3129	-44	28.07572	59.39612	-48
8.388617	21.51335	-38	24.79453	35.23467	-44	27.92658	65.51105	-48
3.914267	28.22487	-38	22.55737	39.70902	-44	28.22487	72.22257	-48
0.3348	35.38382	-38	19.87277	44.48165	-44	28.82145	77.14435	-48
43.28848	0.036517	-40	18.23217	51.78973	-44	29.86547	83.70672	-48
35.9804	5.85315	-40	17.48645	57.90467	-44	31.50605	89.52335	-48
30.16375	10.92408	-40	17.03902	63.57217	-44	33.14665	95.04172	-48
24.3471	16.29328	-40	16.88987	69.83625	-44	34.3398	99.36692	-48
19.87277	20.76763	-40	17.78473	75.6529	-44	78.48665	0.3348	-50
14.95098	25.54027	-40	18.23217	82.81185	-44	74.60888	3.16855	-50
12.11723	30.46203	-40	19.57447	89.22507	-44	69.53797	7.046317	-50
9.283483	35.08553	-40	19.72362	93.84855	-44	65.95848	10.47665	-50
7.3446	40.45473	-40	21.06592	99.21777	-44	61.48415	14.05612	-50
5.704017	45.6748	-40	65.06362	0.3348	-46	56.71152	18.53047	-50
5.256583	52.23717	-40	61.78243	3.3177	-46	53.28118	21.96078	-50
5.107433	58.6504	-40	57.60638	6.748017	-46	48.50855	26.43513	-50
5.405717	64.91447	-40	53.28118	9.581783	-46	44.77993	31.05862	-50
5.256583	71.92428	-40	50	13.75783	-46	41.94618	35.9804	-50
6.300583	77.74092	-40	47.0171	16.74072	-46	38.665	41.05132	-50
7.046317	83.85585	-40	43.88507	19.57447	-46	37.17355	46.12223	-50
7.49375	90.26908	-40	40.3056	23.60138	-46	35.23467	53.13203	-50
7.49375	95.48915	-40	36.72612	27.33	-46	34.48895	59.9927	-50
7.941183	99.96348	-40	32.55007	32.55007	-46	34.3398	66.55507	-50
51.04402	0.633083	-42	29.56717	37.3227	-46	34.48895	73.26658	-50
47.76283	2.870267	-42	27.33	42.24447	-46	35.38382	78.63578	-50
41.94618	8.239467	-42	25.09283	48.21027	-46	37.0244	84.89987	-50
36.57697	12.86297	-42	23.0048	55.07092	-46	38.665	90.41822	-50
30.61118	18.53047	-42	23.0048	61.18585	-46	40.45473	95.34	-50
27.33	21.81165	-42	22.70652	69.38882	-46	41.3496	99.96348	-50
23.60138	24.79453	-42	22.70652	75.3546	-46	89.6725	0.782233	-52
20.0219	29.56717	-42	23.45223	81.0221	-46	83.11013	5.256583	-52
16.88987	35.23467	-42	24.49625	85.94388	-46	75.05632	10.62578	-52
13.90697	41.05132	-42	25.54027	90.71652	-46	68.04652	16.59158	-52
12.56467	47.16625	-42	26.73342	96.23487	-46	61.78243	21.21507	-52
11.22237	54.92178	-42	27.62828	99.36692	-46	56.26408	26.88257	-52
11.37152	62.37902	-42	72.22257	0.18565	-48	50.74572	34.04152	-52
11.37152	68.79223	-42	68.79223	3.0194	-48	46.42053	41.05132	-52
12.26638	76.84605	-42	65.36192	6.449733	-48	43.13935	48.80685	-52
12.71382	83.55757	-42	61.48415	9.134333	-48	40.75303	57.45723	-52
13.60868	89.37422	-42	58.3521	11.6698	-48	40.90217	67.30078	-52
13.45953	94.14685	-42	54.62348	14.80183	-48	42.09533	76.39862	-52

Appendix A
to Modeling the Hydrogeologic Properties of the Yava Silver Mine Sandstone
100 m grid Top of clay - Bottom of sandstone - continued

x (m)	y (m)	z (masl)
43.28848	86.39132	-52
44.77993	93.69942	-52
46.42053	99.6652	-52
98.91948	0.3348	-54
94.44513	3.615983	-54
89.07592	7.792033	-54
81.91697	12.26638	-54
77.14435	17.03902	-54
70.28368	21.36422	-54
63.57217	27.62828	-54
57.15895	35.08553	-54
51.93888	41.94618	-54
49.25428	49.25428	-54
46.71882	57.60638	-54
46.12223	64.91447	-54
46.12223	72.96828	-54
46.86797	81.3204	-54
48.06112	88.47935	-54
49.25428	93.55027	-54
50.74572	99.81435	-54
99.6652	6.897167	-56
95.19085	10.3275	-56
89.97078	14.20527	-56
83.40842	18.23217	-56
79.08322	21.6625	-56
73.56487	24.94368	-56
67.74822	31.05862	-56
62.37902	37.17355	-56
58.3521	43.58678	-56
55.36922	50	-56
53.13203	56.41322	-56
51.6406	64.3179	-56
51.49145	72.96828	-56
51.78973	81.0221	-56
52.9829	88.47935	-56
54.17605	95.63828	-56
55.51835	99.6652	-56
99.81435	14.95098	-58
95.48915	17.03902	-58
88.92678	21.06592	-58
86.39132	22.85565	-58
83.55757	25.39112	-58
80.42553	27.62828	-58
76.54777	31.95348	-58
72.22257	35.38382	-58
67.59908	42.24447	-58
64.91447	46.86797	-58
62.6773	51.93888	-58
59.6944	57.60638	-58
58.20297	64.61618	-58

x (m)	y (m)	z (masl)
57.15895	72.37172	-58
57.3081	78.18835	-58
57.3081	84.30328	-58
58.3521	89.6725	-58
59.39612	94.14685	-58
60.14185	99.51605	-58
99.96348	20.91678	-60
95.48915	23.15395	-60
90.71652	25.39112	-60
85.3473	30.16375	-60
79.9781	34.93638	-60
75.6529	39.70902	-60
72.37172	44.77993	-60
68.79223	52.38632	-60
65.95848	60.44013	-60
64.61618	67.0025	-60
63.12473	73.41572	-60
62.52815	81.3204	-60
62.37902	88.03192	-60
63.72132	93.25198	-60
65.06362	99.51605	-60
100.1126	38.51585	-62
95.19085	42.84105	-62
89.22507	48.80685	-62
86.09303	55.6675	-62
81.76783	64.16875	-62
80.27638	72.37172	-62
78.78493	80.57467	-62
78.18835	88.62848	-62
78.48665	95.34	-62
79.08322	99.21777	-62
99.96348	59.6944	-64
95.78743	67.30078	-64
93.55027	75.20547	-64
92.6554	83.85585	-64
91.90967	91.90967	-64
92.20797	97.12973	-64
92.3571	99.21777	-64

Appendix A
to Modeling the Hydrogeologic Properties of the Yava Silver Mine Sandstone
100 m grid Bottom of clay

x (m)	y (m)	z (masl)
6.588217	0.321033	-54
5.36535	3.3782	-54
4.753917	8.8811	-54
4.29535	14.84257	-54
3.225333	22.4855	-54
2.46105	31.04557	-54
1.69675	40.67563	-54
0.321033	52.75145	-54
22.63835	-0.13755	-60
20.49833	9.339667	-60
18.66403	25.08408	-60
17.89975	39.14705	-60
16.67688	54.73862	-60
15.60687	70.9416	-60
10.86825	85.31028	-60
7.0468	96.01037	-60
5.36535	99.83183	-60
38.38277	0.62675	-64
35.93703	12.70255	-64
32.87987	25.3898	-64
31.35128	38.2299	-64
29.21127	51.37573	-64
27.37697	64.21583	-64
24.77837	74.45733	-64
22.02692	84.39313	-64
18.0526	93.41178	-64
15.1483	99.83183	-64
61.0058	0.015317	-70
56.11433	5.671067	-70
53.21003	11.93827	-70
50.91715	18.66403	-70
48.31855	26.45982	-70
45.71997	33.64415	-70
42.96852	41.59278	-70
40.8285	50.30572	-70
38.68848	58.40722	-70
36.3956	67.273	-70
33.4913	77.5145	-70
31.50413	86.53315	-70
29.66983	94.17607	-70
28.14125	99.2204	-70
82.25312	0.015317	-74
75.22163	5.671067	-74
69.41302	12.09112	-74
64.06297	20.6512	-74
58.1015	30.43413	-74
53.51575	39.60563	-74
47.70712	52.44573	-74
44.4971	61.46438	-74
41.43993	70.78873	-74
38.07705	81.03025	-74
36.08988	91.27177	-74
34.10273	99.67897	-74
99.52612	1.085317	-78
96.16323	3.683917	-78
92.18892	7.0468	-78
87.60317	11.32683	-78
82.55883	16.2183	-78
77.20878	20.95692	-78
72.7759	26.76553	-78
69.1073	32.26843	-78
65.74442	37.61847	-78
61.92295	43.57995	-78
58.25435	50.45858	-78
57.79578	50.30572	-78
53.97432	59.01865	-78
49.69428	68.80158	-78
47.70712	76.29163	-78
45.41425	85.00457	-78
42.96852	94.17607	-78
41.74565	99.52612	-78
99.52612	7.658233	-80
96.16323	10.10397	-80
91.57748	13.77257	-80
86.68602	17.74688	-80
81.1831	23.09693	-80
75.22163	29.36412	-80
71.40017	34.40845	-80
68.03728	39.60563	-80
64.36868	46.17853	-80
60.85295	53.21003	-80
58.1015	58.71293	-80
55.19718	65.74442	-80
52.90432	73.38733	-80
50.45858	80.26595	-80
48.77713	86.53315	-80
46.63712	94.32893	-80
45.10853	99.52612	-80
99.67897	24.01408	-80
94.94037	27.9884	-80
90.50747	33.03272	-80
86.3803	37.61847	-80
83.17027	42.05137	-80
79.19595	46.78997	-80
75.6802	52.90432	-80
72.0116	58.25435	-80
69.1073	64.21583	-80
66.5087	70.48302	-80
64.6744	74.91592	-80
62.22867	81.48882	-80
60.08865	86.68602	-80
58.25435	92.34177	-80
56.57292	99.06753	-80
100.1376	42.81565	-82
96.01037	47.70712	-82
91.27177	52.90432	-82
88.06173	58.1015	-82
85.46315	61.92295	-82
82.55883	67.42585	-82
79.65452	72.92875	-82
77.36165	77.36165	-82
74.6102	81.9474	-82
73.23447	87.75602	-82
71.09445	92.03605	-82
69.71873	96.6218	-82
69.26015	99.2204	-82
99.52612	86.3803	-82
97.08038	90.35462	-82
95.09322	95.39893	-82
94.17607	99.67897	-82
99.98468	63.29868	-84
95.5518	69.41302	-84
91.8832	75.52735	-84
88.82603	82.10025	-84
85.76887	88.67317	-84
83.7817	94.32893	-84
82.10025	98.91468	-84

Appendix B
Packer Testing
from June to November 2004

The packer testing data and results of the hydraulic conductivity calculations are listed in tables in parts B.1.1 to B.1.8 of this appendix.

Bases for Hydraulic Conductivity Computations

$$K = \frac{Q}{2\pi L\ H} \cdot Ln\frac{L}{r}$$

K → Hydraulic Conductivity [cm/s]
Q → Rate of take [Igpm]
H → Applied Head [ρ/γ_w]
ρ → mass density of water [Pa]
γ_w → specific weight of water [N/m^3]
L → Length of packer [L]
r → Radius of hole [L]
p → injection pressure [psi]

p → psi → [lb/in^2] x (6.895×10^3) = [N/m^2] or [Pa]
γ_w → [lb/in^3] x (9.8×10^3) = [N/m^3]
gpm → Q x 7.576×10^{-5} [m^3/s]

Based on the length of the packer, two different calculations were performed. Double packers with a spacing of 1.5m were used on all tests except the one performed on the 16th August 2004 in Well 8, where one packer was used. The length for this test became the distance from the single packer to the bottom of the hole.

(1) L_1 = 5 ft = 1.524 m
(2) L_2 = 42.5 m
r = 3 in = 0.25 ft = 0.0762 m

Test Type (1)

$$K = \frac{\left(Q \times 7.576 \times 10^{-5}\, m^3 \cdot s^{-1}\right)}{2\pi(1.524m)\left(\dfrac{p \times 6.895 \times 10^3\, N \cdot m^{-2}}{9.8 \times 10^3\, N \cdot m^{-3}}\right)} \cdot Ln\left(\frac{1.524}{0.0762}\right)$$

$$K = \frac{\left(7.576 \times 10^{-5} \times [Q\, m^3 \cdot s^{-1}] \times 6.324\right)}{p \times \left(9.576m \times 0.7036m\right)}$$

$$K = \frac{Q \times 2.2696 \times 10^{-4}\, m^3 \cdot s^{-1}}{p \times 6.74m^2}$$

$$K = \left(3.369 \times 10^{-5}\right)\frac{Q}{p} m \cdot s^{-1} = 3.369 \times 10^{-3} \frac{Q}{p} cm \cdot s^{-1}$$

Test Type (2)

$$K = \frac{\left(Q \times 7.576 \times 10^{-5}\, m^3 \cdot s^{-1}\right)}{2\pi(42.5m)\left(\dfrac{p \times 6.895 \times 10^3\, N \cdot m^{-2}}{9.8 \times 10^3\, N \cdot m^{-3}}\right)} \cdot Ln\left(\frac{42.5}{0.0762}\right)$$

$$K = \frac{\left(7.576 \times 10^{-5} \times [Q\, m^3 \cdot s^{-1}] \times 6.324\right)}{p \times \left(267.035m \times 0.7036m\right)}$$

$$K = \frac{Q \times 4.79 \times 10^{-4}\, m^3 \cdot s^{-1}}{p \times 187.88m^2}$$

$$K = \left(2.55 \times 10^{-6}\right)\frac{Q}{p} m \cdot s^{-1} = 2.55 \times 10^{-4} \frac{Q}{p} cm \cdot s^{-1}$$

Appendix B.1.1
Packer Test Well 9 - 22 June 2004

Test #	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)
1	0924 - 0934	71.5	21.7932	2000	157	0	0	0.00E+00
2	0943 - 0953	81.2	24.74976	2000	156	54	0.42	9.07E-06
3	1012 - 1022	90.9	27.70632	2000	157	0	0	0.00E+00
4	1028 - 1038	100.5	30.6324	2000	157	6	0.046666667	1.00E-06
5	10415 - 1055	110	33.528	2000	157	6	0.046666667	1.00E-06
6	1101 - 1111	119.7	36.48456	2000	157	0	0	0.00E+00
7	1118 - 1128	129.25	39.3954	2000	157	6	0.046666667	1.00E-06
8	1136 - 1146	138.9	42.33672	2000	157	12	0.093333333	2.00E-06
9	1153 - 1203	148.5	45.2628	2000	157	6	0.046666667	1.00E-06
10	1212 - 1222	158.1	48.18888	2000	158	6	0.046666667	9.95E-07
11	1228 - 1238	167.25	50.9778	2000	158	6	0.046666667	9.95E-07
12	1244 - 1254	176.9	53.91912	2000	157	9	0.07	1.50E-06
13	1300 - 1310	186.5	56.8452	2000	157	6	0.046666667	1.00E-06
14	1317 - 1327	190.67	58.116216	2000	158	6	0.046666667	9.95E-07
15	1332 - 1342	196.1	59.77128	2000	158	3	0.023333333	4.97E-07
16	1347 - 1357	201.67	61.469016	2000	158	6	0.046666667	9.95E-07
17	1402 - 1412	205.75	62.7126	2000	158	6	0.046666667	9.95E-07
18	1418 - 1428	210.8	64.25184	2000	158	6	0.046666667	9.95E-07
19	1432 - 1442	215.4	65.65392	2000	158	9	0.07	1.49E-06
20a	1448 - 1458	220.3	67.14744	2000	141	846	6.58	1.57E-04
20b	1500 - 1505	220.3	67.14744	2000	100	612	4.76	1.60E-04
20c	1507 - 1510	220.3	67.14744	2000	60	510	3.966666667	2.23E-04
20d	1512 - 1520	220.3	67.14744	2000	20	288	2.24	3.77E-04
21	1536 - 1546	224.95	68.56476	2000	154	54	0.42	9.19E-06
22	1600 - 1610	230.75	70.3326	2000	156	12	0.093333333	2.02E-06
23	1615 - 1625	234.6	71.50608	2000	157	24	0.186666667	4.01E-06
24	1630 - 1640	239.7	73.06056	2000	156	12	0.093333333	2.02E-06
25	1644 - 1654	244.2	74.43216	2000	156	6	0.046666667	1.01E-06
26a	1659 - 1709	249.9	76.16952	2000	125	1344	10.45333333	2.82E-04
26b	1710 - 1720	249.9	76.16952	2000	60	858	6.673333333	3.75E-04
27	1726 - 1734	253.9	77.38872	2000	156	0	0	0.00E+00
28	1738 - 1743	259.1	78.97368	2000	155	0	0	0.00E+00
29	1748 - 1753	263.5	80.3148	2000	155	0	0	0.00E+00

Appendix B.1.2
Packer Test Well 10 - 18 & 21 June 2004

Test #	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)
1	1203 - 1218	71.25	21.717	1500	150	9	0.07	1.57E-06
2	1312 - 1322	81.25	24.765	1200	156	24	0.186666667	4.03E-06
3	1339 - 1349	90.75	27.6606	1200	153	240	1.866666667	4.11E-05
4	1400 - 1410	100.5	30.6324	1200	156	162	1.26	2.72E-05
5	1417 - 1427	109.9	33.49752	1200	155	150	1.166666667	2.54E-05
6	1437 - 1447	119.5	36.4236	1200	157	0	0	0.00E+00
7	1457 - 1507	129.1	39.34968	1200	157	24	0.186666667	4.01E-06
8a	1525 - 1535	138.75	42.291	1200	157	10	0.077777778	1.67E-06
8b	1535 - 1537	138.75	42.291	1200	120	6	0.046666667	1.31E-06
9	1548 - 1550	148.4	45.23232	1200	65	0	0	0.00E+00
10	1602 - 1612	153.2	46.69536	2000	157	6	0.046666667	1.00E-06
11	1637 - 1647	164	49.9872	2000	156	30	0.233333333	5.04E-06
Weekend								
12	0931 - 0941	172.7	52.63896	2000	156	0	0	0.00E+00
13	0955 - 1005	181.9	55.44312	2000	157	0	0	0.00E+00
14	1012 - 1022	186.9	56.96712	2000	156	0	0	0.00E+00
15	1030 - 1040	191.75	58.4454	2000	157	6	0.046666667	1.00E-06
16	1050 - 1100	196.5	59.8932	2000	156	6	0.046666667	1.01E-06

Appendix B.1.3
Packer Test Well 6 - 7 to 9 July 2004

Test #	Date	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)
1	7-Jul-04	1044 - 1047	71.5	21.7932	2100	70	3200	24.88888889	1.20E-03
1a		1101 - 1111	69.5	21.1836	2100	75	2370	18.43333333	8.28E-04
1b		1113 - 1121	69.5	21.1836	2100	30	390	3.033333333	3.41E-04
2		1130 - 1140	81.1	24.71928	2000	90	2190	17.03333333	6.38E-04
2a		1242 - 1251	81.1	24.71928	2000	50	780	6.066666667	4.09E-04
3		1300 - 1311	90.7	27.64536	2000	95	1920	14.93333333	5.30E-04
3a		1311 - 1321	90.7	27.64536	2000	50	1260	9.8	6.60E-04
4		1328 - 1338	100.3	30.57144	2000	62	2670	20.76666667	1.13E-03
4a		1458 - 1508	100.3	30.57144	3000	40	2118	16.47333333	1.39E-03
5		1515 - 1525	110	33.528	3100	156	0	0	0.00E+00
6		1543 - 1554	119.6	36.45408	2900	150	264	2.053333333	4.61E-05
6a		1554 - 1604	119.6	36.45408	2900	100	180	1.4	4.72E-05
7	8-Jul-04	0937 - 0947	129.25	39.3954	2000	9	3120	24.26666667	9.08E-03
7a		0947 - 0957	129.5	39.4716	3000	5	2910	22.63333333	1.52E-02
8		1049 - 1059	134.7	41.05656	3000	157	6	0.046666667	1.00E-06
9		1120 - 1140	138.9	42.33672	2000	157	6	0.046666667	1.00E-06
10		11425 - 1152	144.1	43.92168	2000	156	30	0.233333333	5.04E-06
11		1158 - 1208	148.5	45.2628	2000	156	0	0	0.00E+00
12		1229 - 1238	153.75	46.863	2000	152	186	1.446666667	3.21E-05
12a		1239 - 1248	153.75	46.863	2000	100	120	0.933333333	3.14E-05
13		1255 - 1305	158.1	48.18888	2000	156	6	0.046666667	1.01E-06
14		1312 - 1321	163.35	49.78908	2000	7	3300	25.66666667	1.24E-02
14a		1322 - 1331	163.35	49.78908	2000	3	3030	23.56666667	2.65E-02
15		1435 - 1444	167.75	51.1302	2000	98	1878	14.60666667	5.02E-04
15a		1445 - 1454	167.75	51.1302	2000	50	1440	11.2	7.55E-04
16		1504 - 1505	173	52.7304	2000	156	0	0	0.00E+00
17		1519 - 1529	177.35	54.05628	2000	154	18	0.14	3.06E-06
18		1536 - 1546	182.6	55.65648	2000	155	0	0	0.00E+00
19		1551 - 1607	187	56.9976	2000	154	12	0.093333333	2.04E-06
20	9-Jul-04	0855 - 0904	192.2	58.58256	2000	112	1626	12.64666667	3.80E-04
20a		0905 - 0914	192.2	58.58256	2000	50	1092	8.493333333	5.72E-04
21		0919 - 0928	196.61	59.926728	2000	125	1266	9.846666667	2.65E-04
21a		0929 - 0938	196.6	59.92368	2000	60	912	7.093333333	3.98E-04
22		0948 - 0958	201.9	61.53912	2000	154	30	0.233333333	5.10418E-06
23		1002 - 1012	206.2	62.84976	2000	155	0	0	0
24		1023 - 1032	211.5	64.4652	2000	146	390	3.033333333	6.99902E-05
25		1038 - 1047	215.9	65.80632	2000	148	288	2.24	5.09866E-05
26		1051 - 1101	221.1	67.39128	2000	155	0	0	0
27a		1142 - 1152	223.1	68.00088	2000	155	0	0	0

Appendix B.1.4
Packer Test Well 7 - 12 & 13 July 2004

Test #	Date	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)
1	12-Jul-04	1423 - 1434	71.5	21.7932	2000	105	1680	13.06666667	4.19E-04
1a		1434 - 1444	71.5	21.7932	2000	50	588	4.573333333	3.08E-04
1b		1444 - 1449	71.5	21.7932	2000	75	1008	7.84	3.52E-04
2		1500 - 1509	81.1	24.71928	2000	96	1722	13.39333333	4.70E-04
2a		1510 - 1519	81.1	24.71928	2000	50	660	5.133333333	3.46E-04
3		1605 - 1615	90.75	27.6606	2000	153	48	0.373333333	8.22E-06
4		1622 - 1631	100.35	30.58668	2000	120	1404	10.92	3.07E-04
4a		1632 - 1641	100.35	30.58668	2000	60	960	7.466666667	4.19E-04
5	13-Jul-04	0835 - 0845	110	33.528	2000	155	6	0.046666667	1.01E-06
6		0853 - 0903	119.6	36.45408	2000	155	6	0.046666667	1.01E-06
7		0913 - 0923	129.2	39.38016	2000	153	192	1.493333333	3.29E-05
8		0932 - 0941	134.5	40.9956	2000	144	426	3.313333333	7.75E-05
9		0948 - 0958	138.9	42.33672	2000	154	6	0.046666667	1.02E-06
10		1010 - 1018	144.1	43.92168	2000	154	0	0	0.00E+00
10a		1018 - 1020	144.1	43.92168	2000	11	2100	16.33333333	5.00E-03
11		1032 - 1100	148.5	45.2628	2000	154	12	0.093333333	2.04E-06
12		1051 - 1100	153.75	46.863	2000	125	1164	9.053333333	2.44E-04
12a		1102 - 1110	153.75	46.863	2000	60	696	5.413333333	3.04E-04
13		1229 - 1239	153.75	46.863	2000	152	186	1.446666667	3.21E-05

Lost packer at 165 ft

Appendix B.1.5

Packer Test Well 10 - 12 & 13 August 2004

Test #	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)
1	1659 - 1709	200.1	60.99048	2000	155	18	0.14	3.04E-06
2	1726 - 1735	205	62.484	2000	155	0	0	0.00E+00
3	0911 - 0923	209.9	63.97752	2000	155	6	0.046666667	1.01E-06
3a	0958 - 1007	215	65.532	2000	155	0	0	0.00E+00
4	1015 - 1024	220	67.056	2000	150	258	2.006666667	4.51E-05
5	1040 - 1049	224.9	68.54952	2000	150	252	1.96	4.40E-05
6	1101 - 1109	230.4	70.22592	2000	155	0	0	0.00E+00
7	1116 - 1125	234.5	71.4756	2000	154	54	0.42	9.19E-06
8a	1135 - 1144	240	73.152	2000	137	912	7.093333333	1.74E-04
8b	1145 - 1154	240	73.152	2000	60	474	3.686666667	2.07E-04
9a	1200 - 1209	244.1	74.40168	2000	139	792	6.16	1.49E-04
9b	1210 - 1219	244.1	74.40168	2000	60	474	3.686666667	2.07E-04
10	1255 -1304	249.67	76.099416	2000	155	36	0.28	6.09E-06
11	1313 - 1322	253.8	77.35824	2000	155	6	0.046666667	1.01E-06
12	1332 - 1333	259.3	79.03464	2000	155	6	0.046666667	1.01E-06
13	1347 - 1358	263.4	80.28432	2000	155	0	0	0.00E+00
14	1410 - 1419	268.9	81.96072	2000	155	0	0	0.00E+00
15	1425 - 1432	273	83.2104	2000	148	154	1.197777778	2.73E-05

Appendix B.1.6
Packer Test Well 8 - 16 August 2004

Test #	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)	
1	1121 - 1220	129.7	39.53256	2000	138	716	5.568888889	6.92E-05	
	1220 - 1240	129.7	39.53256	2000	138	786	6.113333333	7.59E-05	
	1240 - 1250	129.7	39.53256	2000	138	630	4.9	6.09E-05	
	1250 - 1300	129.7	39.53256	2000	138	774	6.02	7.48E-05	
					138	726.5	5.650555556	7.02E-05	**Ave**

Appendix B.1.7
Packer Test Well 5 - 4 November 2004

Test #	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)
1	1357 - 1407	148	45.1104	2000	142.5	6	0.046666667	1.10E-06
2	1412 - 1422	153	46.6344	2000	142.5	2094	16.28666667	3.85E-04

Appendix B.1.8
Packer Test Well 8 - 8 & 9 November 2004

Test #	Date	Time	Depth (ft)	Depth (m)	packer (psi)	Pump (psi)	Take (mm/hr)	Take (gpm)	K (cm/s)
1	8-Nov-04	1353 - 1403	128.7	39.22776	2000	156	12	0.093333333	2.02E-06
2		1413 - 1423	133.9	40.81272	2000	156	6	0.046666667	1.01E-06
3		1429 - 1439	138.3	42.15384	2000	156	0	0	0.00E+00
4		1453 - 1503	143.5	43.7388	2000	156	12	0.093333333	2.02E-06
5		1509 - 1519	148	45.1104	2000	150	258	2.006666667	4.51E-05
6		1531 - 1541	153.1	46.66488	2000	153	132	1.026666667	2.26E-05
7		1551 - 1605	157.6	48.03648	2000	156	6	0.046666667	1.01E-06
8		1618 - 1428	162.9	49.65192	2000	156	6	0.046666667	1.01E-06
9		1637 - 1647	167.2	50.96256	2000	156	3	0.023333333	5.04E-07
10		1659 - 1709	172.4	52.54752	2000	156	24	0.186666667	4.03E-06
11		1719 - 1729	176.8	53.88864	2000	157	6	0.046666667	1.00E-06
12		1746 - 1755	182	55.4736	2000	156	6	0.046666667	1.01E-06
13		1805 - 1815	186.5	56.8452	2000	156	0	0	0.00E+00
14		1825 - 1835	191.6	58.39968	2000	156	3	0.023333333	5.04E-07
15		1840 - 1850	196.1	59.77128	2000	156	3	0.023333333	5.04E-07
16		1901 - 1911	201.25	61.341	2000	156	0	0	0.00E+00
17		1916 - 1926	205.7	62.69736	2000	156	0	0	0.00E+00
18		1935 - 1942	210.9	64.28232	2000	156	3	0.023333333	5.04E-07
19		1952 - 2002	215.3	65.62344	2000	156	0	0	0.00E+00
20	9-Nov-04	1029 - 1039	220.5	67.2084	2000	156	0	0	0.00E+00
21		1047 - 1057	225	68.58	2000	156	60	0.466666667	1.01E-05
22		1106 - 1116	230.1	70.13448	2000	156	0	0	0.00E+00
23		1123 - 1133	234.6	71.50608	2000	150	294	2.286666667	5.14E-05
24		1143 - 1153	239.8	73.09104	2000	156	0	0	0.00E+00
25		1159 - 1209	244.2	74.43216	2000	145	498	3.873333333	9.00E-05
26		1216 - 1226	249.4	76.01712	2000	155	0	0	0.00E+00
27		1235 - 1245	253.8	77.35824	2000	156	0	0	0.00E+00
28		1252 - 1302	259	78.9432	2000	156	0	0	0.00E+00
29		1309 - 1319	263.4	80.28432	2000	156	0	0	0.00E+00

Packer Test Flow Plots

C1.1 to C1.46 represent plots outlining which monitoring wells were influenced by packer testing conducted during 2004. The headings to these plots indicate the injection well, the depth the packer was at and the resulting hydraulic conductivity of the packer test. The arrows indicate the direction of flow resulting from the field of influence from the injection well (contained packer). The field of influence of the packer was 1.5 m.

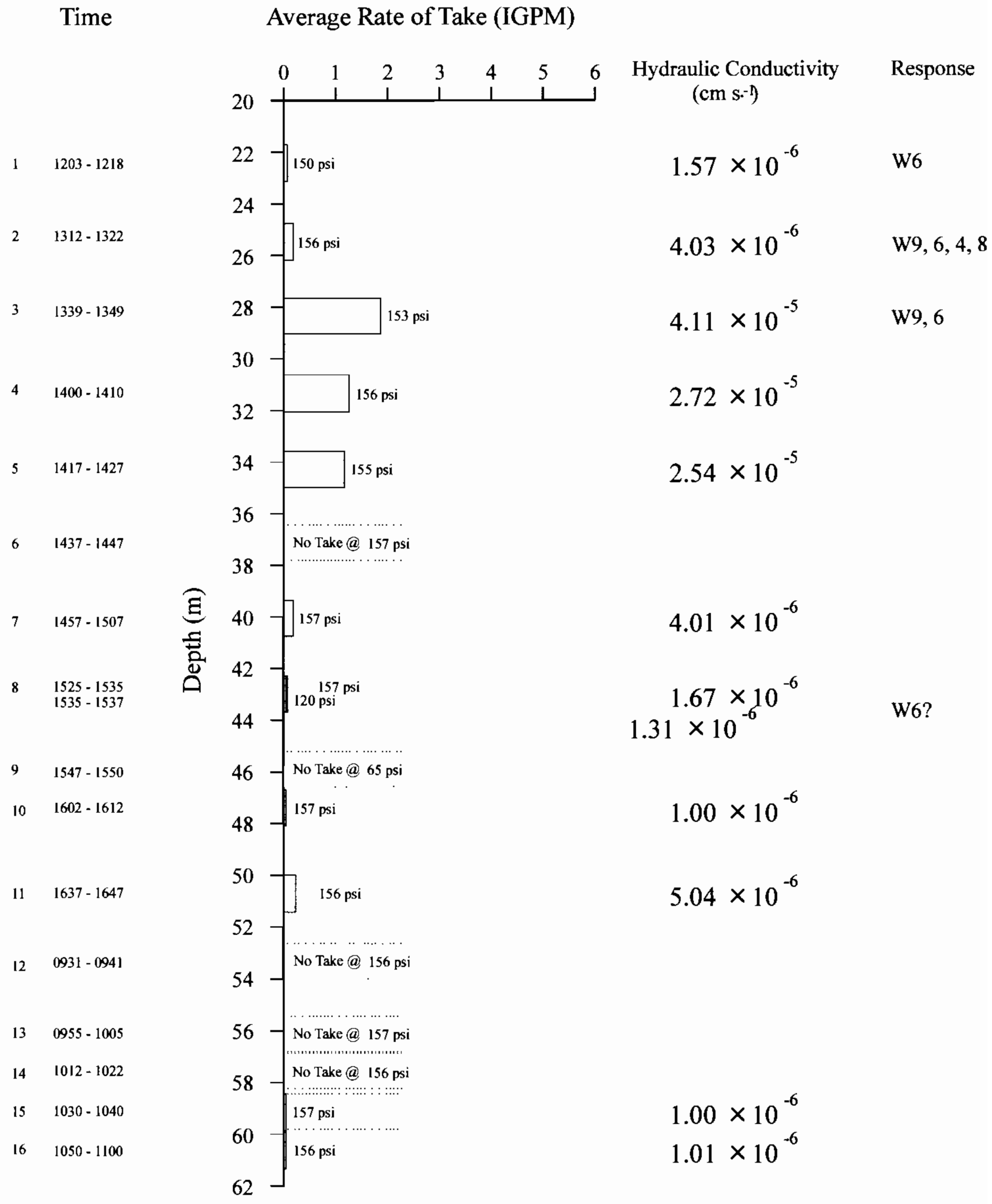


Figure C1 - Packer test in Well 10 for the 18th and 21st of June 2004 indicating average take, hydraulic conductivity and responding wells

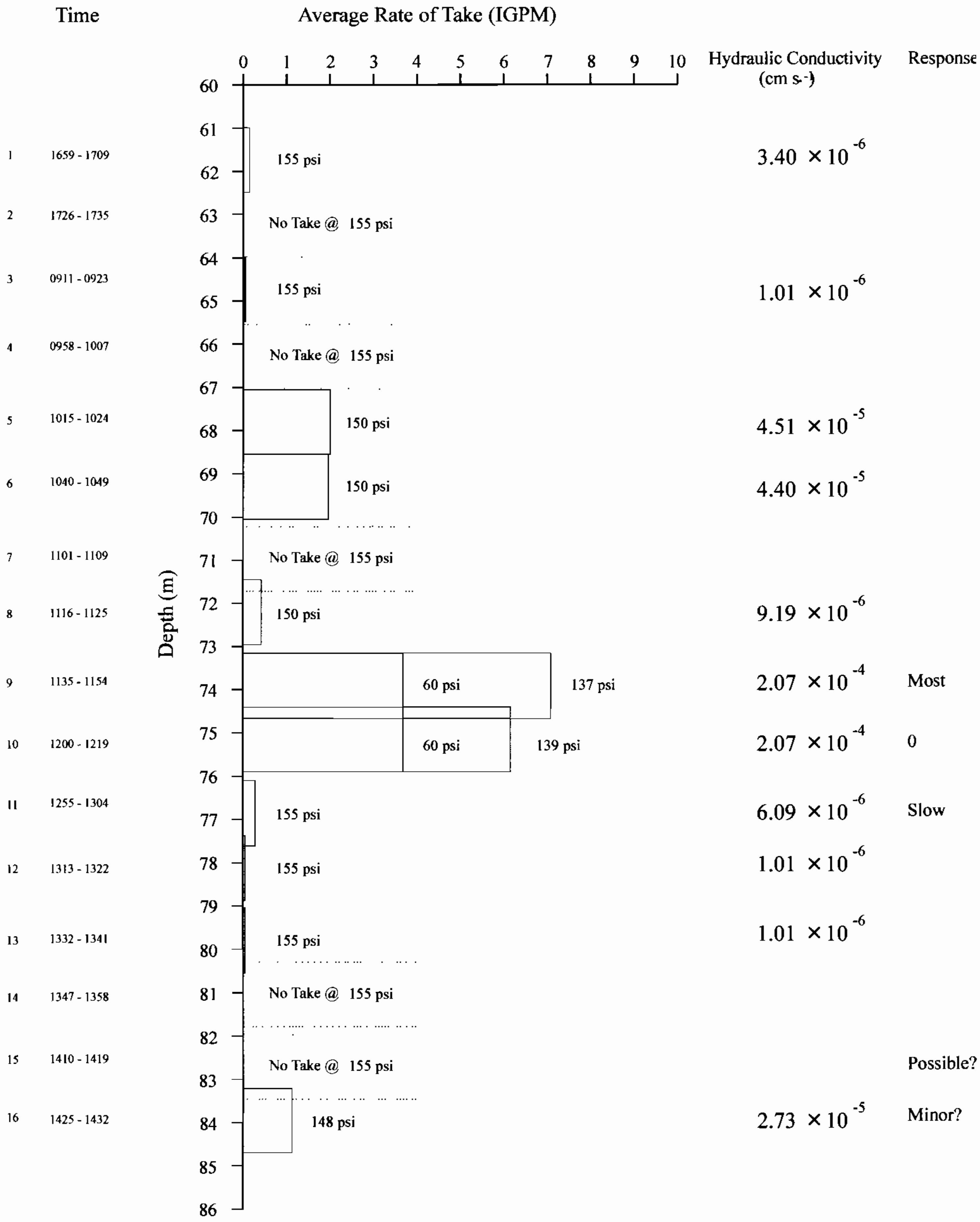


Figure C2 - Packer test in Well 10 for the 12th and 13th of August 2004 indicating average take, hydraulic conductivity and responding wells

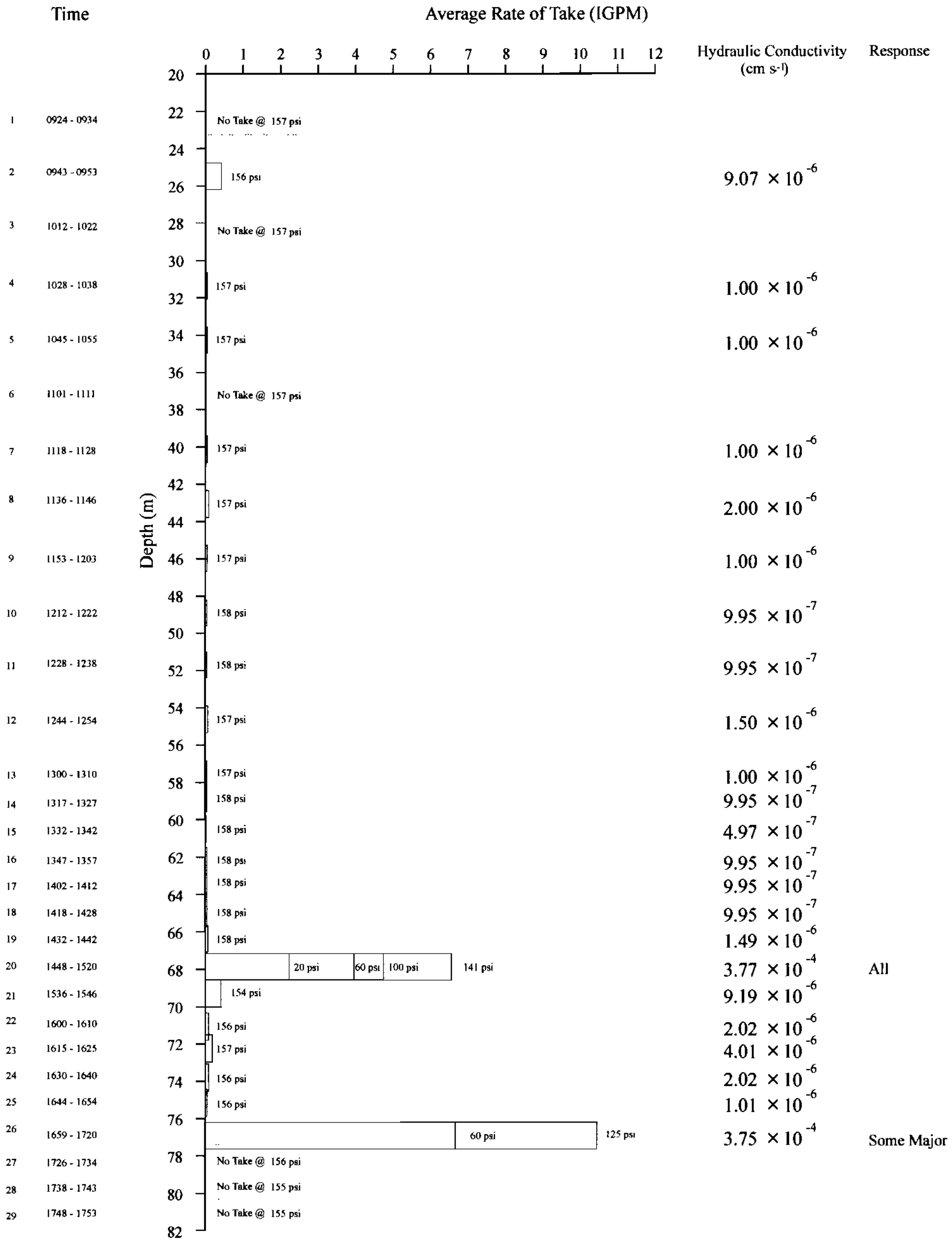


Figure C3 - Packer test in Well 9 for the 22nd of June 2004 indicating average take, hydraulic conductivity and responding wells

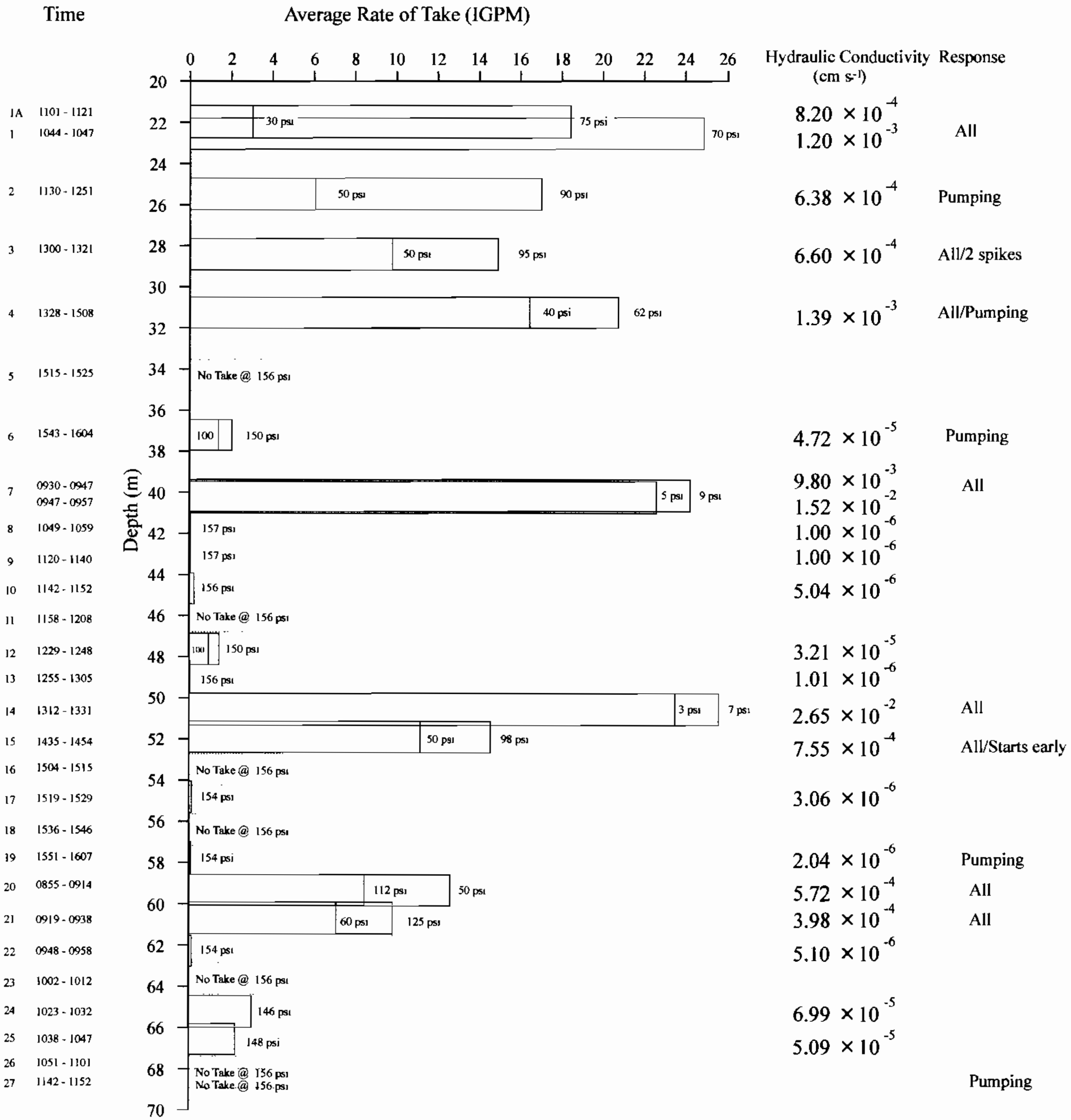


Figure C4 - Packer test in Well 6 for the 7th to the 9th of July 2004 indicating average take, hydraulic conductivity and responding wells

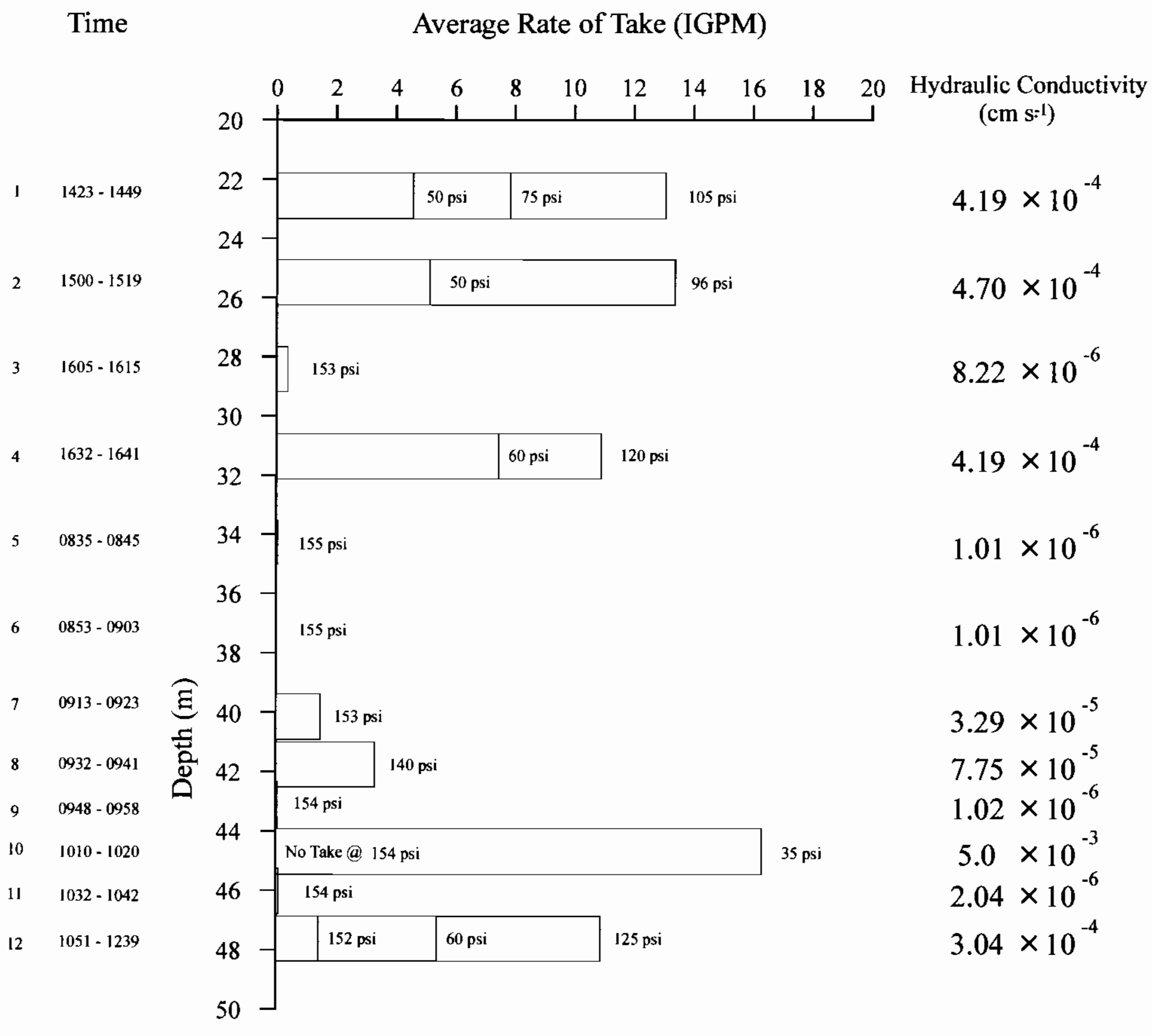


Figure C5 - Packer test in Well 7 for the 12th and 13th of July 2004 indicating average take, hydraulic conductivity and responding wells

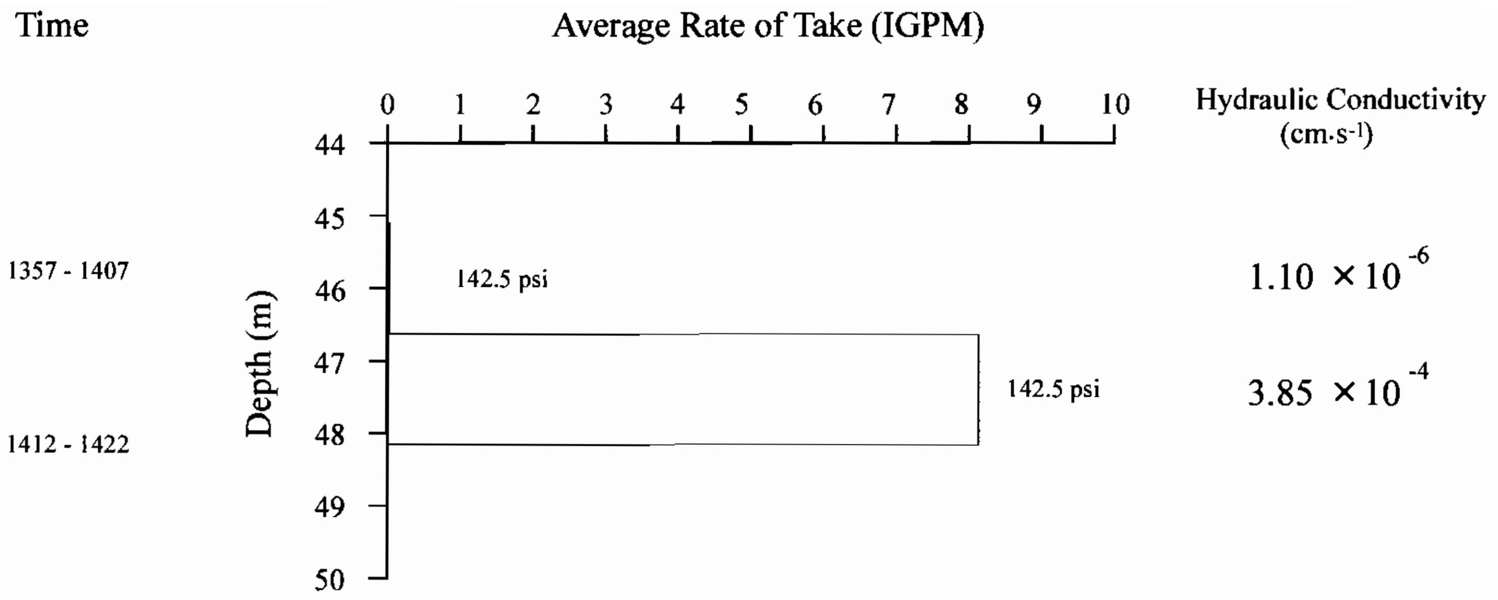
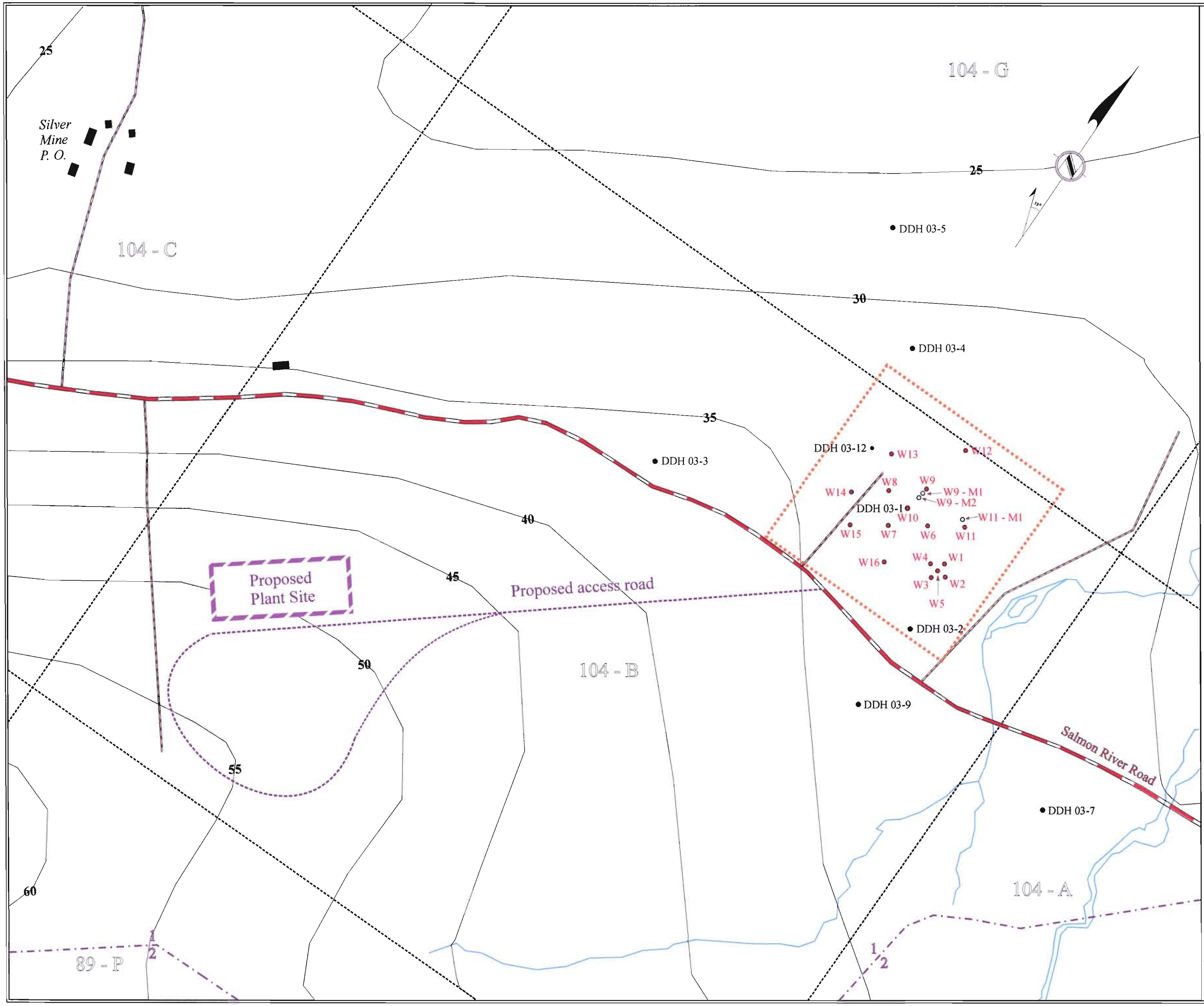


Figure C8 - Packer test in Well 5 for the 4th of November 2004 indicating average take, hydraulic conductivity and responding wells



Map 2006 - 7

YAVA TECHNOLOGIES INC

Hydrogeologic Test Wells

SILVER MINE PROPERTY, CAPE BRETON COUNTY
NOVA SCOTIA

R. E. A. BOUDREAU

SCALE 1::160

0 100 200 300 400 500 feet

0 50 100 150 meters

Contour interval in meters above sea-level

2006

Explanation

CARBONIFEROUS

2 **RIVERSDALE GROUP**
Silver Mine Formation
sandstone

1 **CANSO GROUP**
Mackeigan Lake Formation
grey shale (mudstone)

Gravel road, trail, proposed

Claim line

Geologic boundary (assumed) 2/1

Diamond drill hole ● DDH 03-9

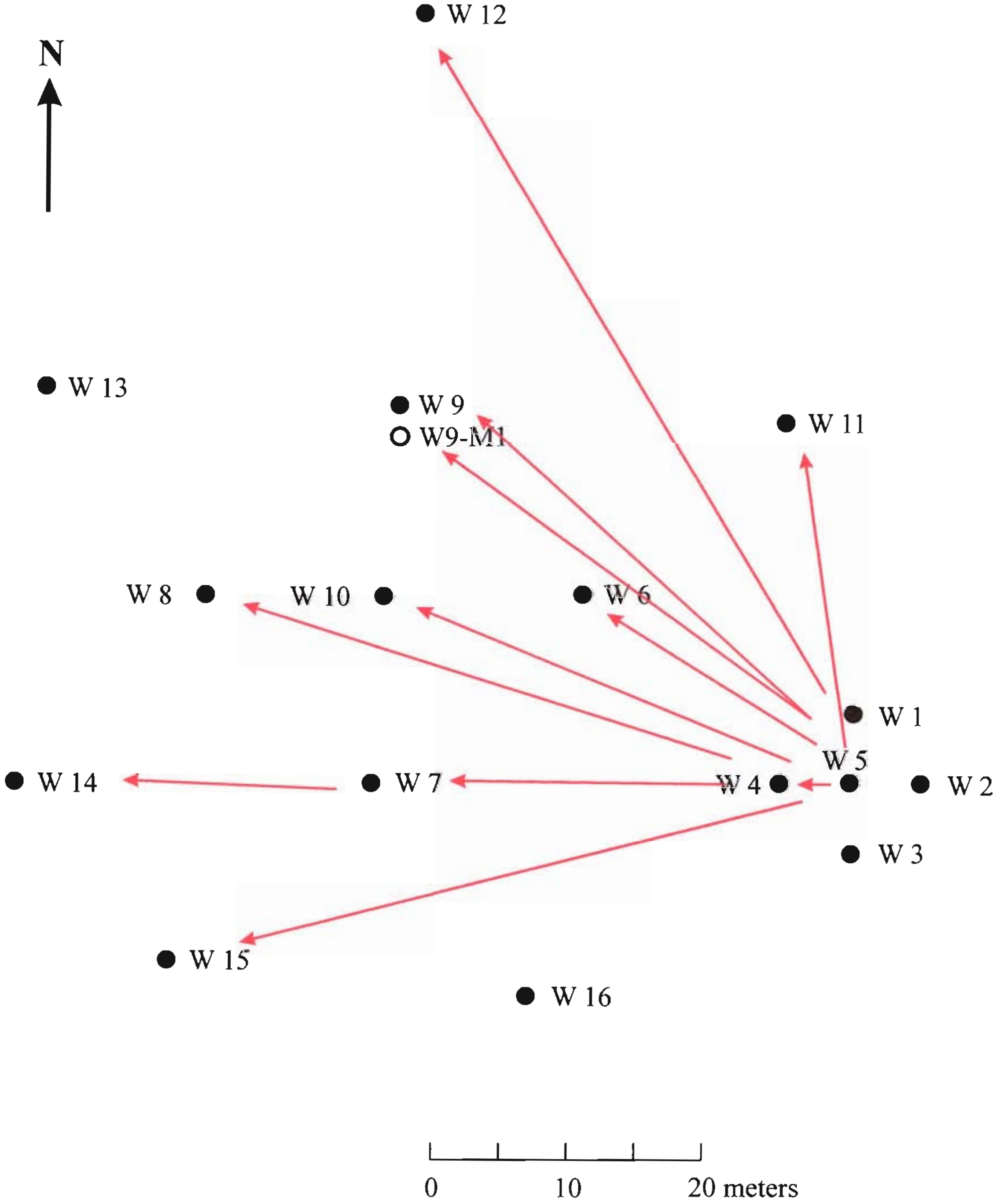
Test Well ● W10

Monitoring Well ○ W9 - M1

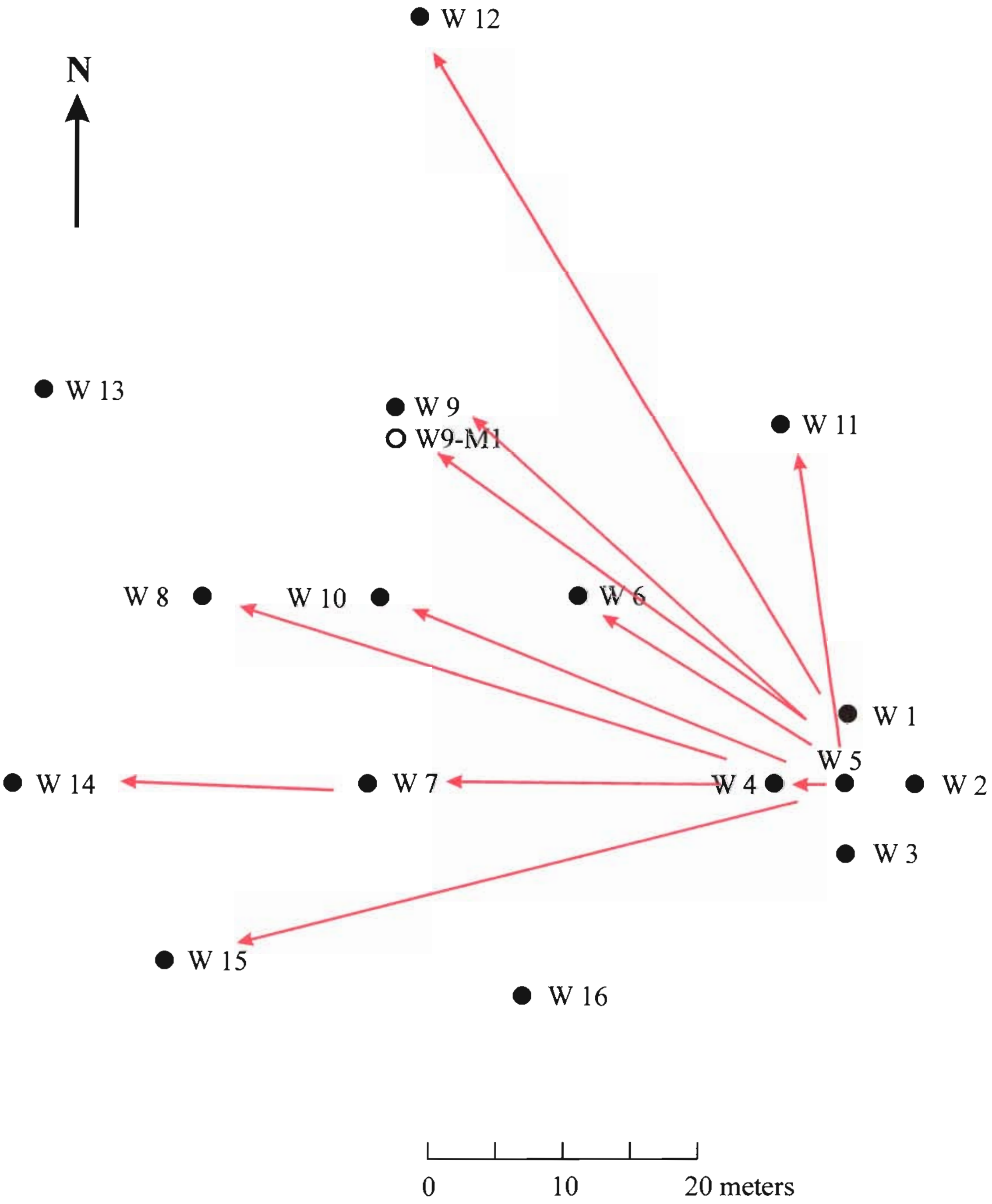
Stream, brook

Contours 25

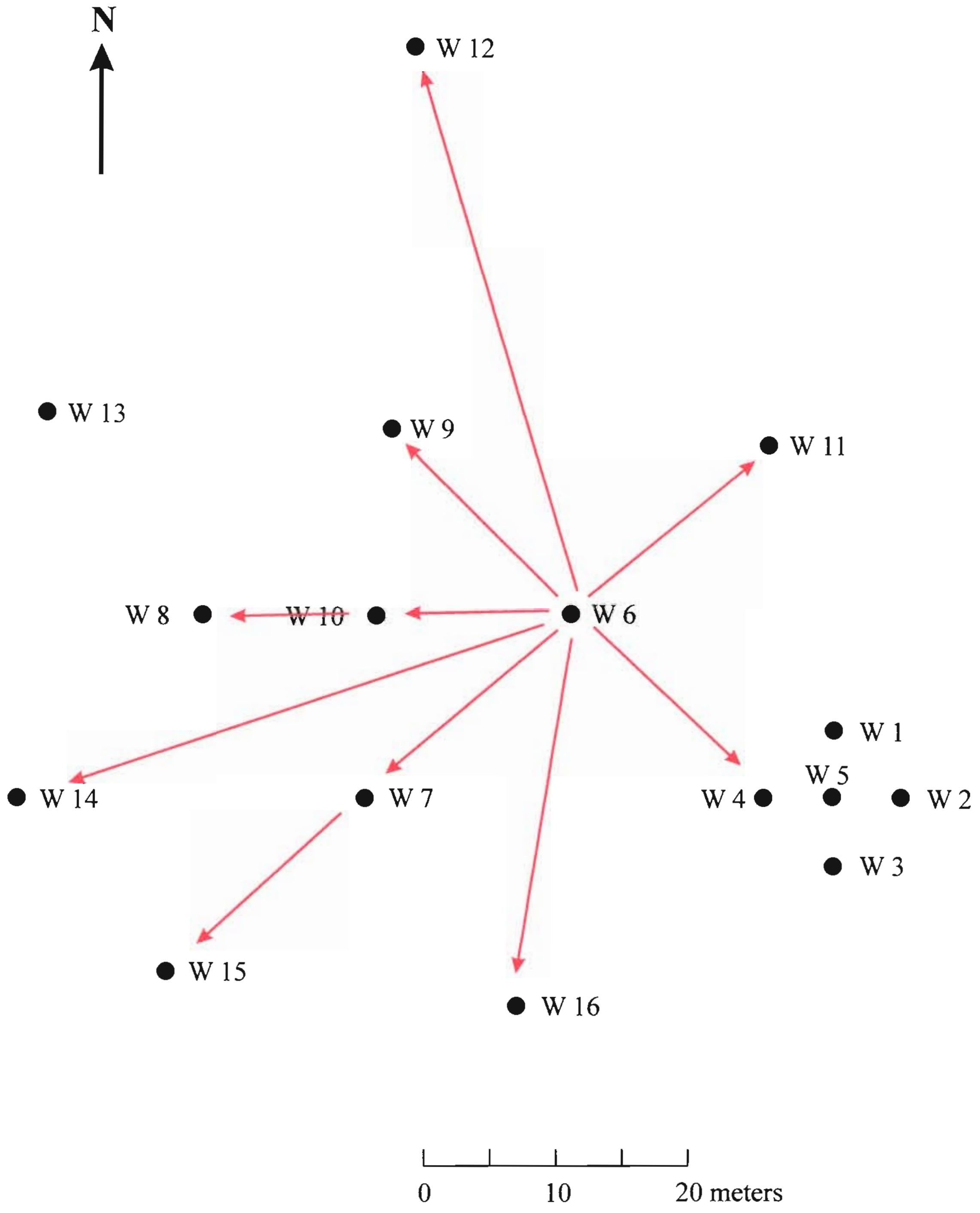
Appendix C.1.1
W5 - 04 Nov 2004
PL 1 - -13.86 to -15.36 masl
1.10E-6 cm/s



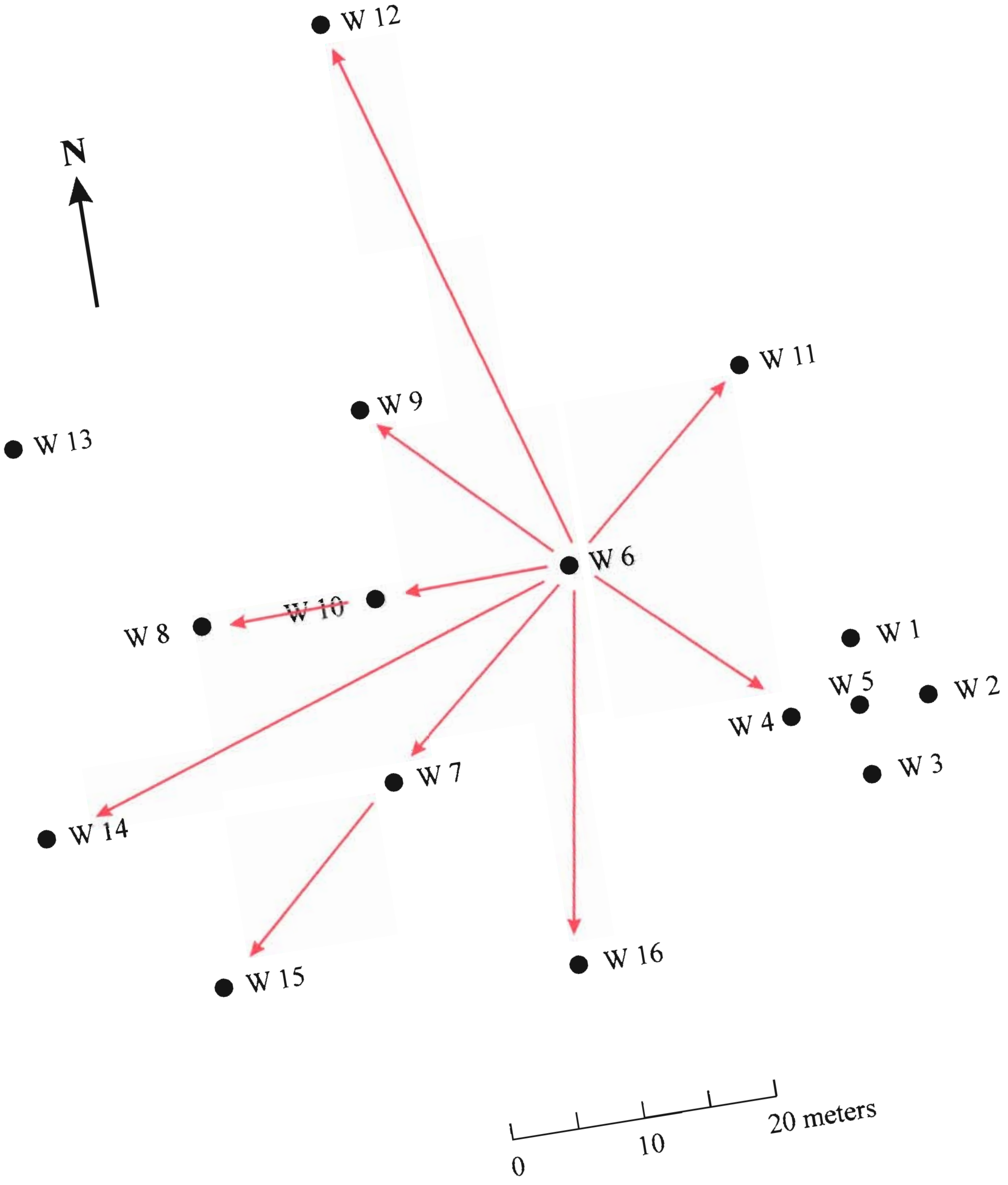
Appendix C.1.2
W5 - 04 Nov 2004
PL 2 - -15.36 to -16.86 masl
3.85E-4 cm/s



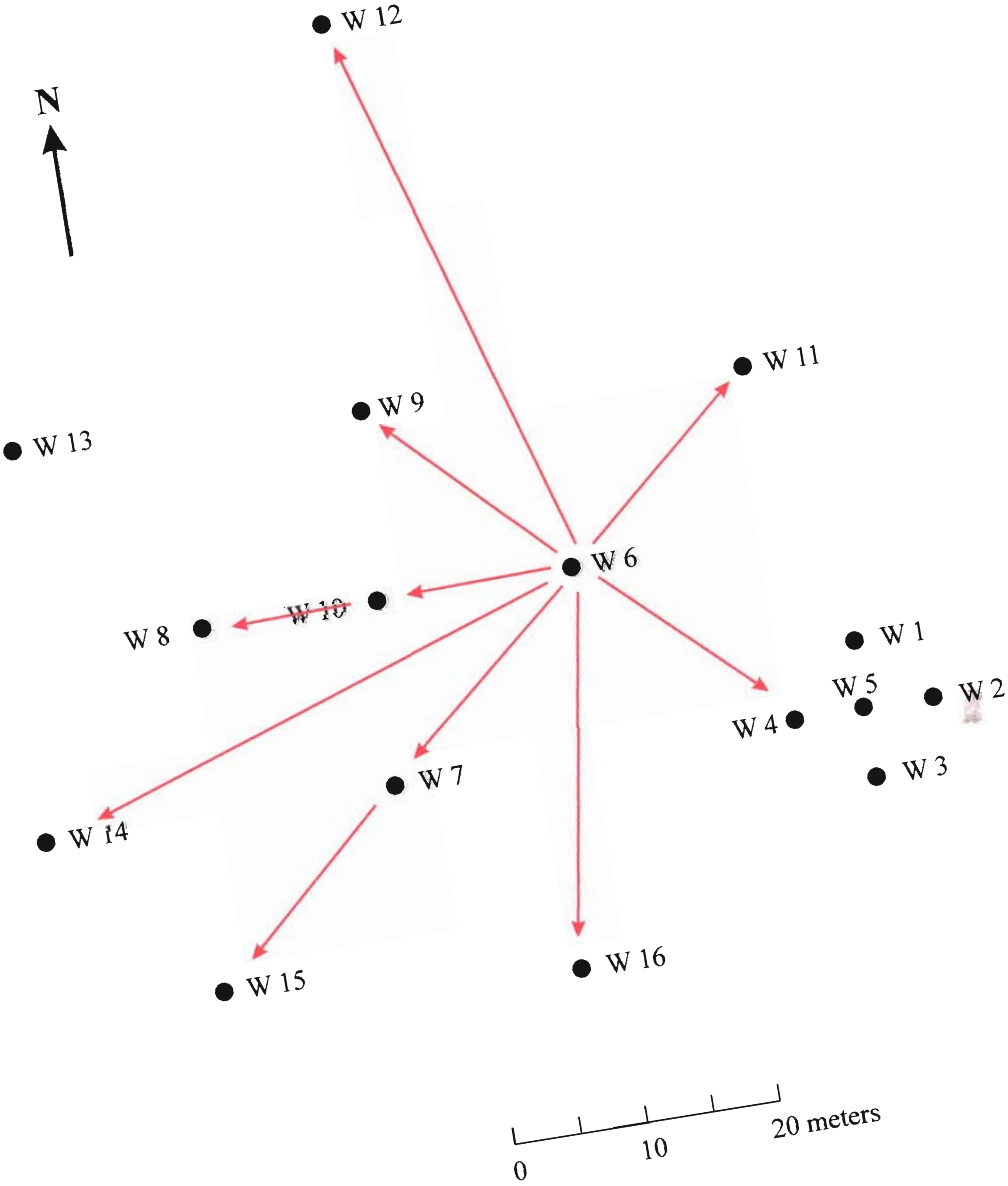
Appendix C.1.3
W6 - 09 July 2004
PL 1 - 11.42 to 9.92 masl
8.20E-4 cm/s



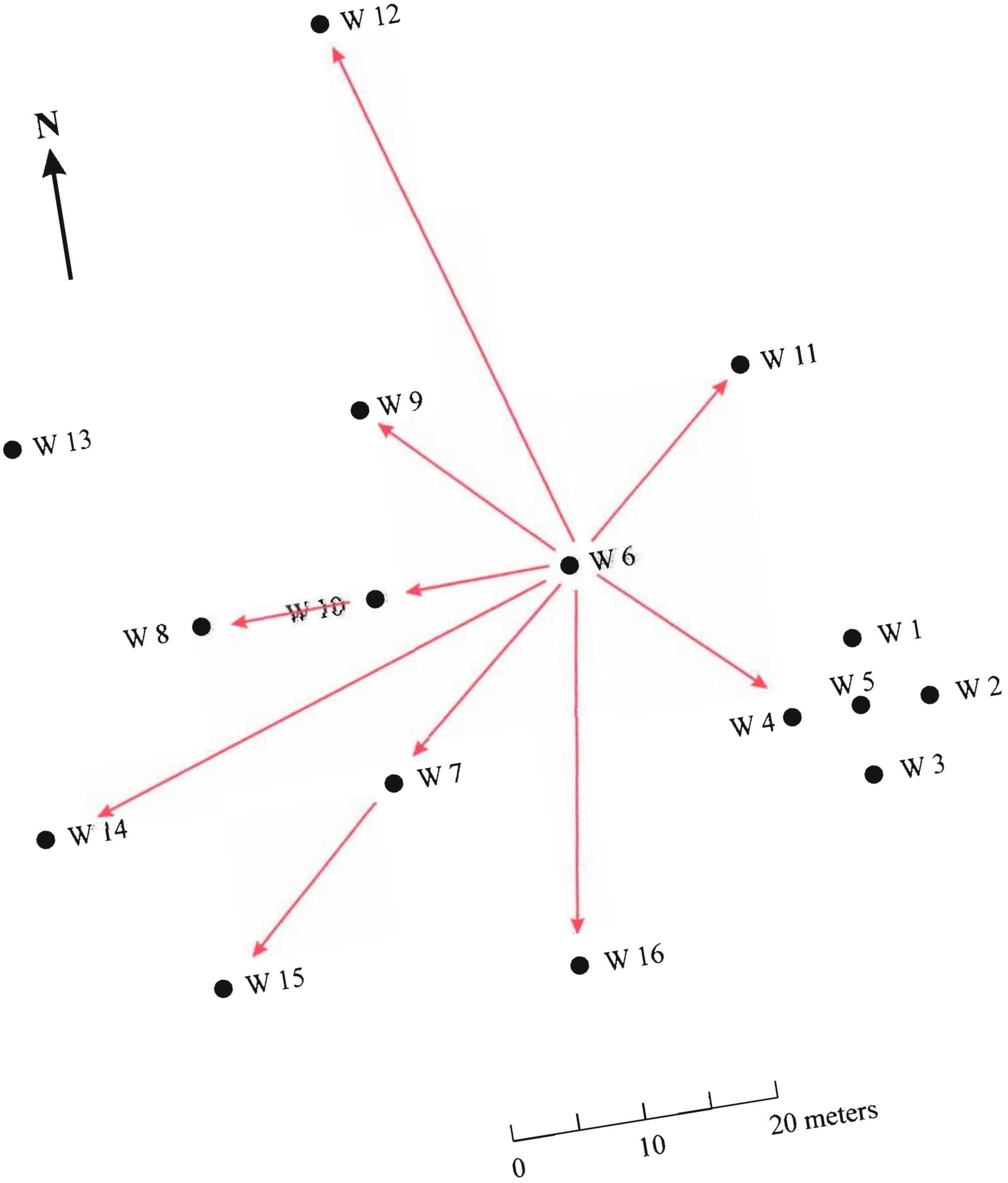
Appendix C.1.4
W6 - 09 July 2004
PL 2 - 10.92 to 9.42 masl
1.20E-3 cm/s



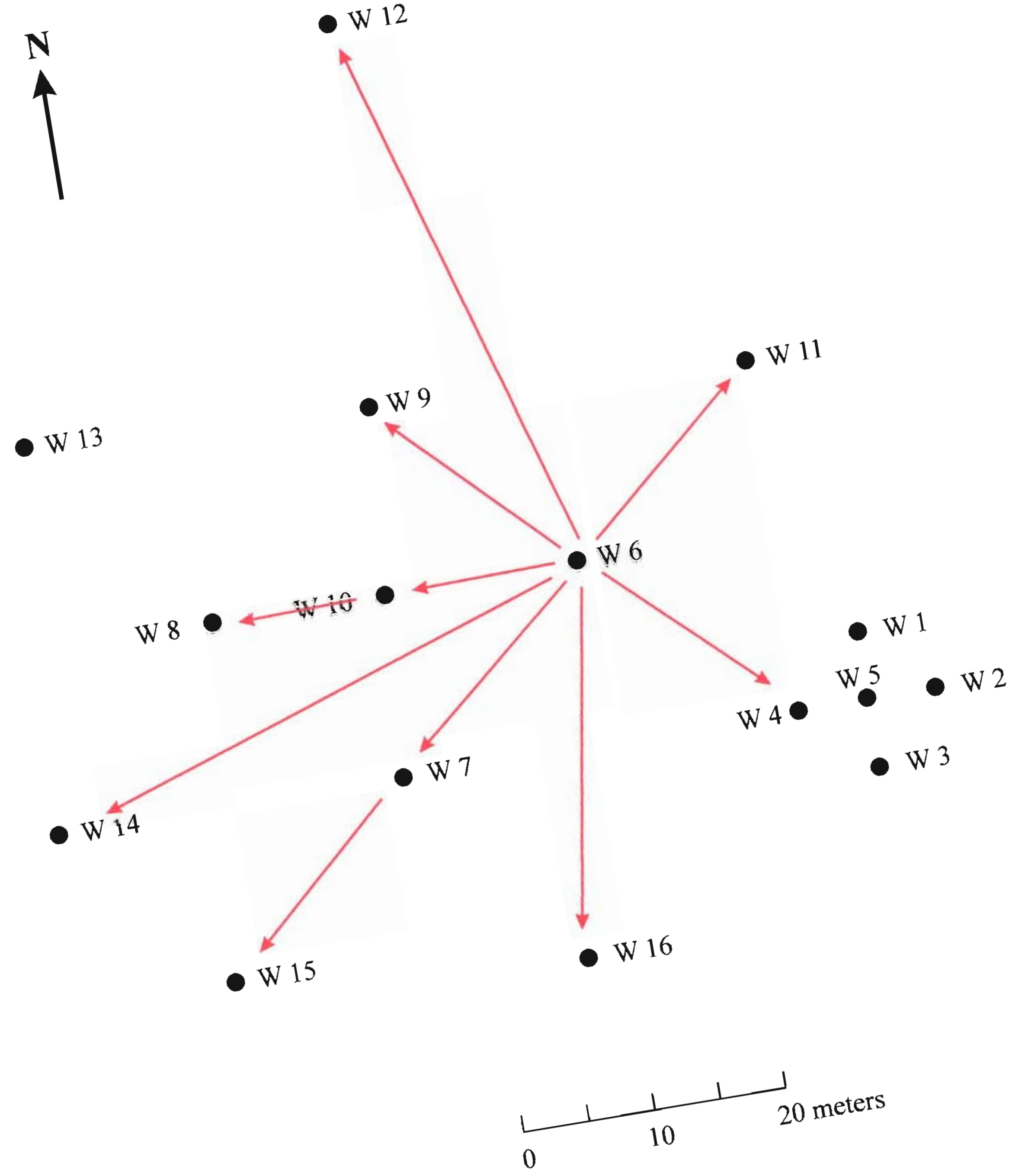
Appendix C.1.5
W6 - 09 July 2004
PL 3 - 7.67 to 6.17 masl
6.38E-4 cm/s



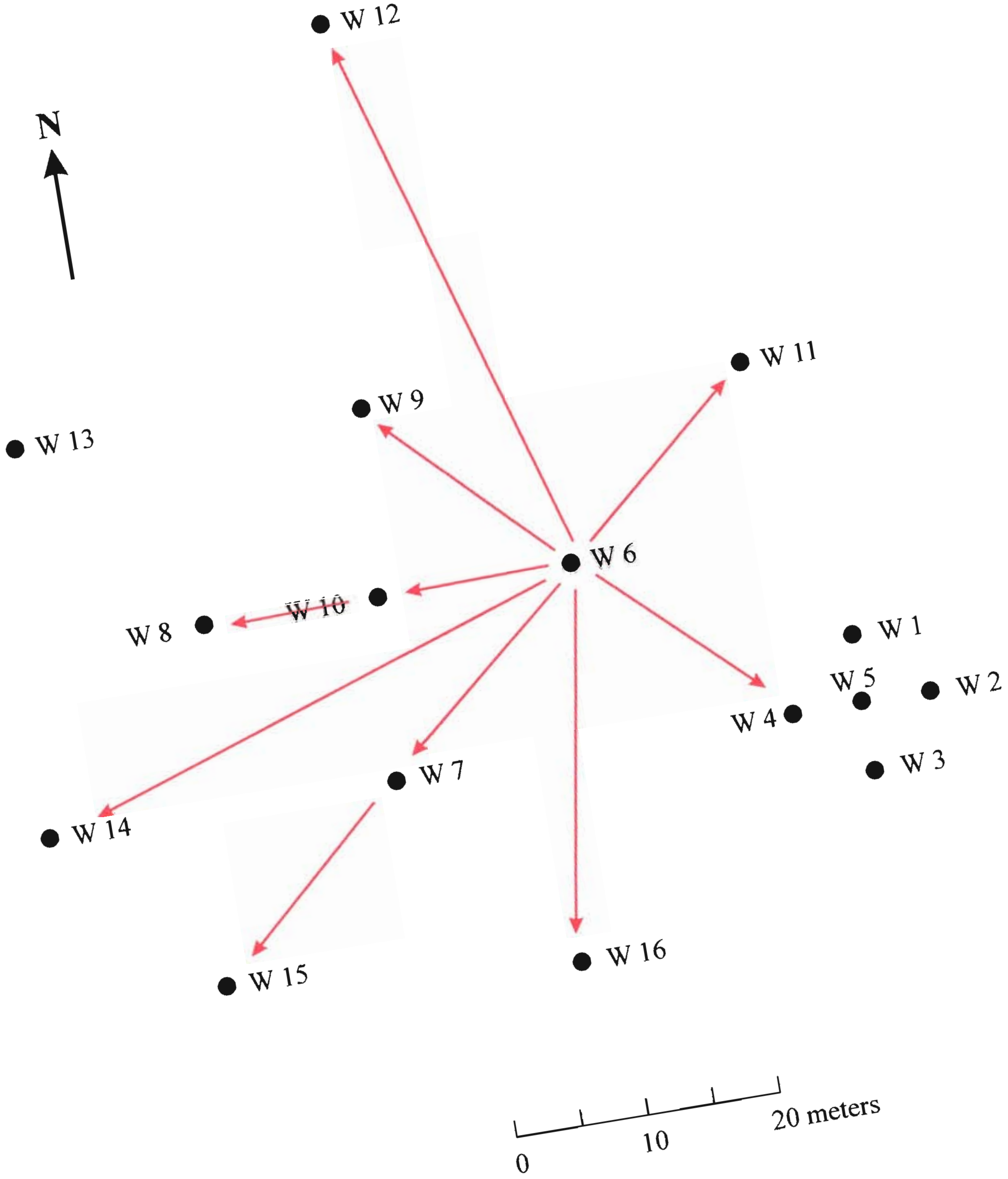
Appendix C.1.6
W6 - 09 July 2004
PL 4 - 1.92 to 0.42 masl
1.39E-3 cm/s



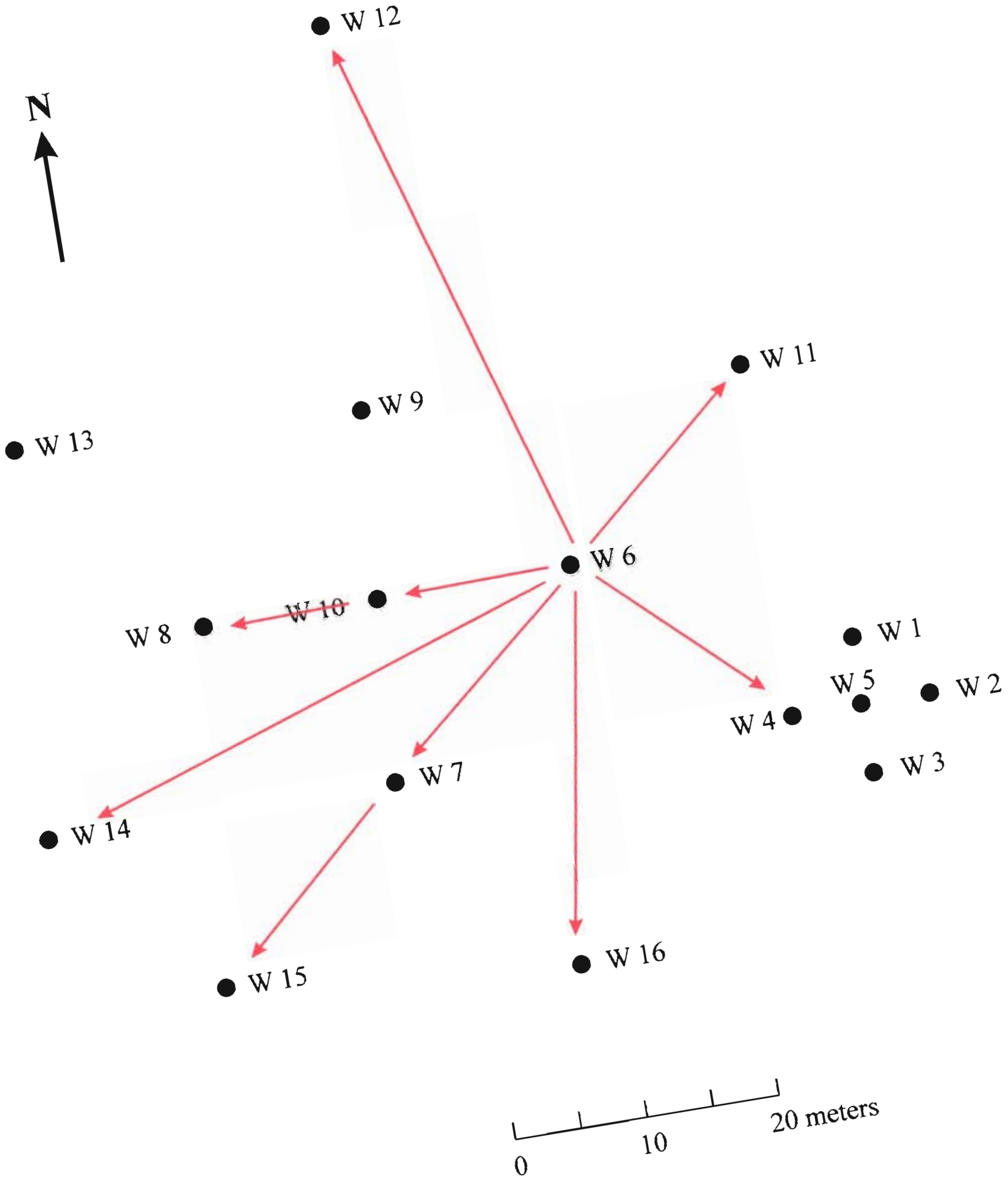
Appendix C.1.7
W6 - 09 July 2004
PL 5 - -4.08 to -5.58 masl
4.72E-5 cm/s



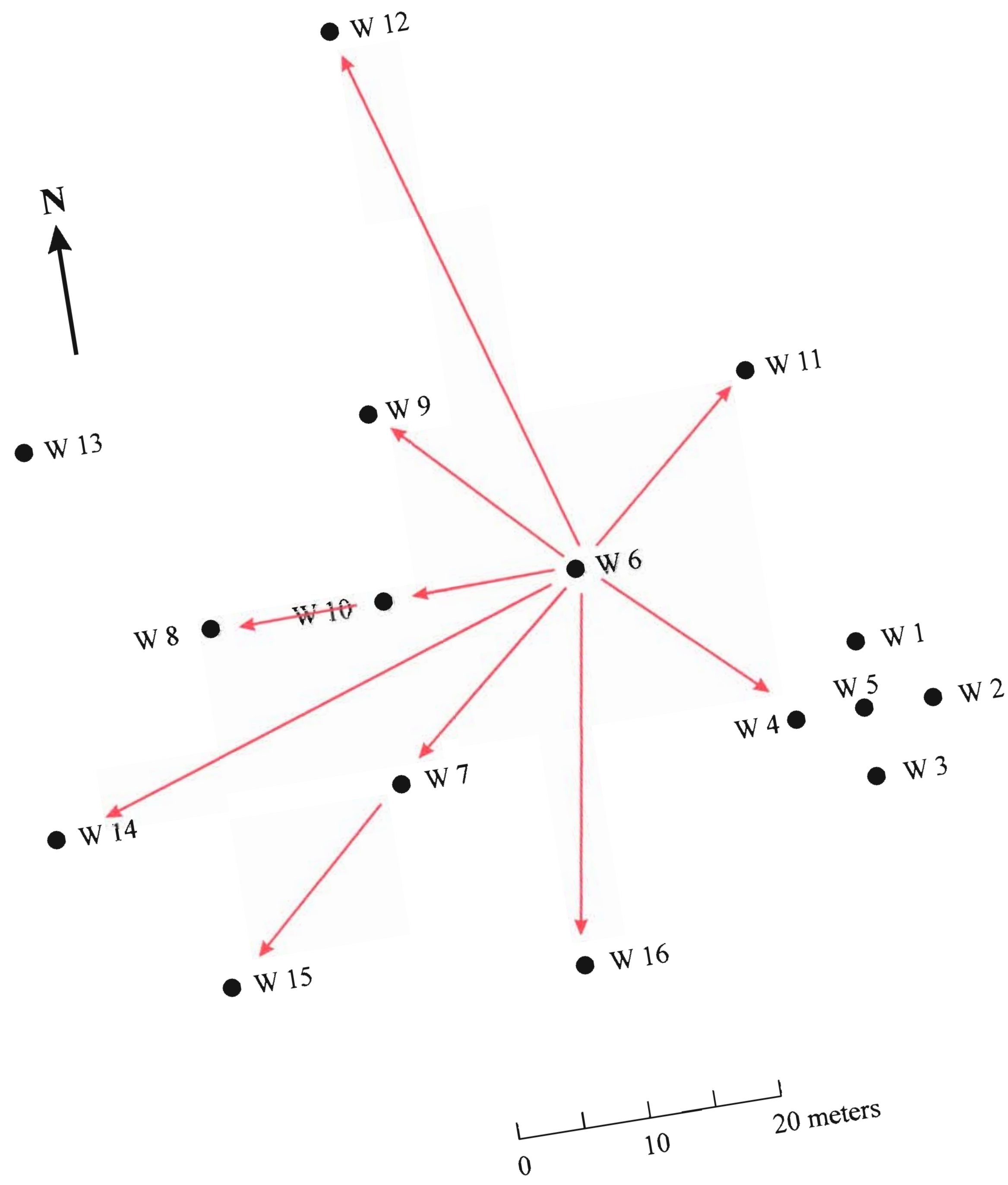
Appendix C.1.8
W6 - 09 July 2004
PL 6 - -7.08 to -8.58 masl
1.52E-2 cm/s



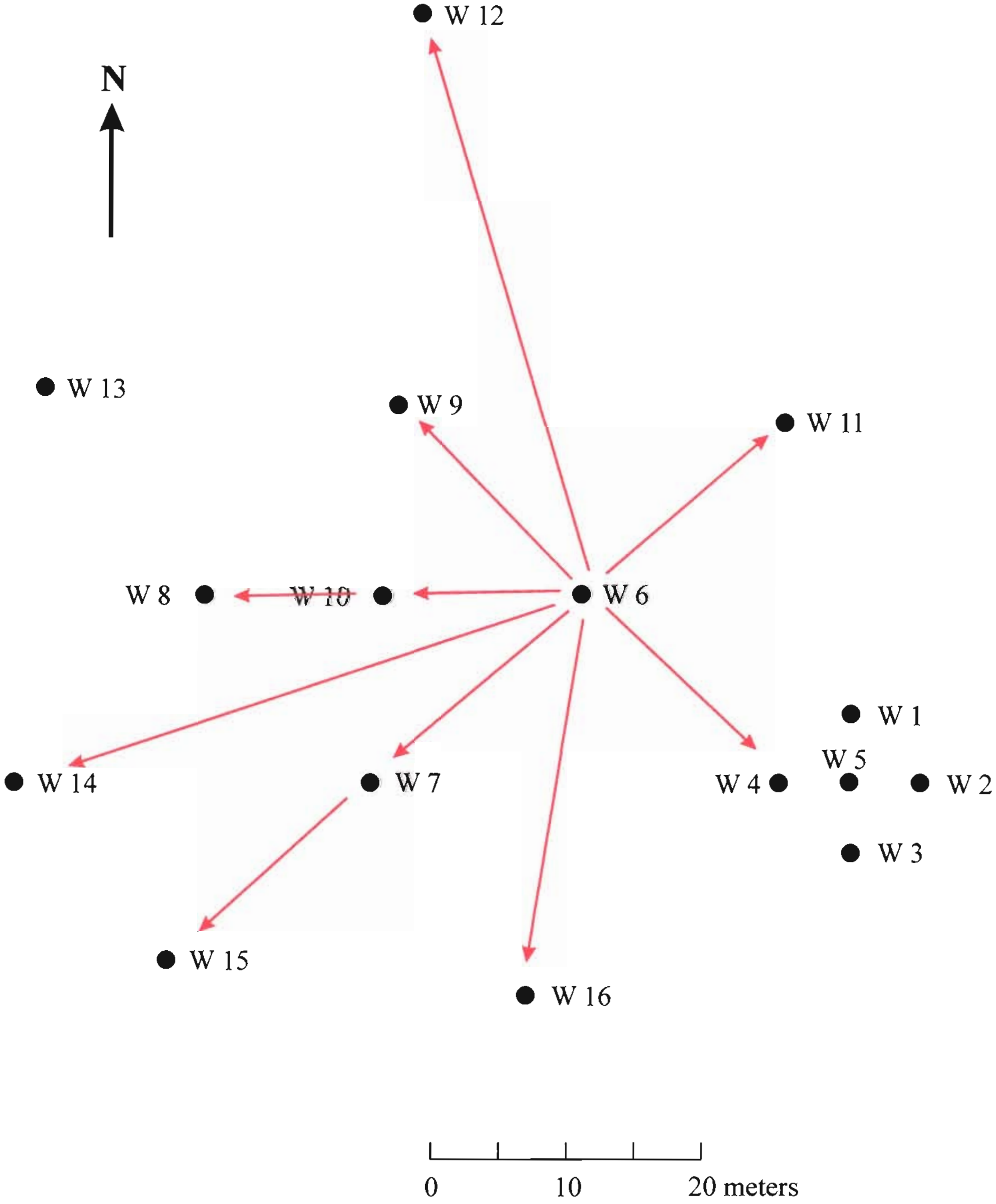
Appendix C.1.9
W6 - 09 July 2004
PL 7 - -14.58 to -16.08 masl
3.21E-5 cm/s



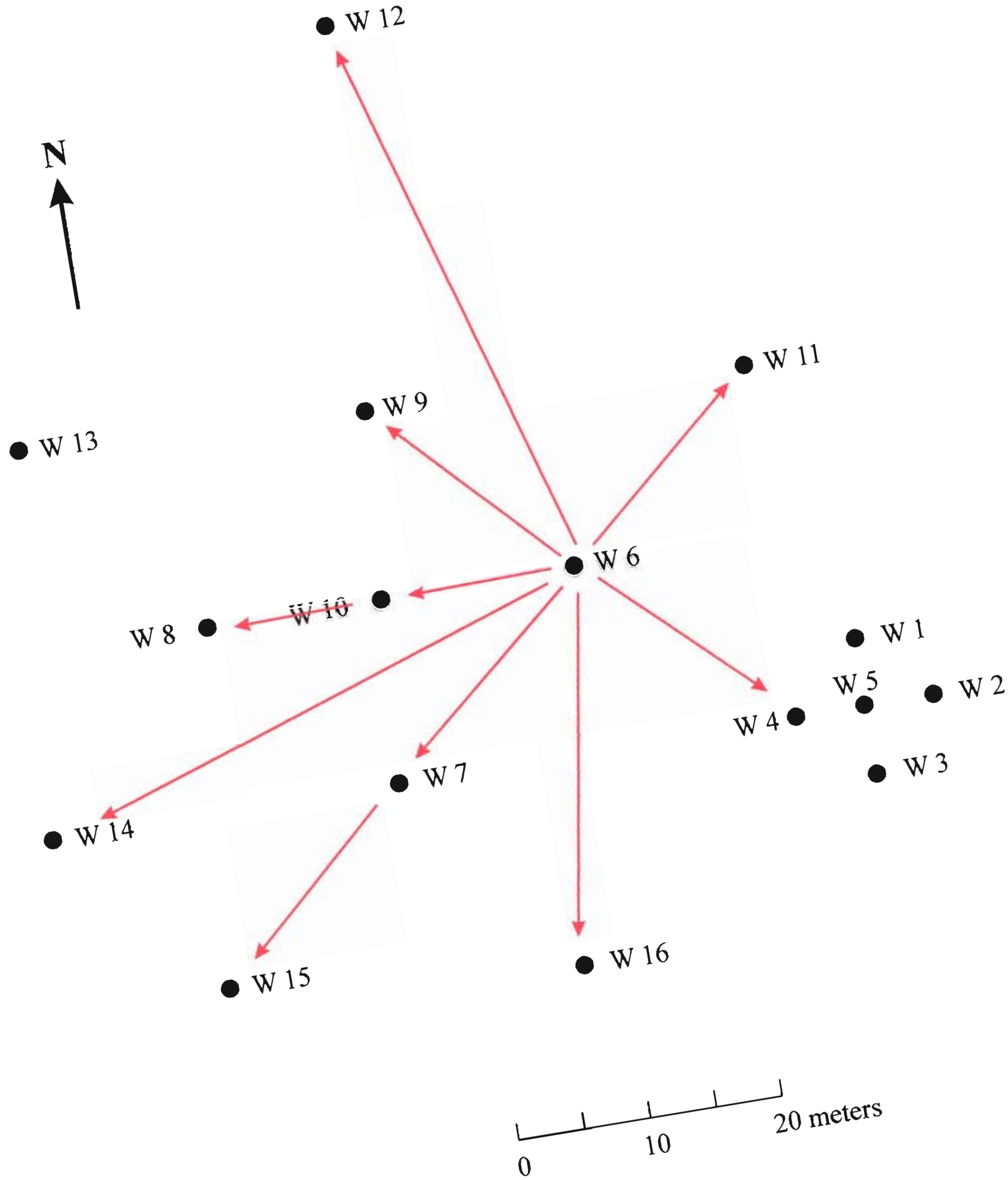
Appendix C.1.10
W6 - 09 July 2004
PL 8 - -17.08 to -18.58 masl
2.65E-2 cm/s



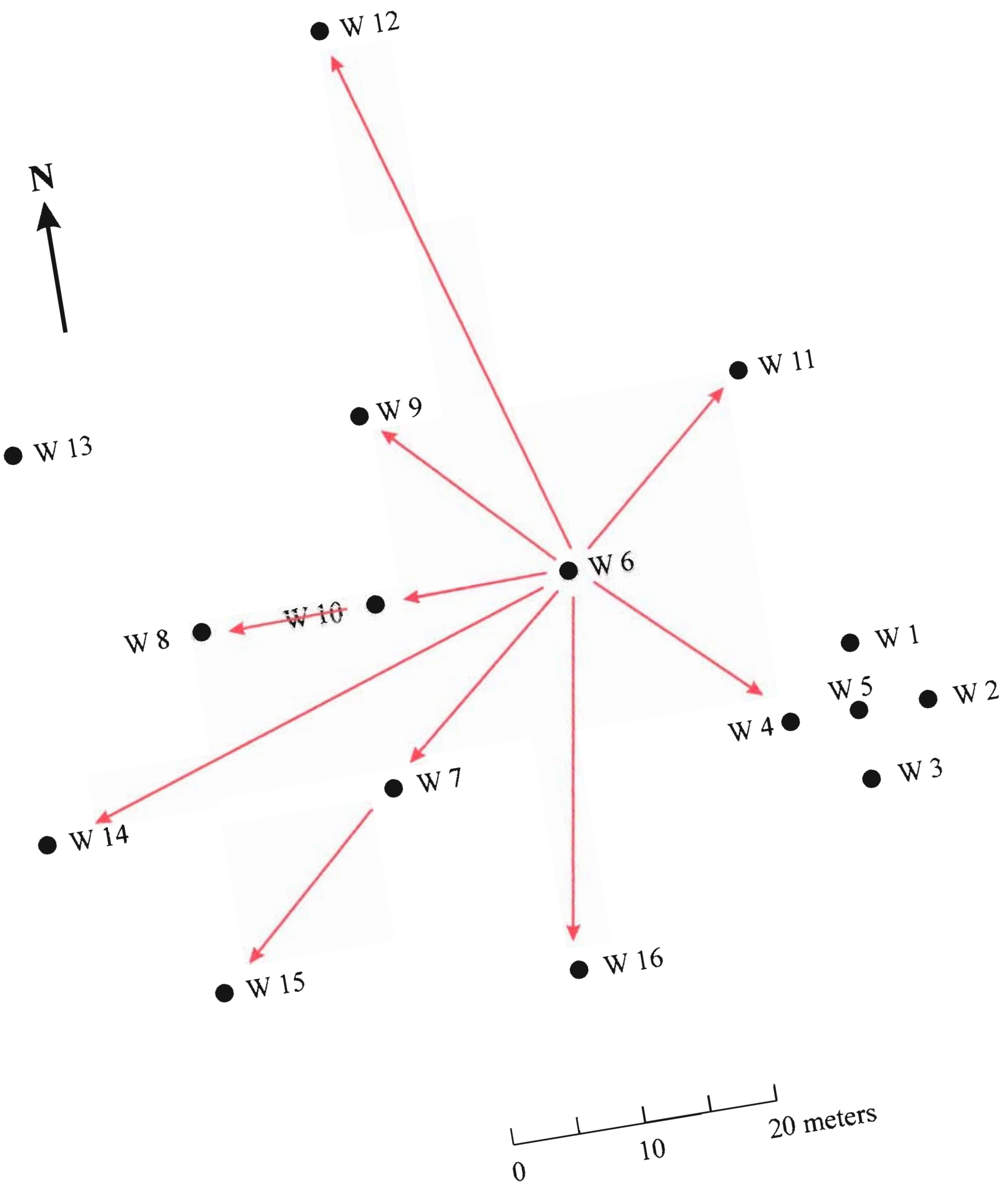
Appendix C.1.11
W6 - 09 July 2004
PL 9 - -18.83 to -20.33 masl
7.55E-4 cm/s



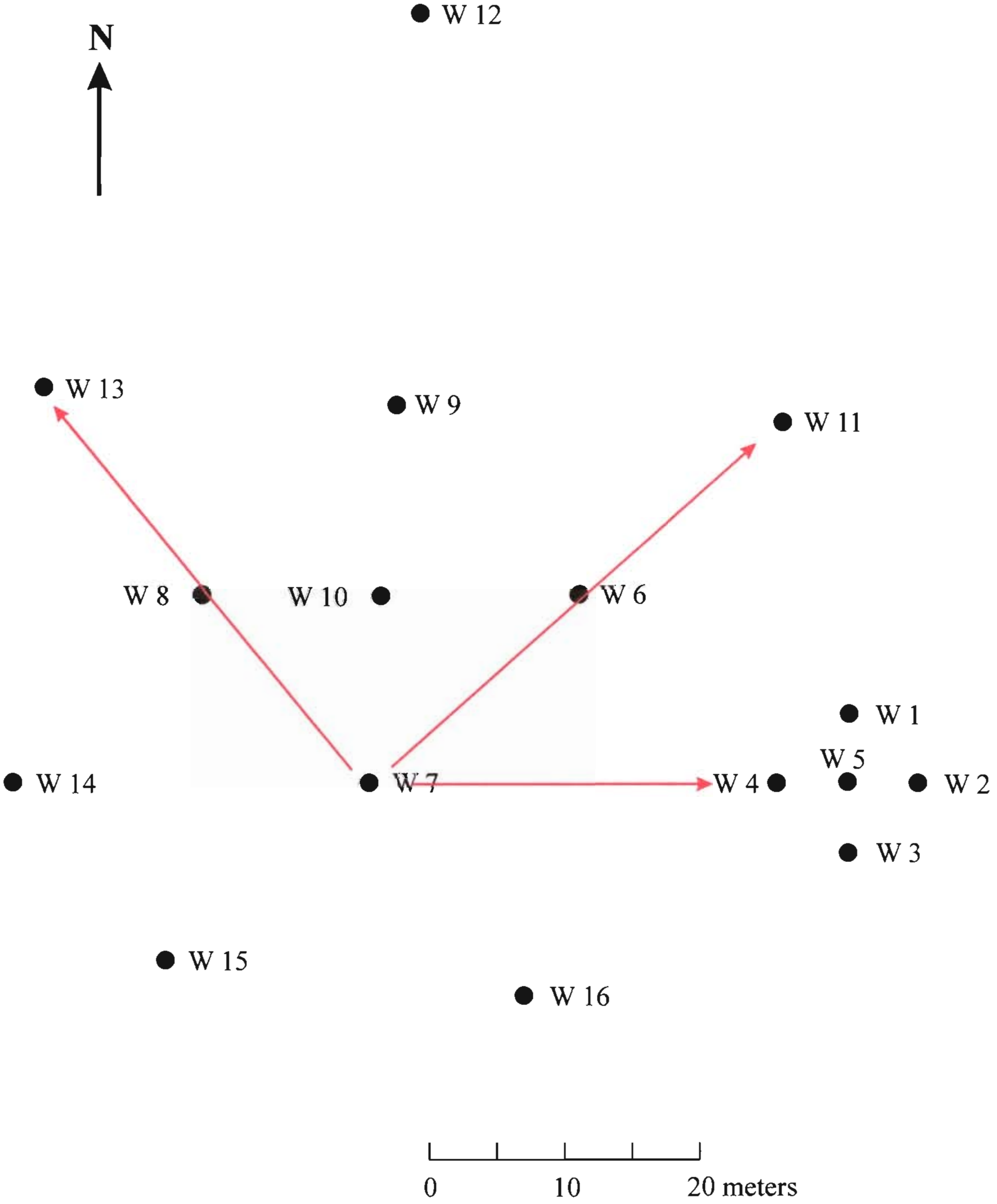
Appendix C.1.12
W6 - 09 July 2004
PL 10 - -26.08 to -27.58 masl
2.04E-6 cm/s



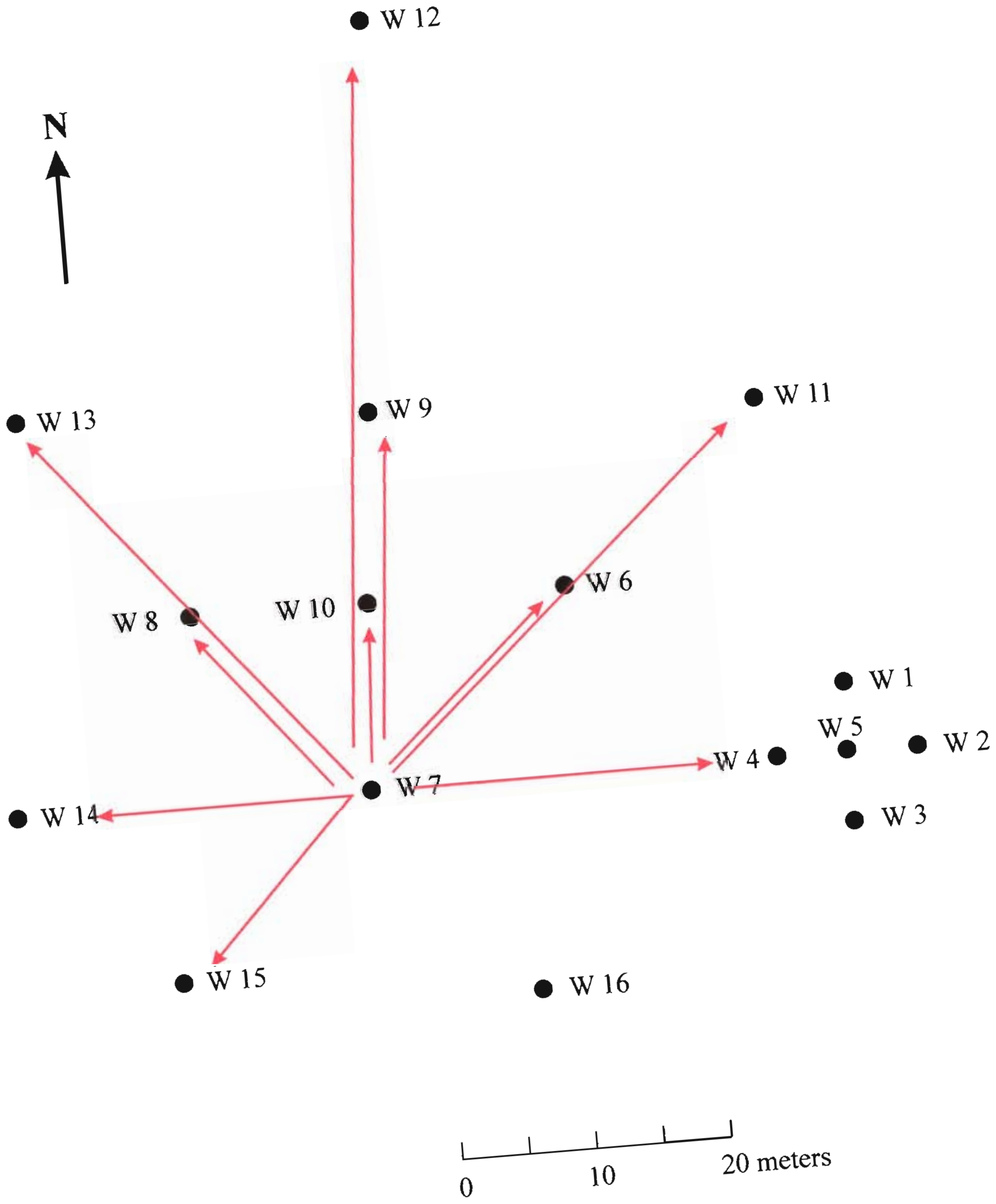
Appendix C.1.13
W6 - 09 July 2004
PL 11 - -27.58 to -29.08 masl
5.72E-4 cm/s



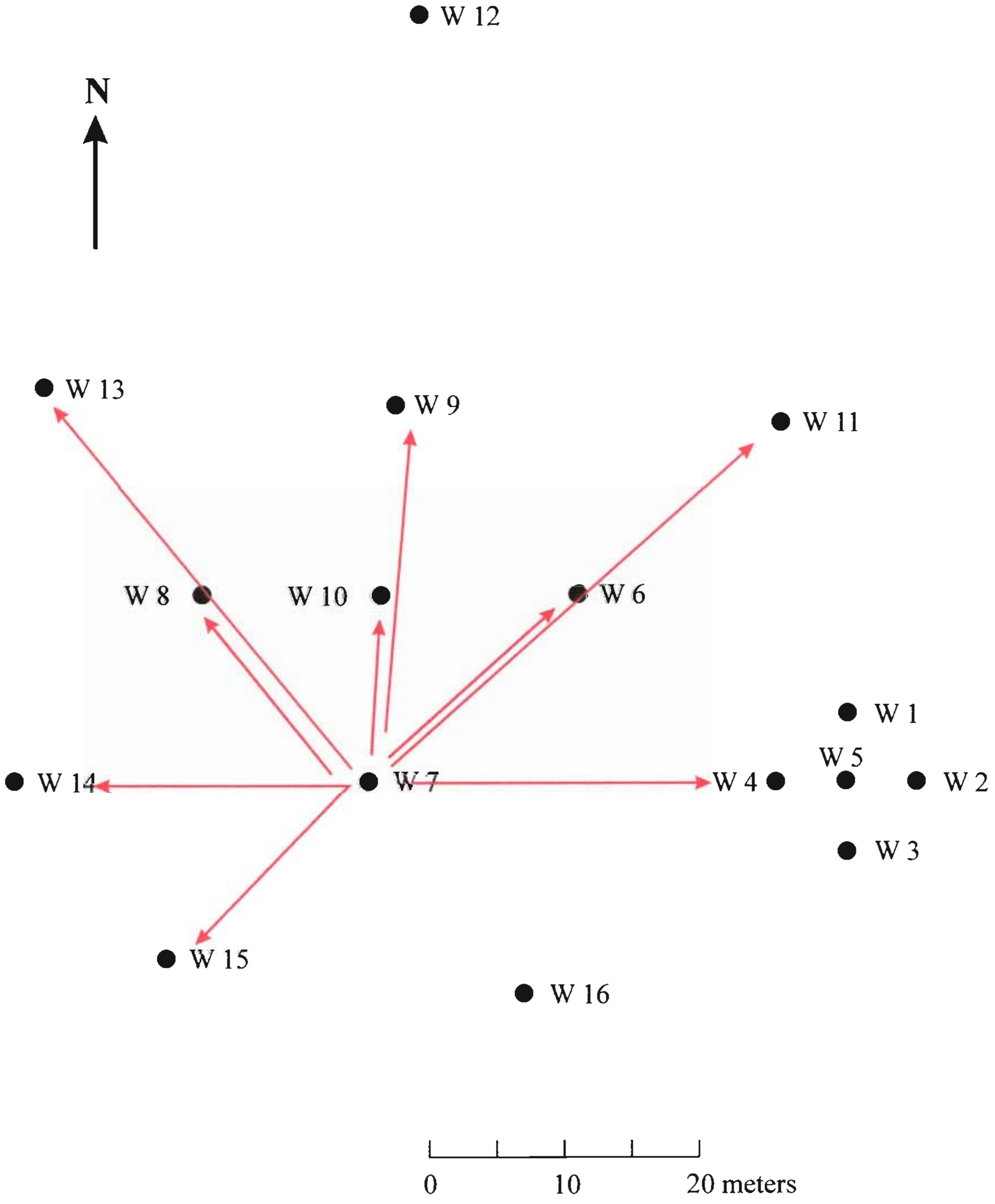
Appendix C.1.14
W7 - 12 July 2004
PL 1 - 9.72 to 8.22 masl
4.19E-4 cm/s



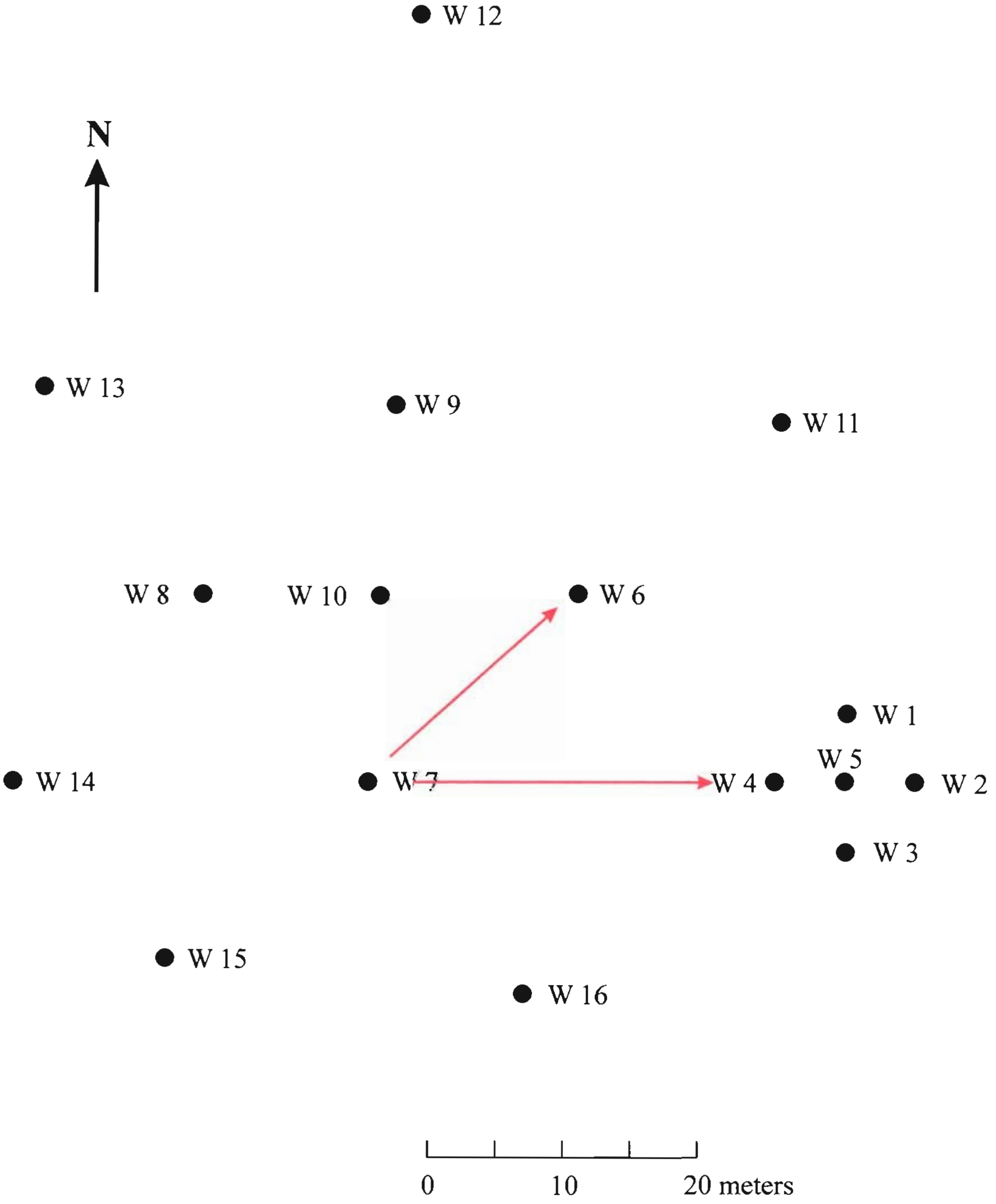
Appendix C.1.15
W7 - 12 July 2004
PL 2 - 8.22 to 6.72 masl
4.70E-4 cm/s



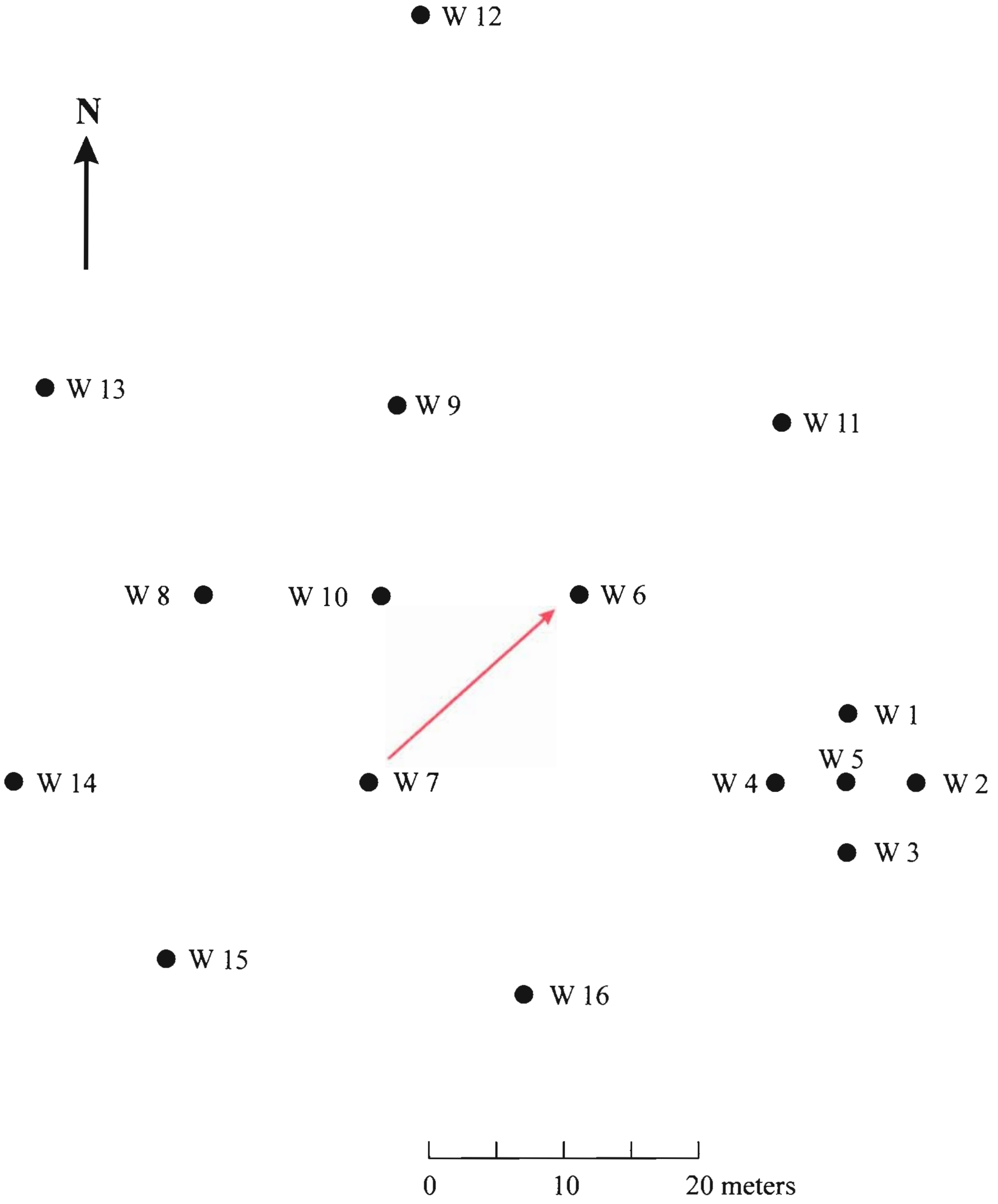
Appendix C.1.16
W7 - 12 July 2004
PL 3 - 2.22 to 0.72 masl
4.19E-4 cm/s



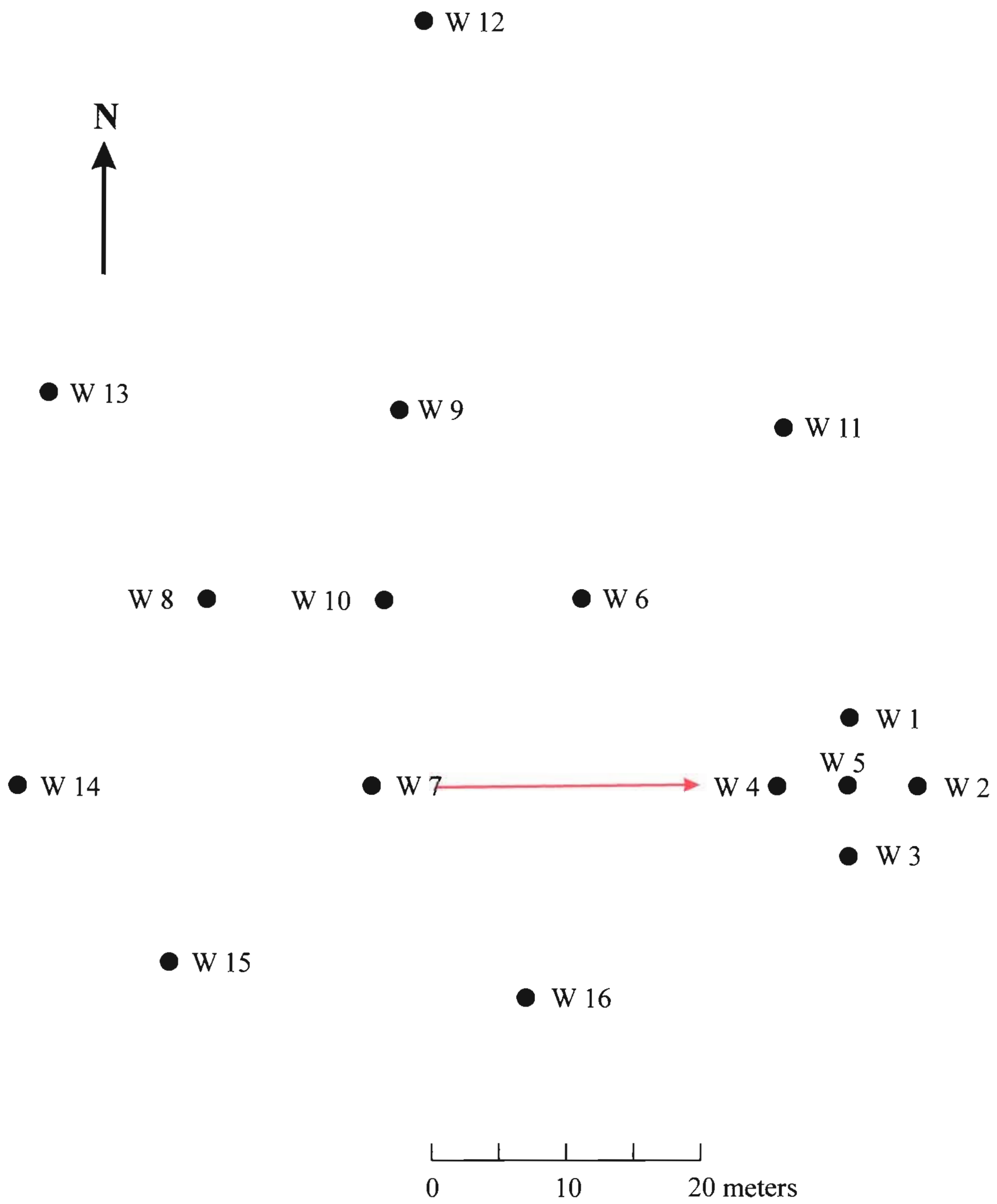
Appendix C.1.17
W7 - 12 July 2004
PL 4 - -6.28 to -7.78 masl
3.29E-5 cm/s



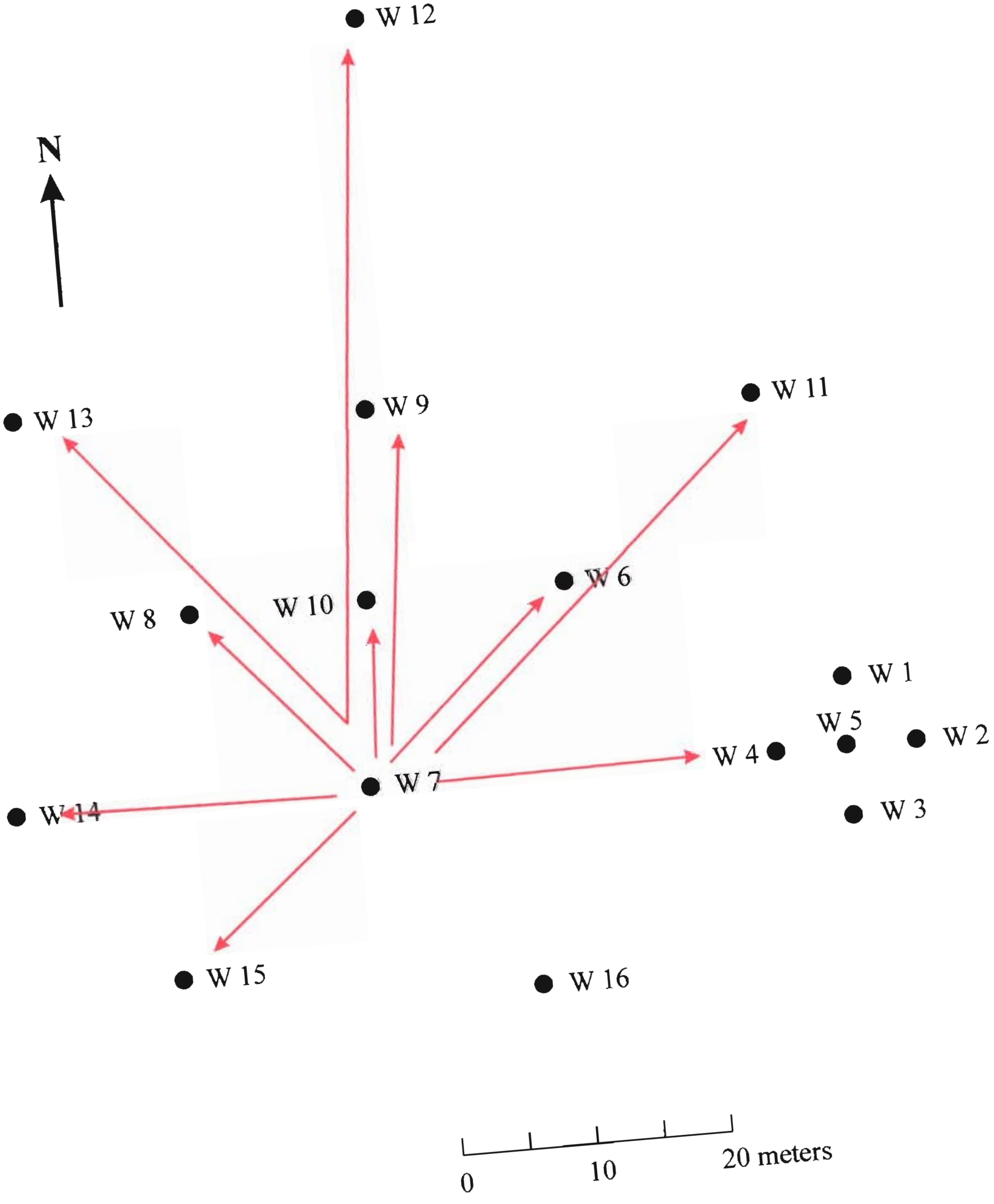
Appendix C.1.18
W7 - 12 July 2004
PL 5 - -8.03 to -9.53 masl
7.75E-5 cm/s



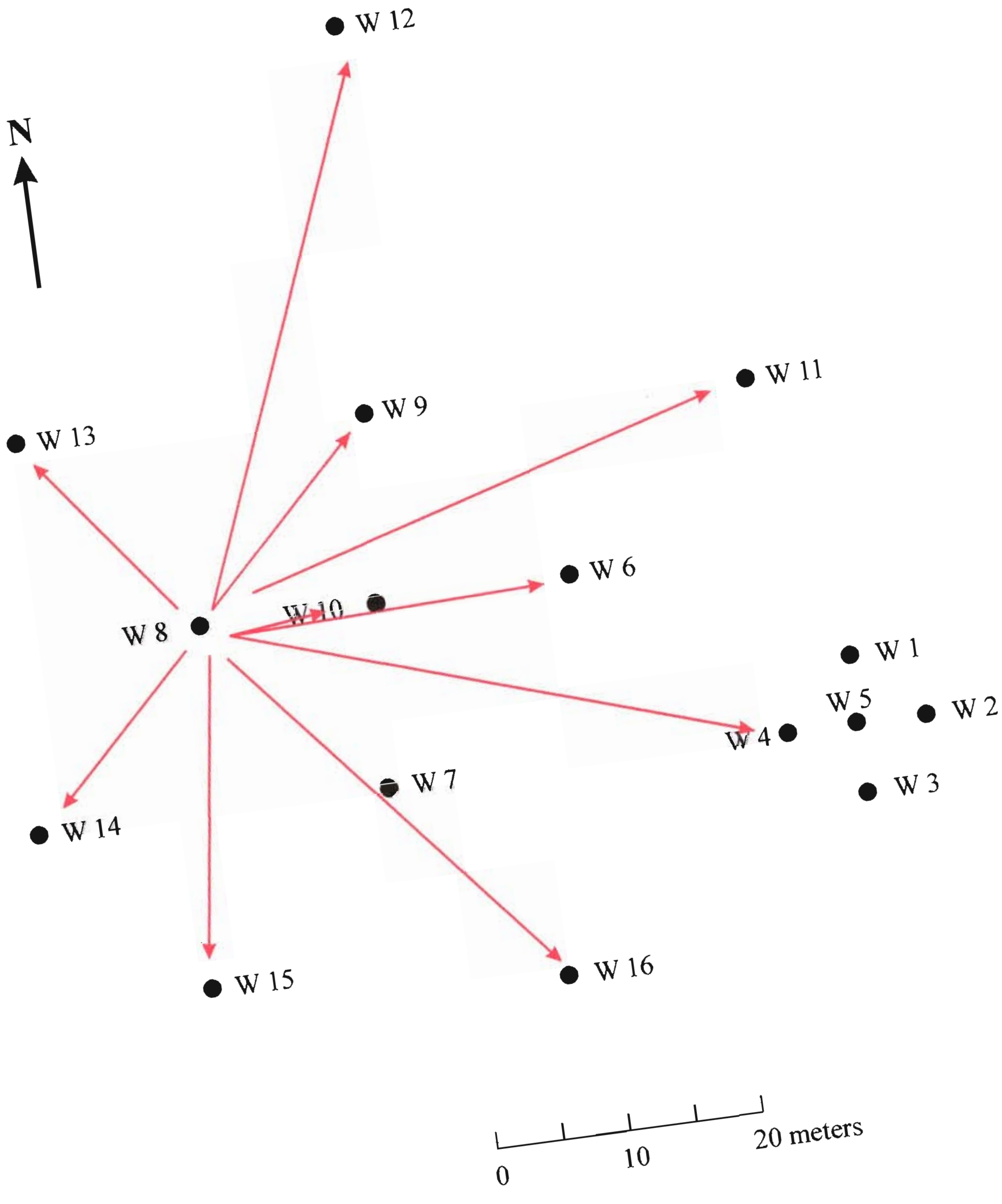
Appendix C.1.19
W7 - 12 July 2004
PL 6 - -9.78 to -12.28 masl
5.0E-3 cm/s



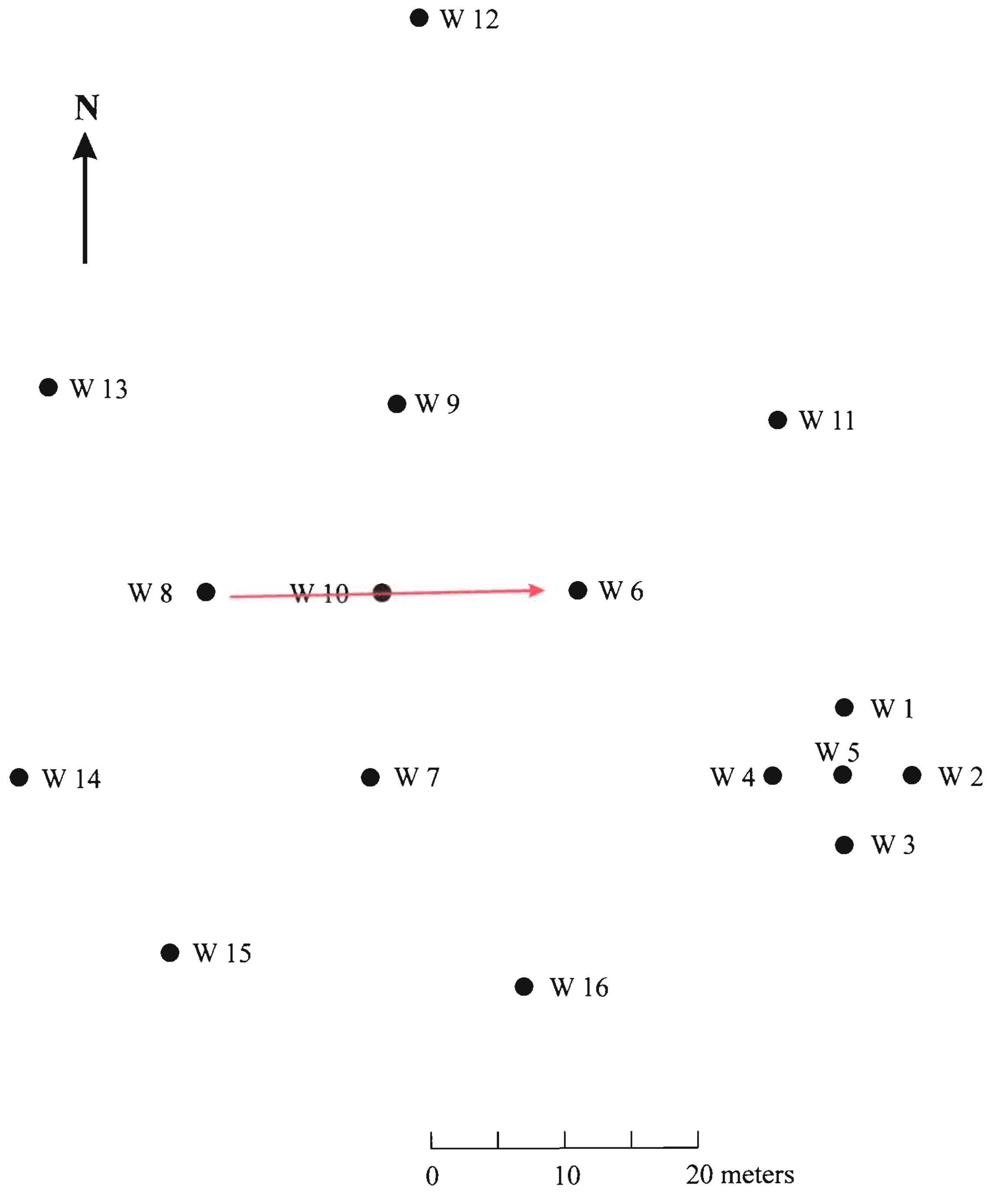
Appendix C.1.20
W7 - 12 July 2004
PL 7 - -14.03 to -15.53 masl
3.04E-4 cm/s



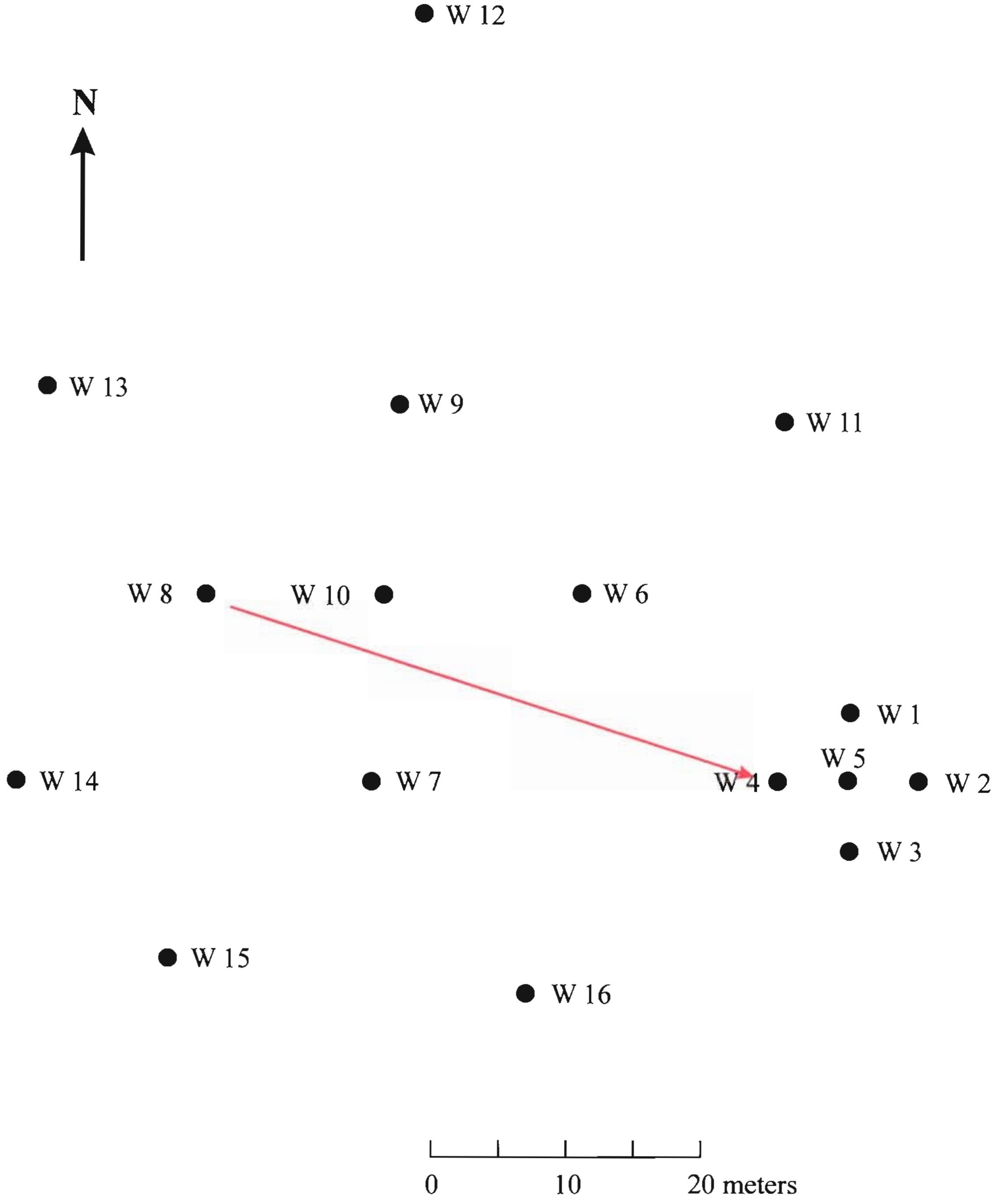
Appendix C.1.21
W8 - 08 Nov 2004
PL 1 - -7.18 to -46.68 masl
7.02E-5 cm/s



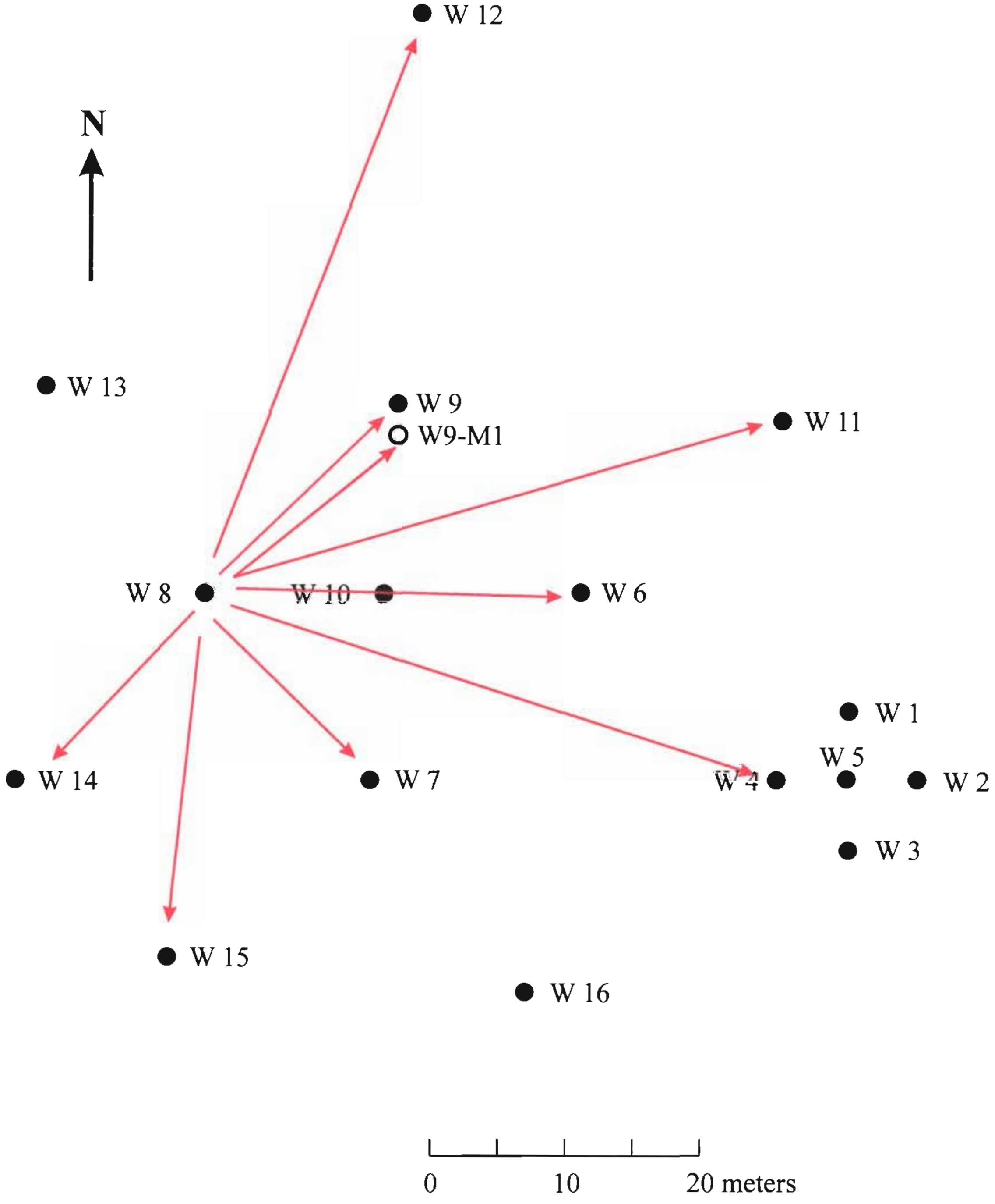
Appendix C.1.22
W8 - 08 Nov 2004
PL 2 - -6.93 to -8.43 masl
2.02E-6 cm/s



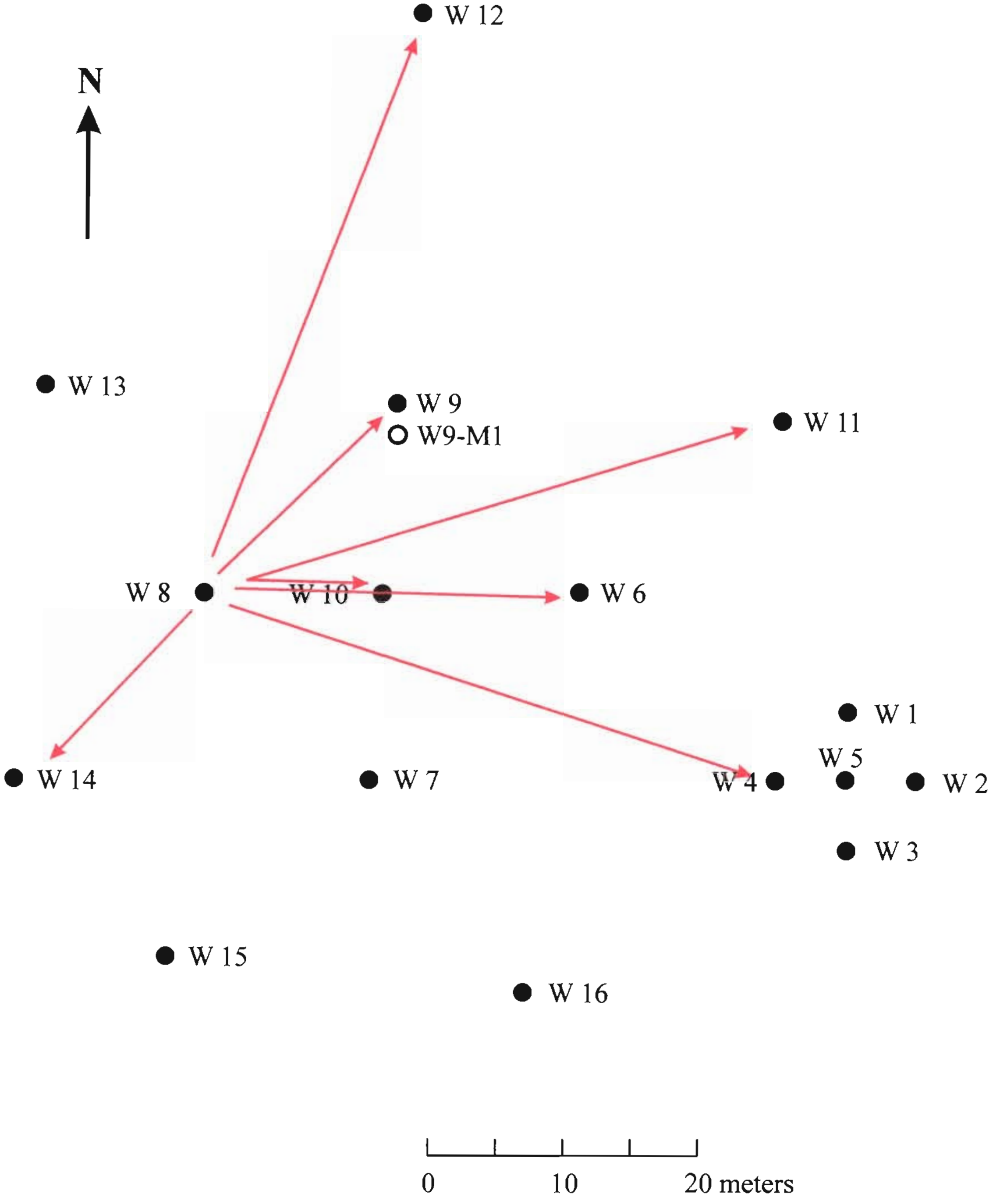
Appendix C.1.23
W8 - 08 Nov 2004
PL 3 - -11.43 to -12.93 masl
2.02E-6 cm/s



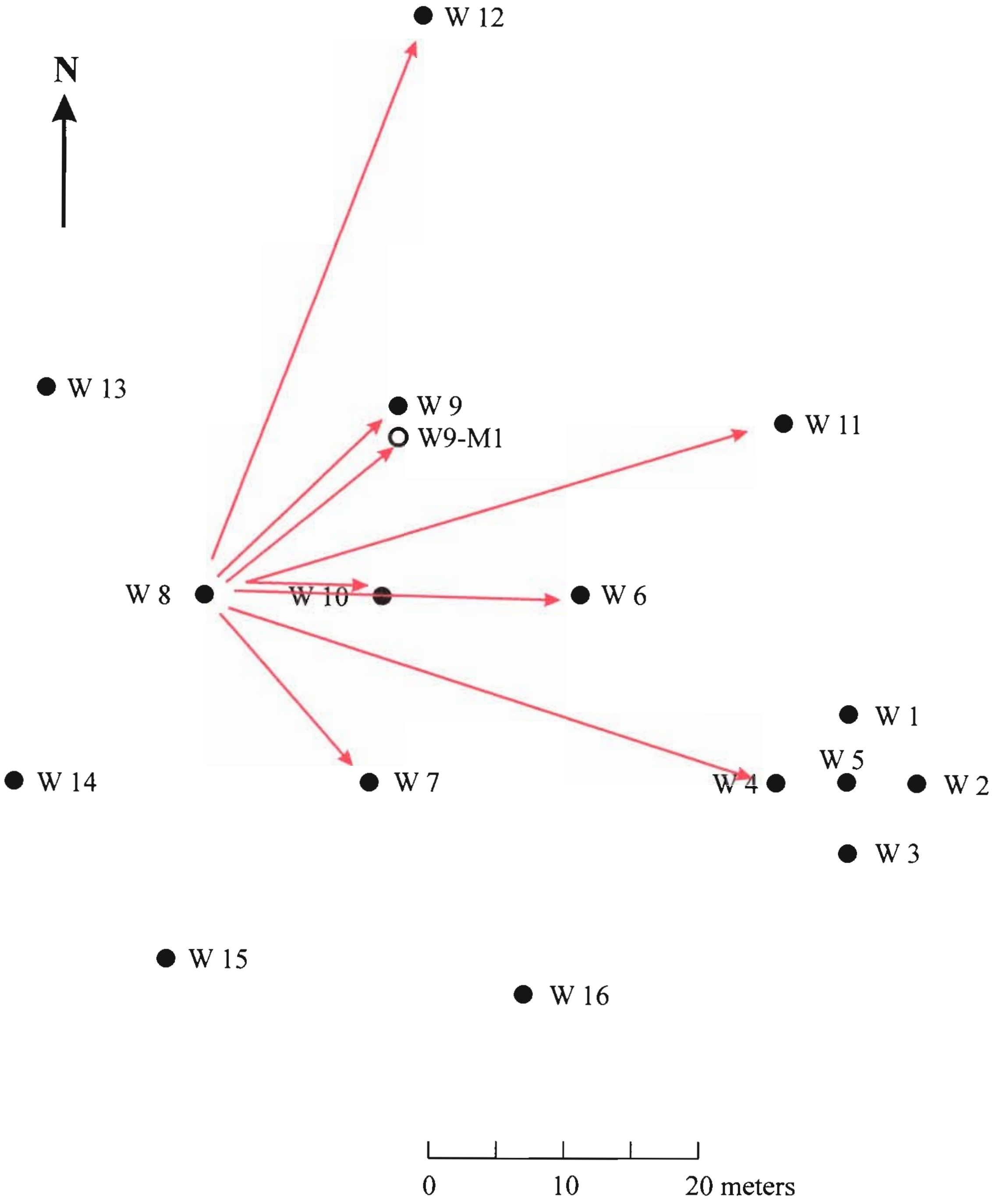
Appendix C.1.24
W8 - 08 Nov 2004
PL 4 - -12.68 to -14.18 masl
4.51E-5 cm/s



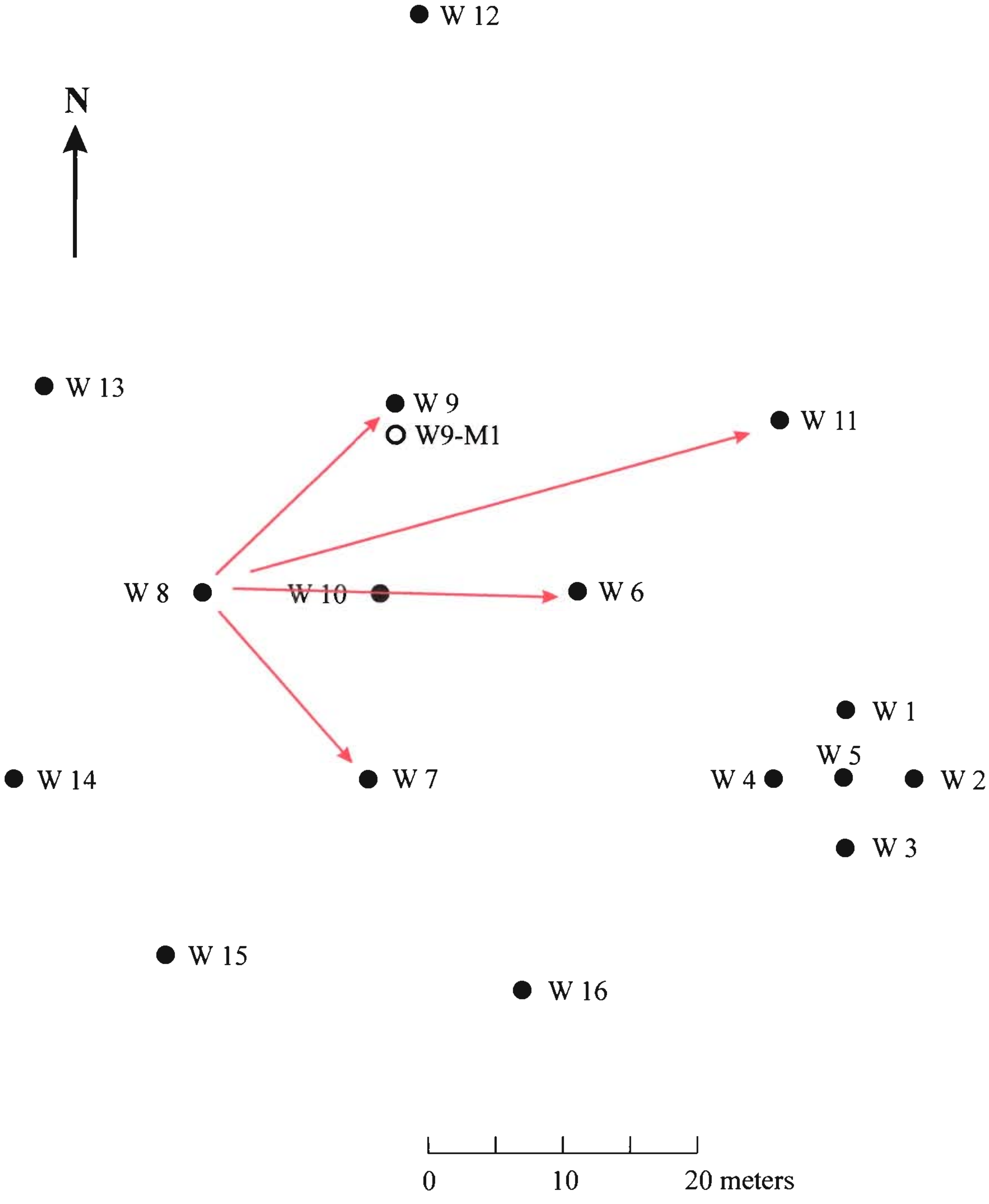
Appendix C.1.25
W8 - 08 Nov 2004
PL 5 - -14.18 to -15.68 masl
2.26E-5 cm/s



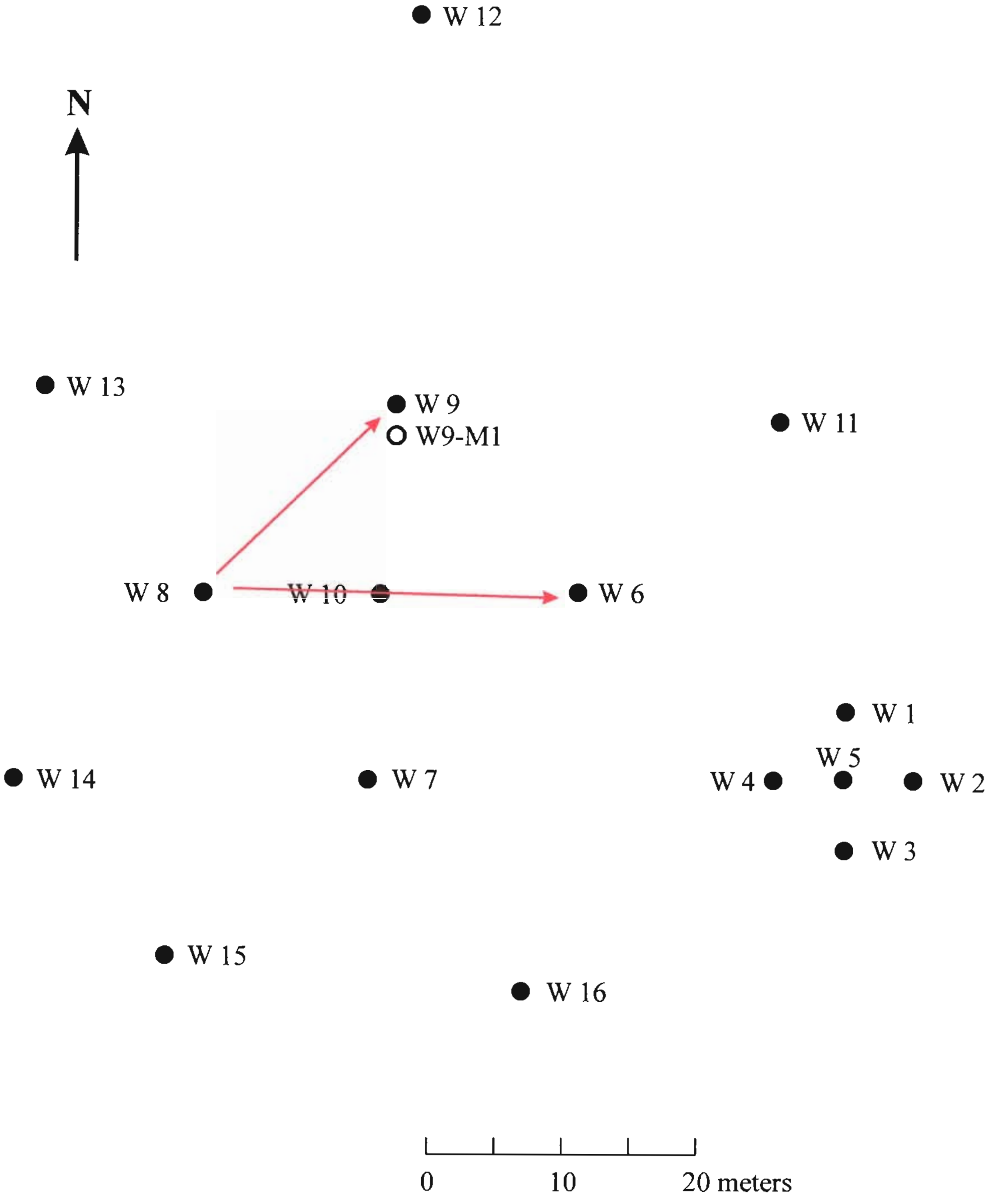
Appendix C.1.26
W8 - 08 Nov 2004
PL 6 - -15.68 to -17.18 masl
1.0E-6 cm/s



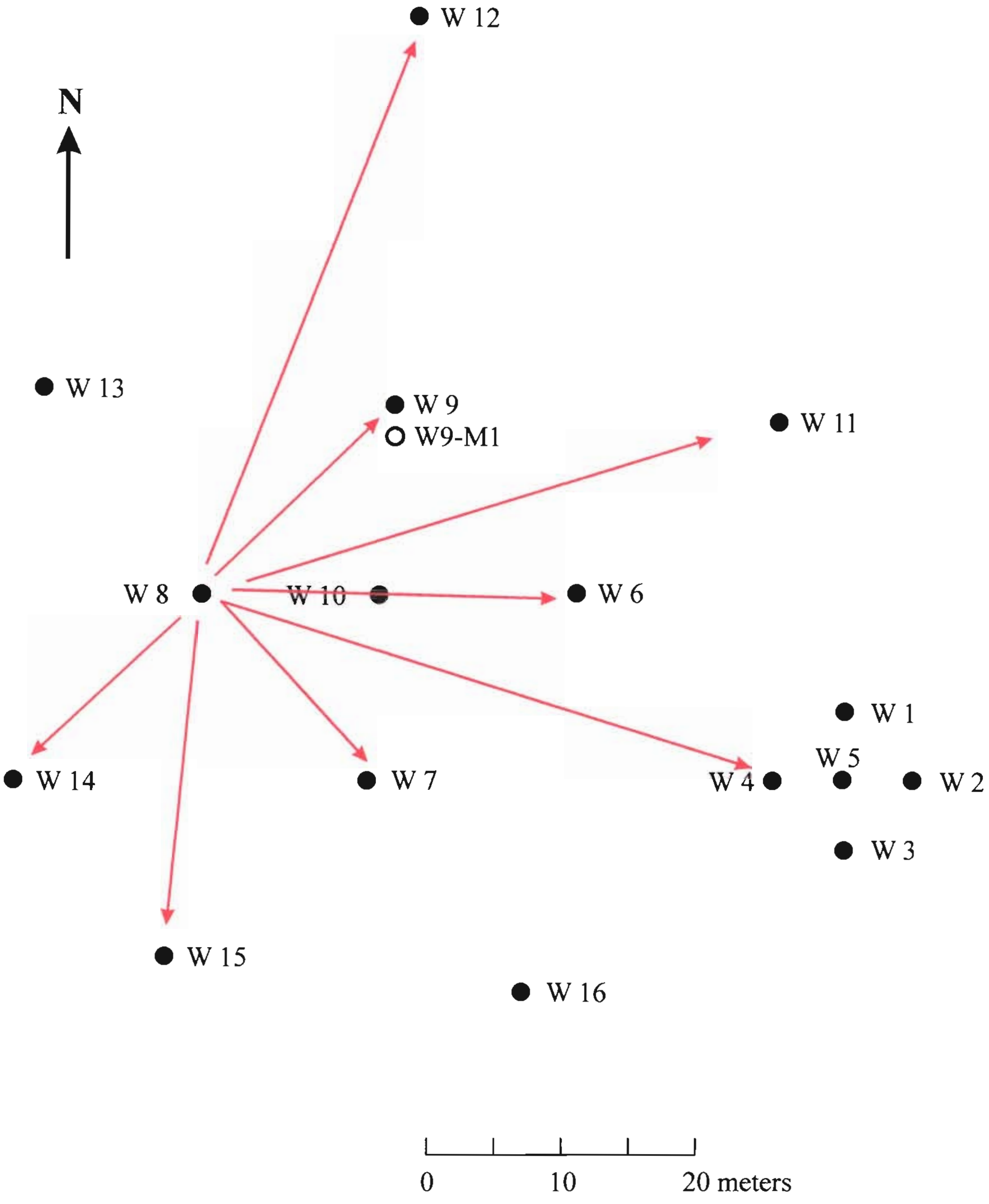
Appendix C.1.27
W8 - 08 Nov 2004
PL 7 - -16.18 to -17.68 masl
1.0E-6 cm/s



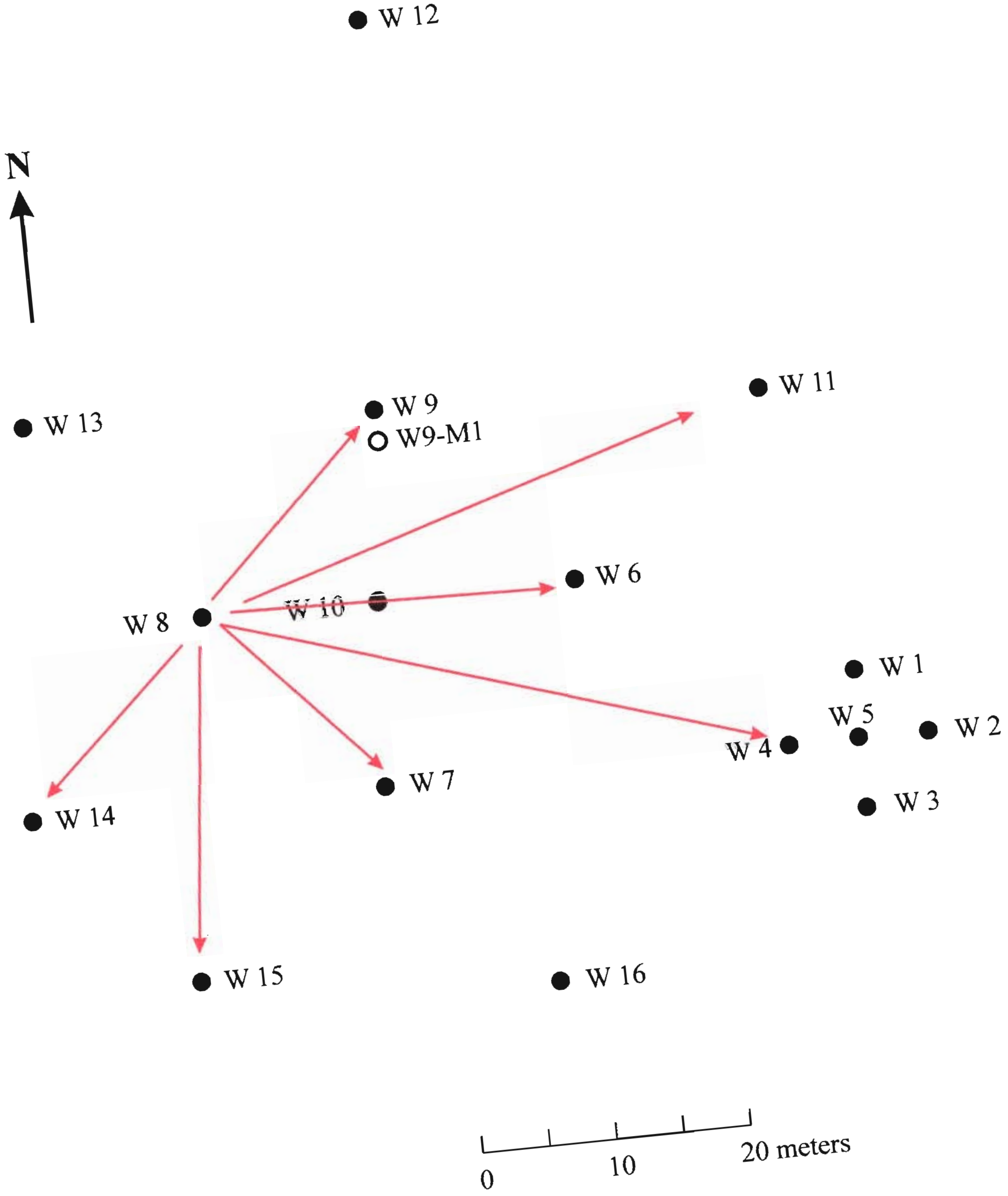
Appendix C.1.28
W8 - 08 Nov 2004
PL 8 - -20.18 to -21.68 masl
4.03E-6 cm/s



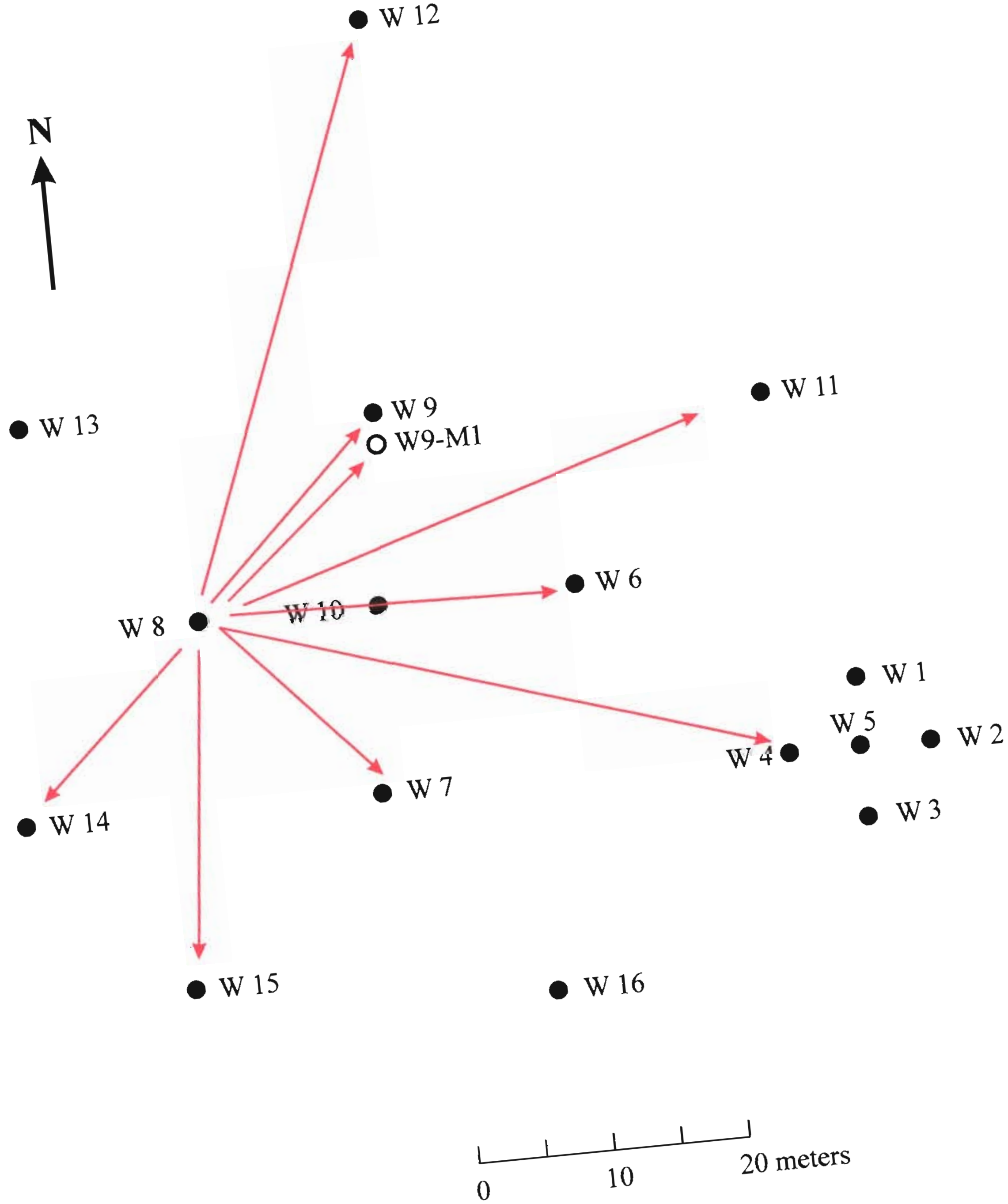
Appendix C.1.29
W8 - 08 Nov 2004
PL 9 - -36.18 to -37.68 masl
1.01E-5 cm/s



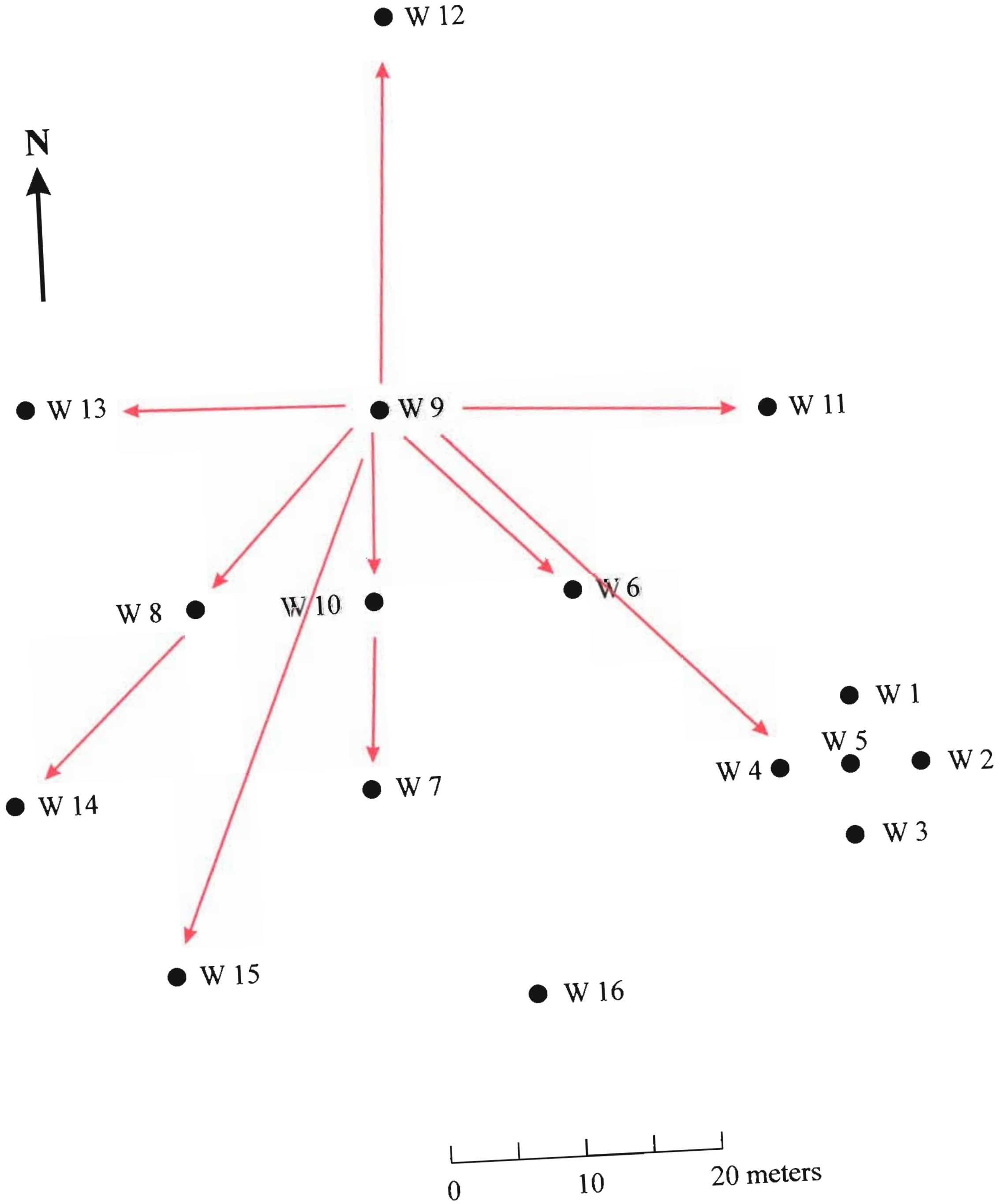
Appendix C.1.30
W8 - 08 Nov 2004
PL 10 - -39.18 to -40.68 masl
5.14E-5 cm/s



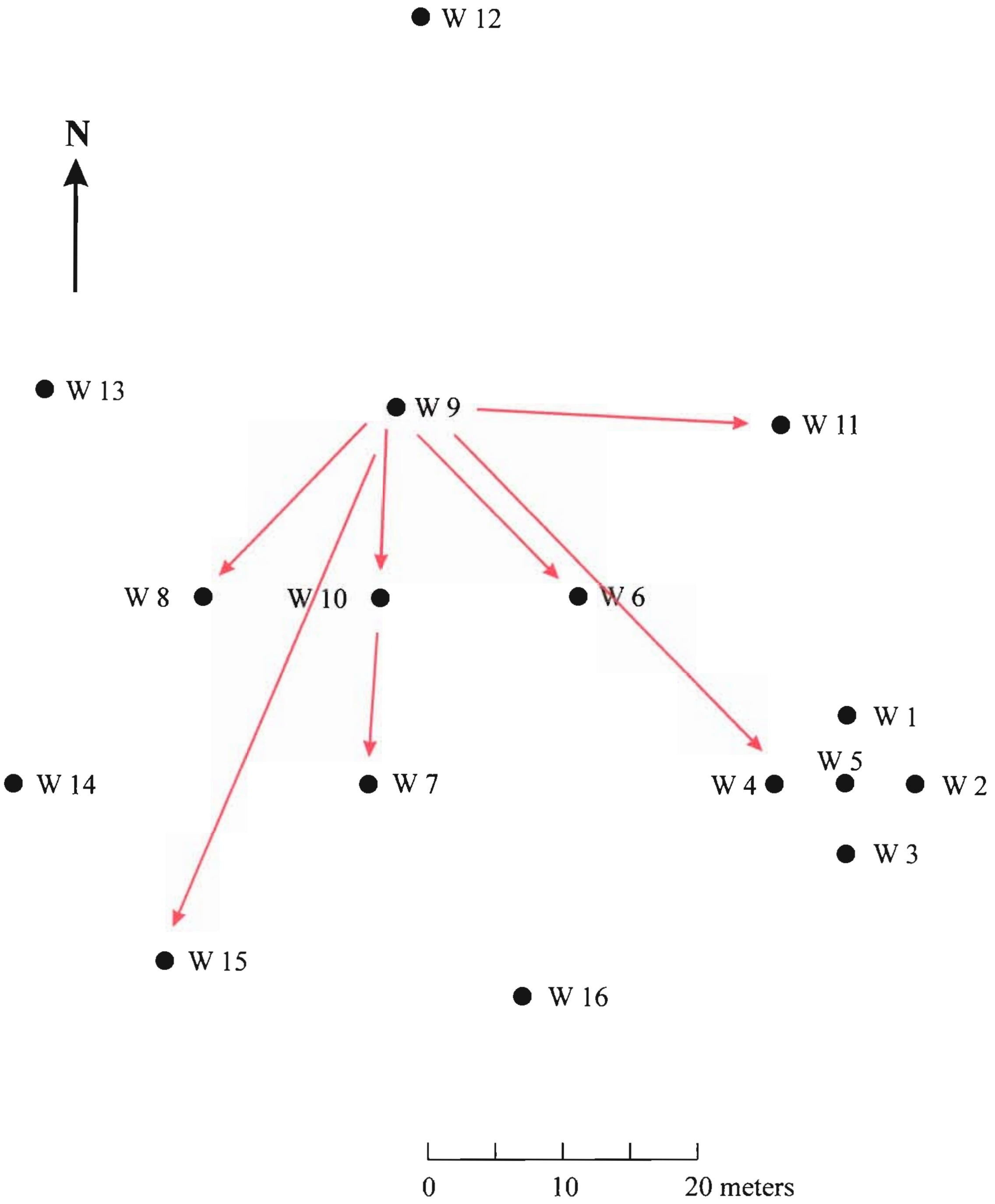
Appendix C.1.31
W8 - 08 Nov 2004
PL 11 - -42.18 to -43.68 masl
9.0E-5 cm/s



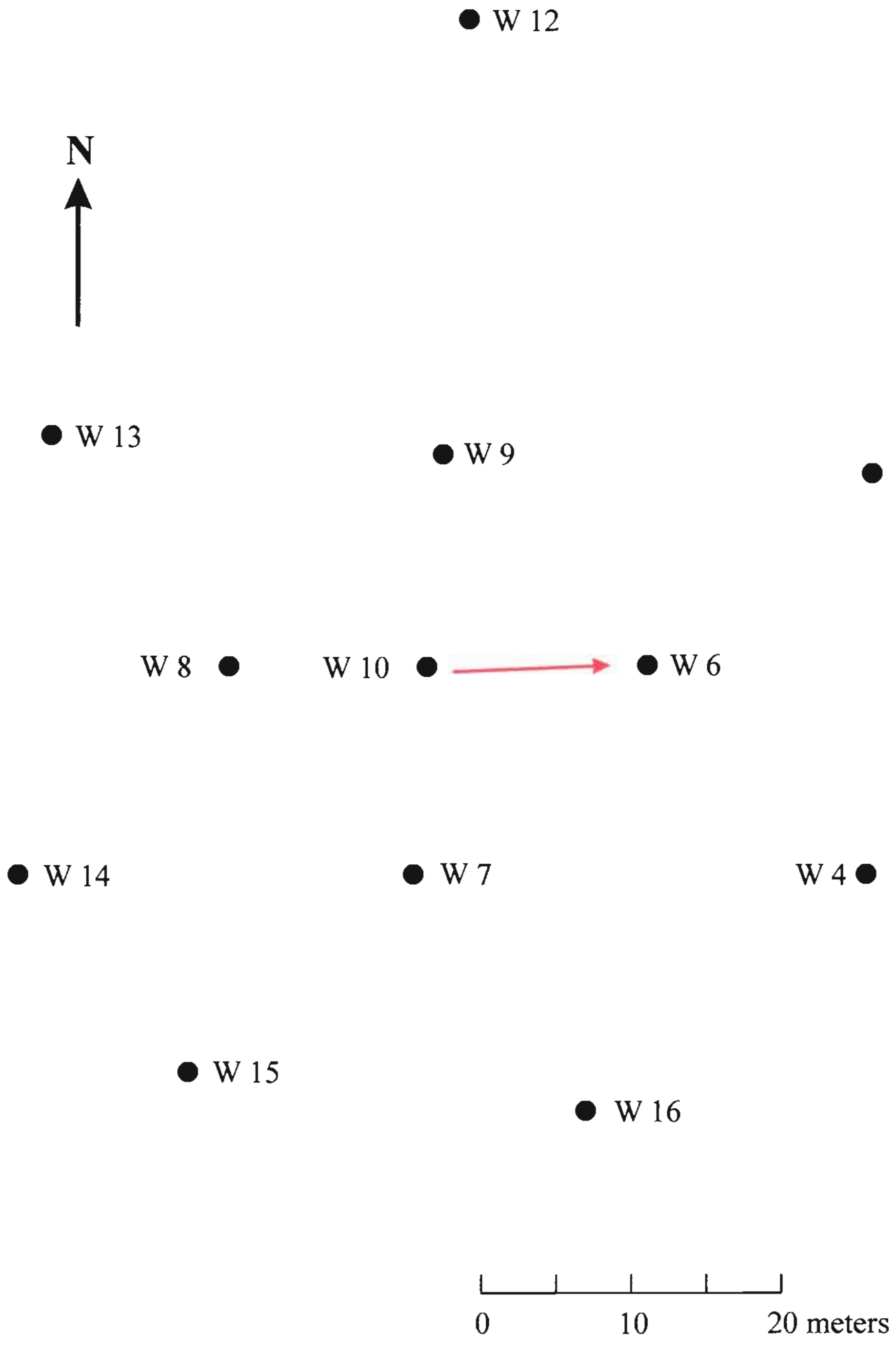
Appendix C.1.32
W9 - 22 June 2004
PL 1 -35.26 to -36.76 masl
3.77E-4 cm/s



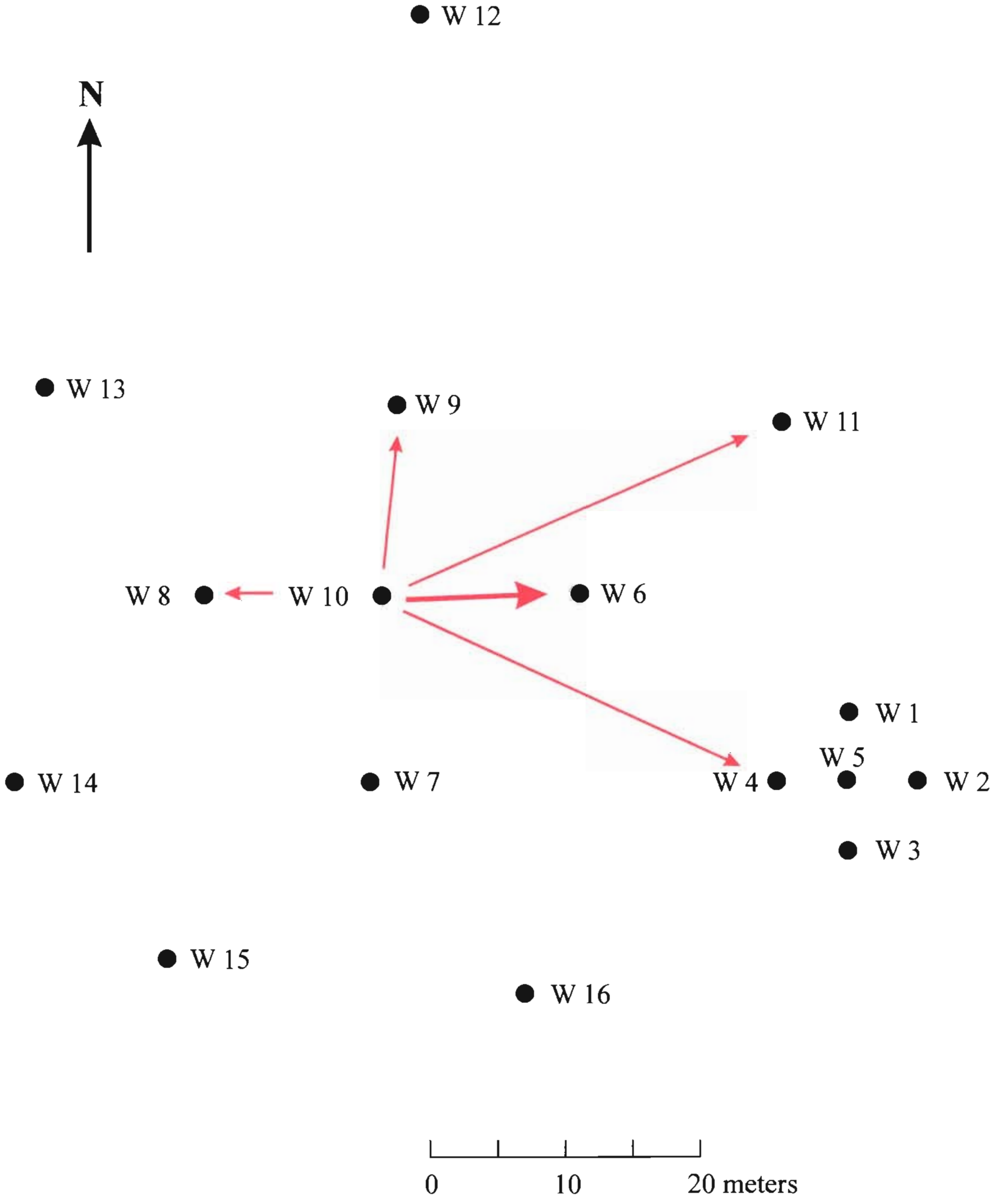
Appendix C.1.33
W9 - 22 June 2004
PL 2 -44.46 to -45.96 masl
3.75E-4 cm/s



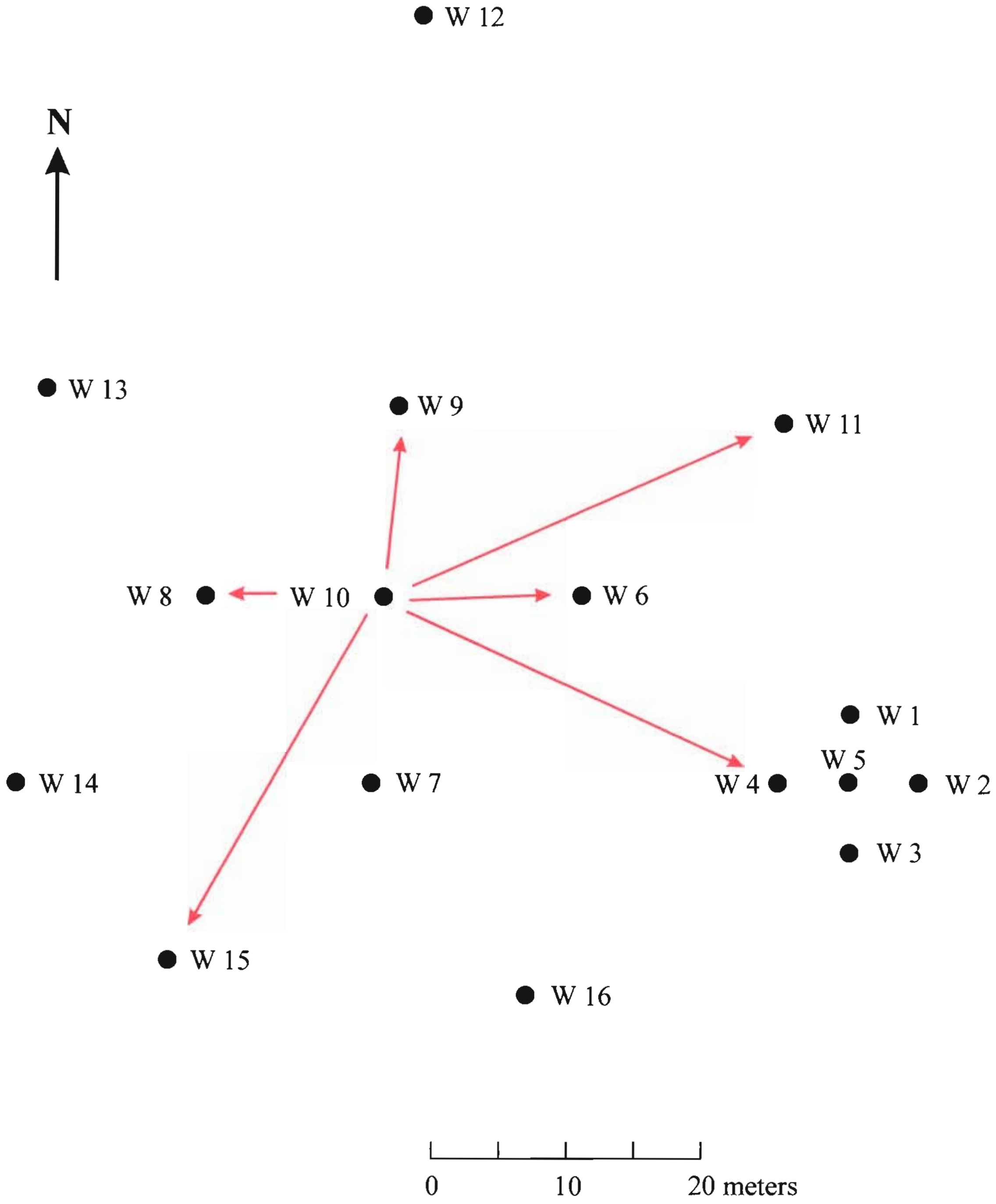
Appendix C.1.34
W10 - 18 June 2004 & 22 Aug 2004
PL 1- 10.44 to 8.94 masl
1.57E-6 cm/s



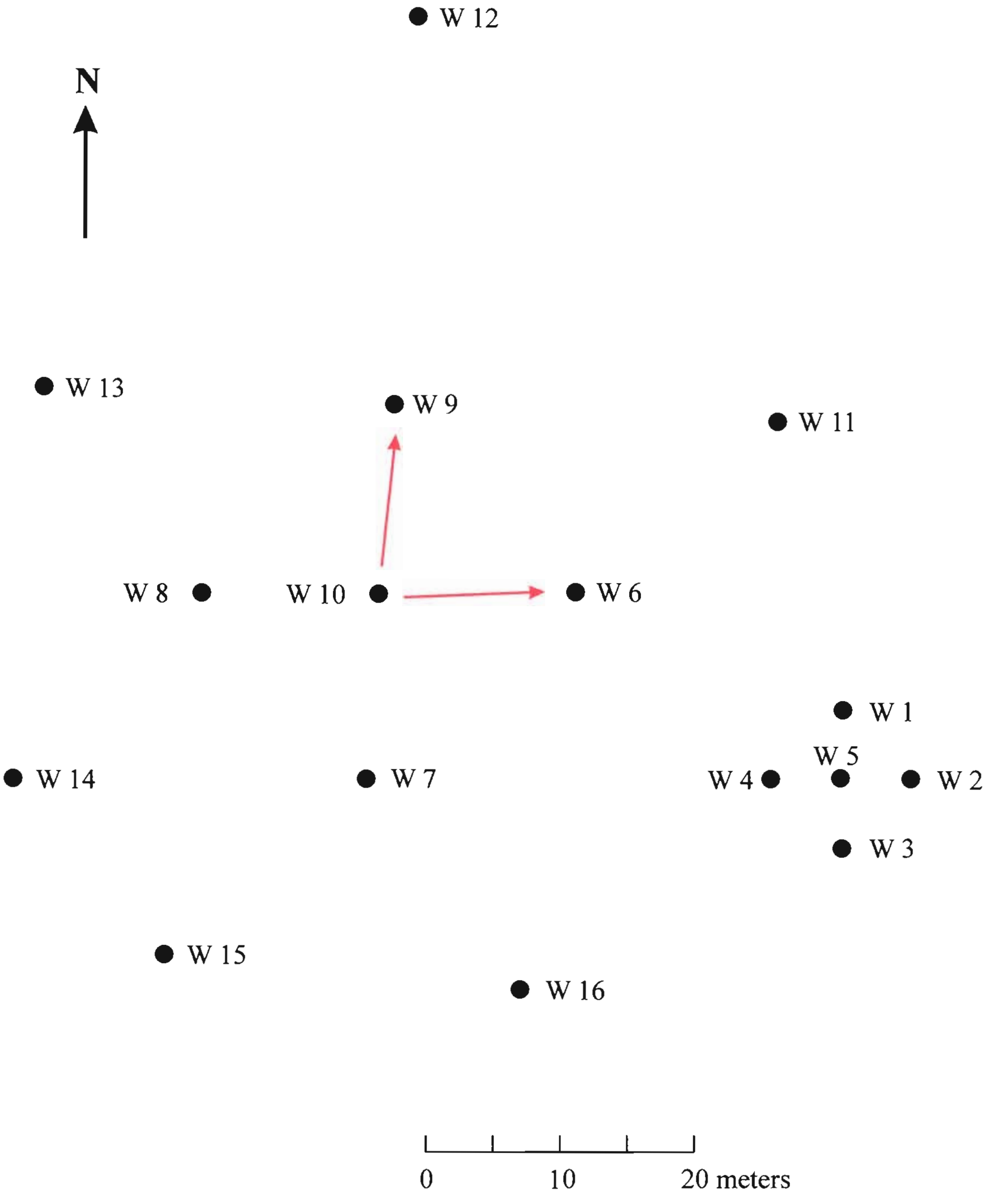
Appendix C.1.35
W10 - 18 June 2004 & 22 Aug 2004
PL 2 - 7.54 to 6.04 masl
4.03E-6 cm/s



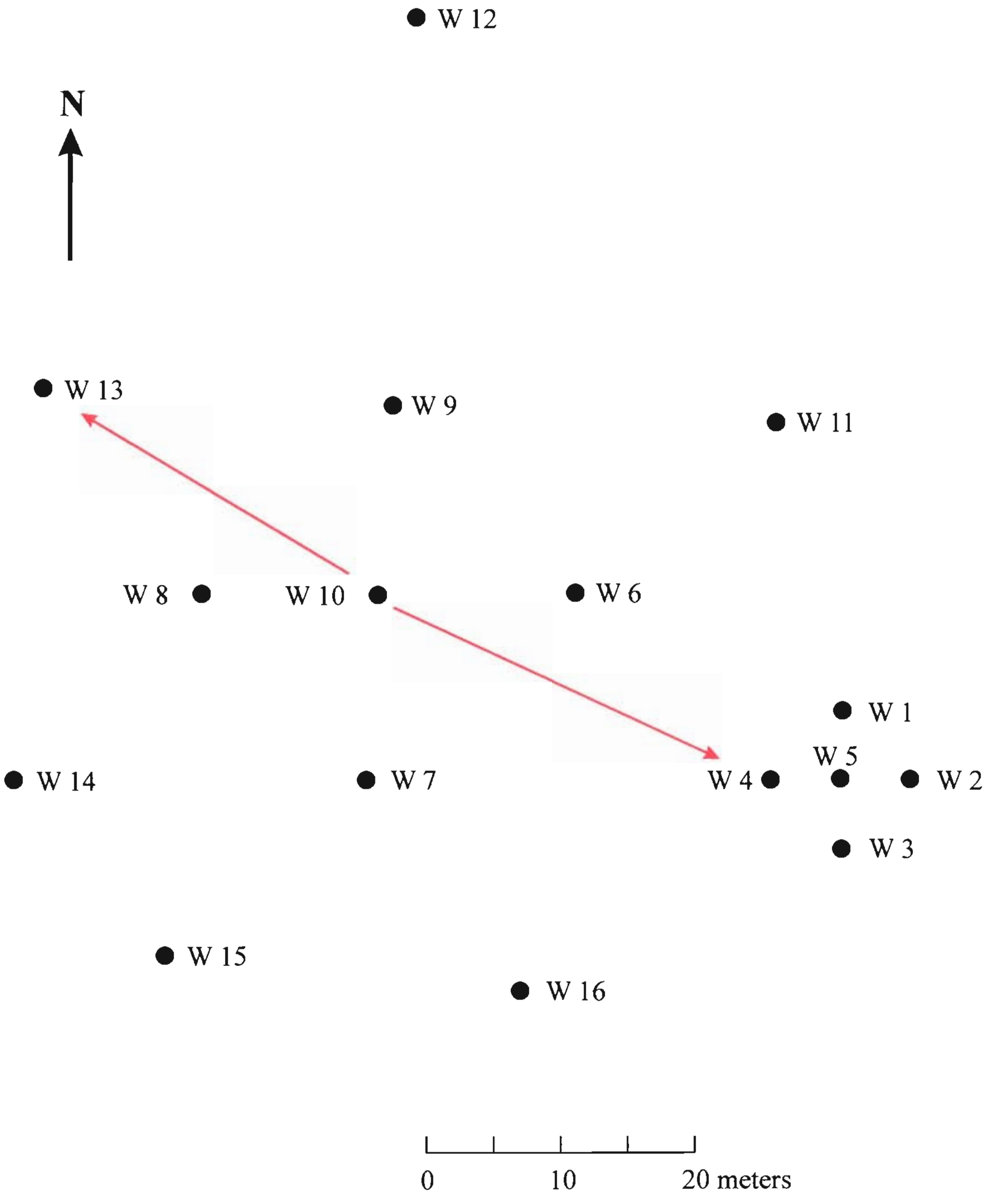
Appendix C.1.36
W10 - 18 June 2004 & 22 Aug 2004
PL 3 - 4.74 to 3.24 masl
4.11E-5 cm/s



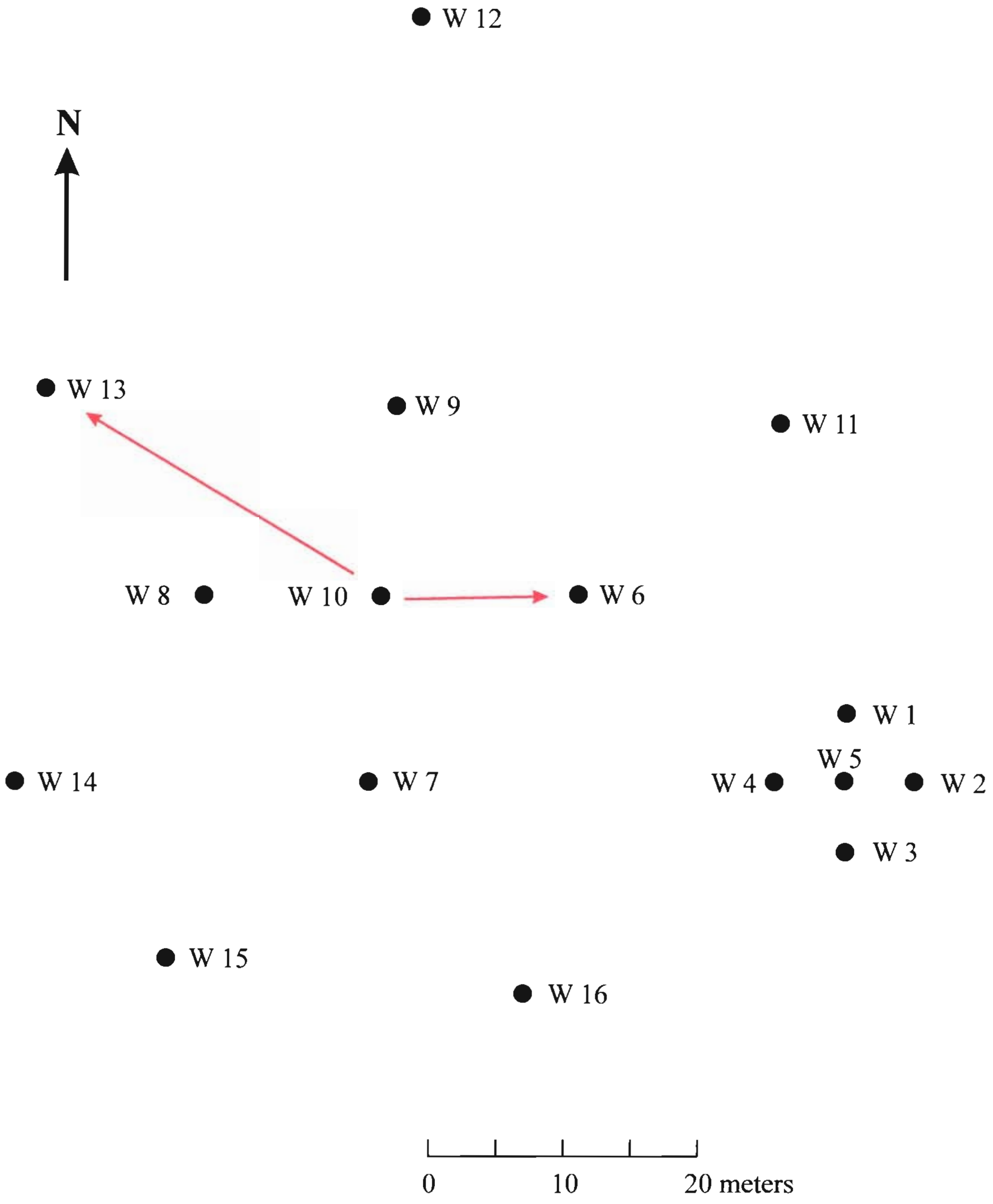
Appendix C.1.37
W10 - 18 June 2004 & 22 Aug 2004
PL 4 - 1.74 to 0.24 masl
2.72E-5 cm/s



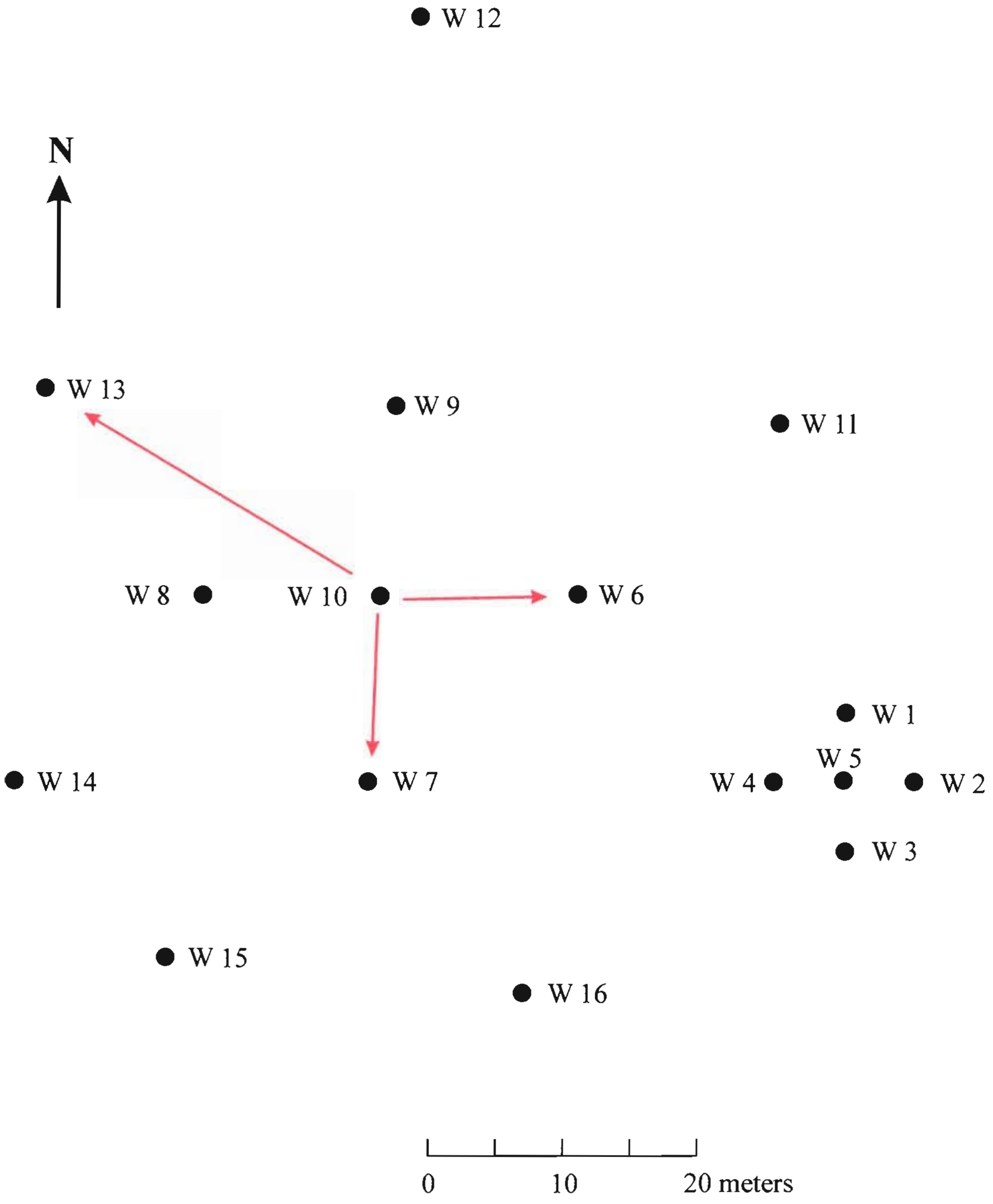
Appendix C.1.38
W10 - 18 June 2004 & 22 Aug 2004
PL 5 - -1.26 to -2.76 masl
2.54E-5 cm/s



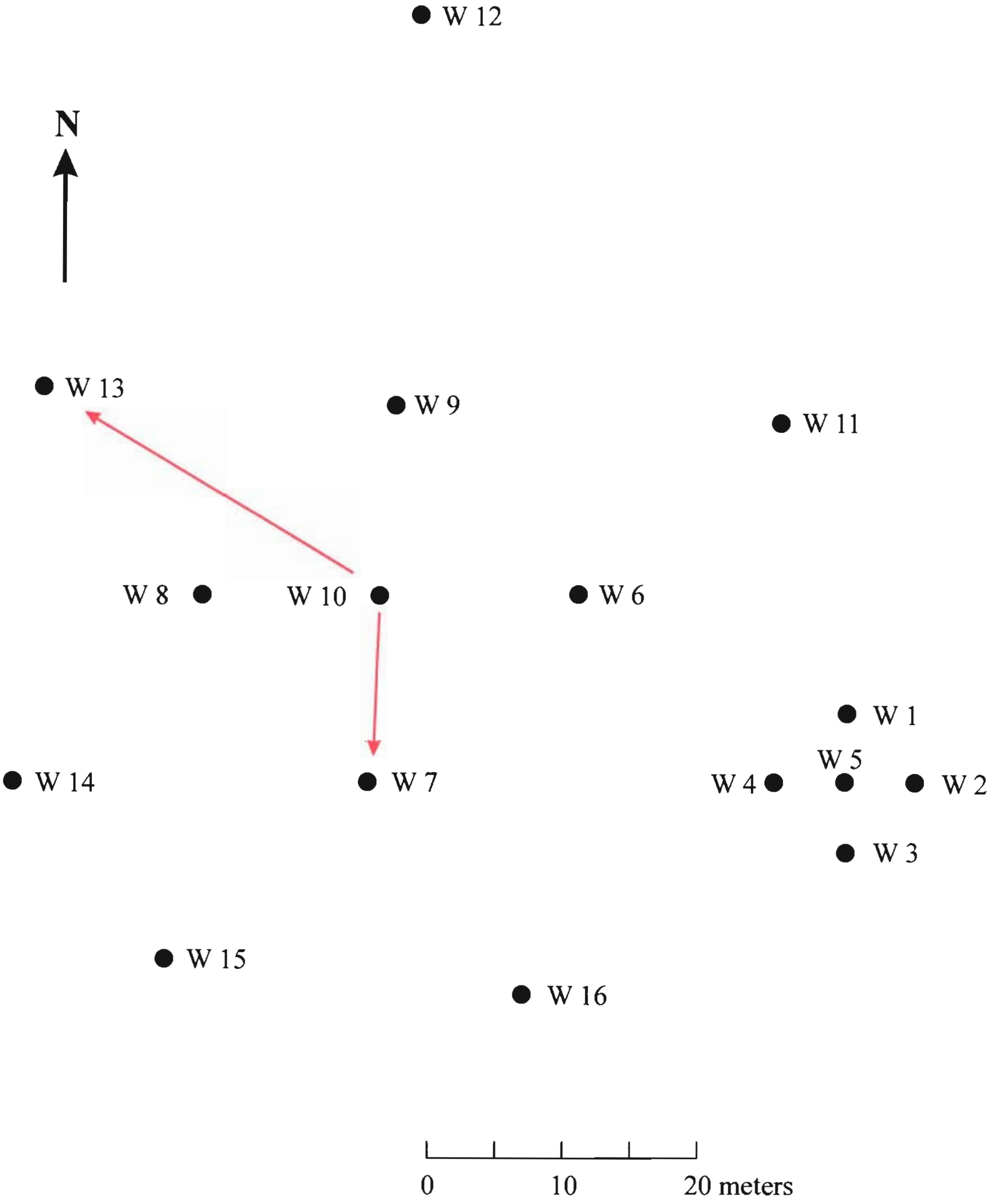
Appendix C.1.39
W10 - 18 June 2004 & 22 Aug 2004
PL 6 - -7.26 to -8.76 masl
4.01E-6 cm/s



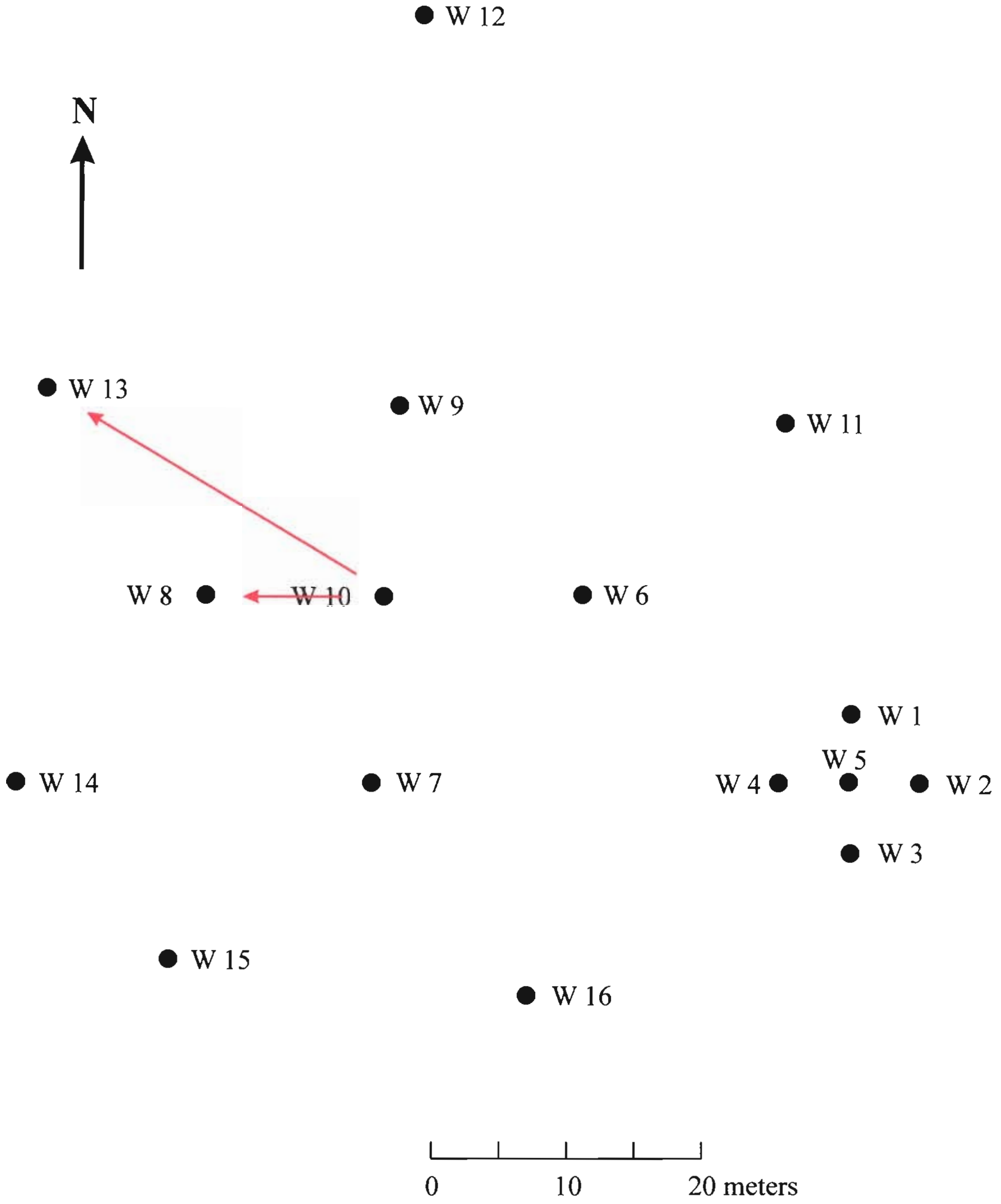
Appendix C.1.40
W10 - 18 June 2004 & 22 Aug 2004
PL 7 - -7.01 to -8.26 masl
1.31E-6 cm/s



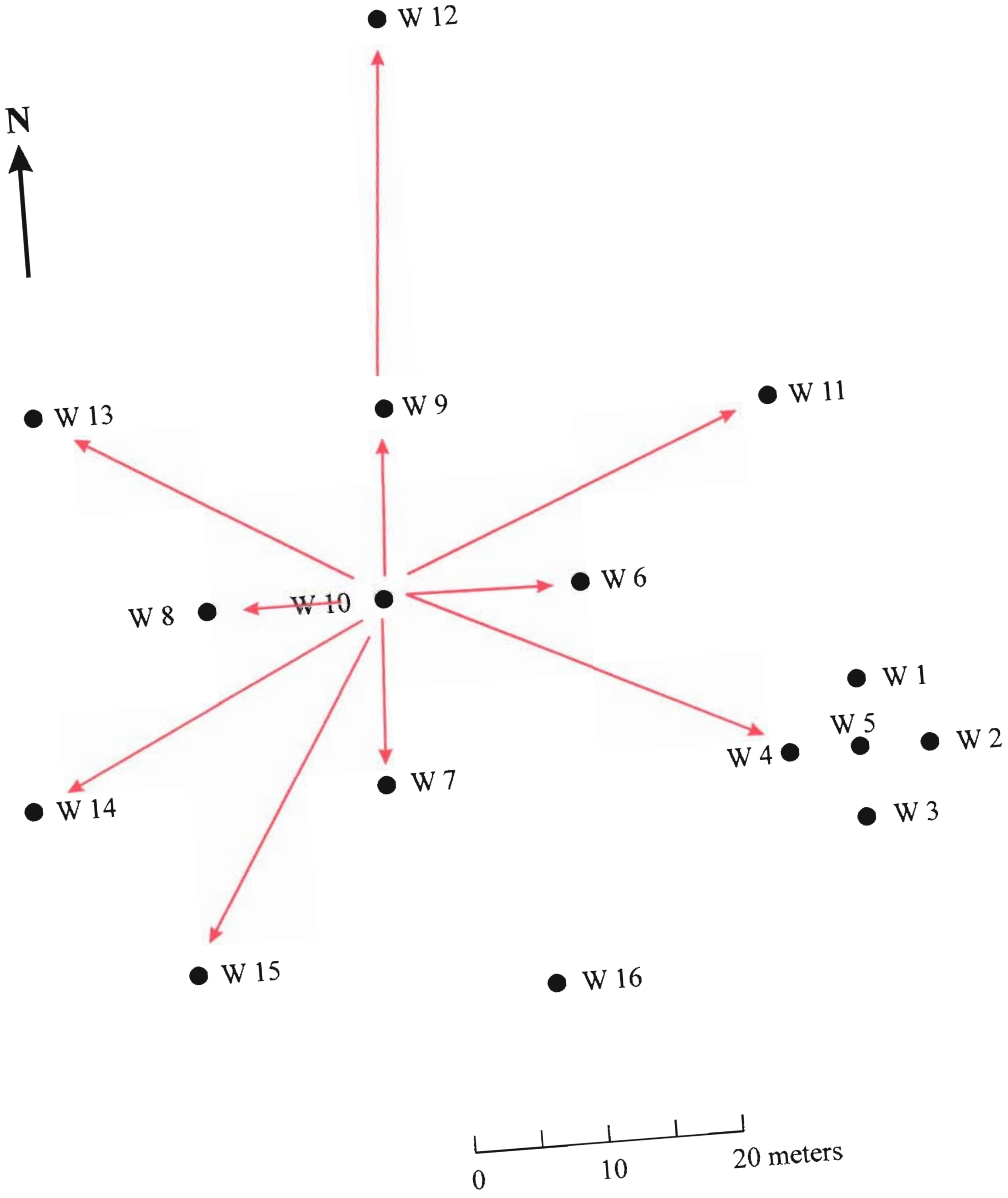
Appendix C.1.41
W10 - 18 June 2004 & 22 Aug 2004
PL 8 - -34.76 to -36.26 masl
4.51E-5 cm/s



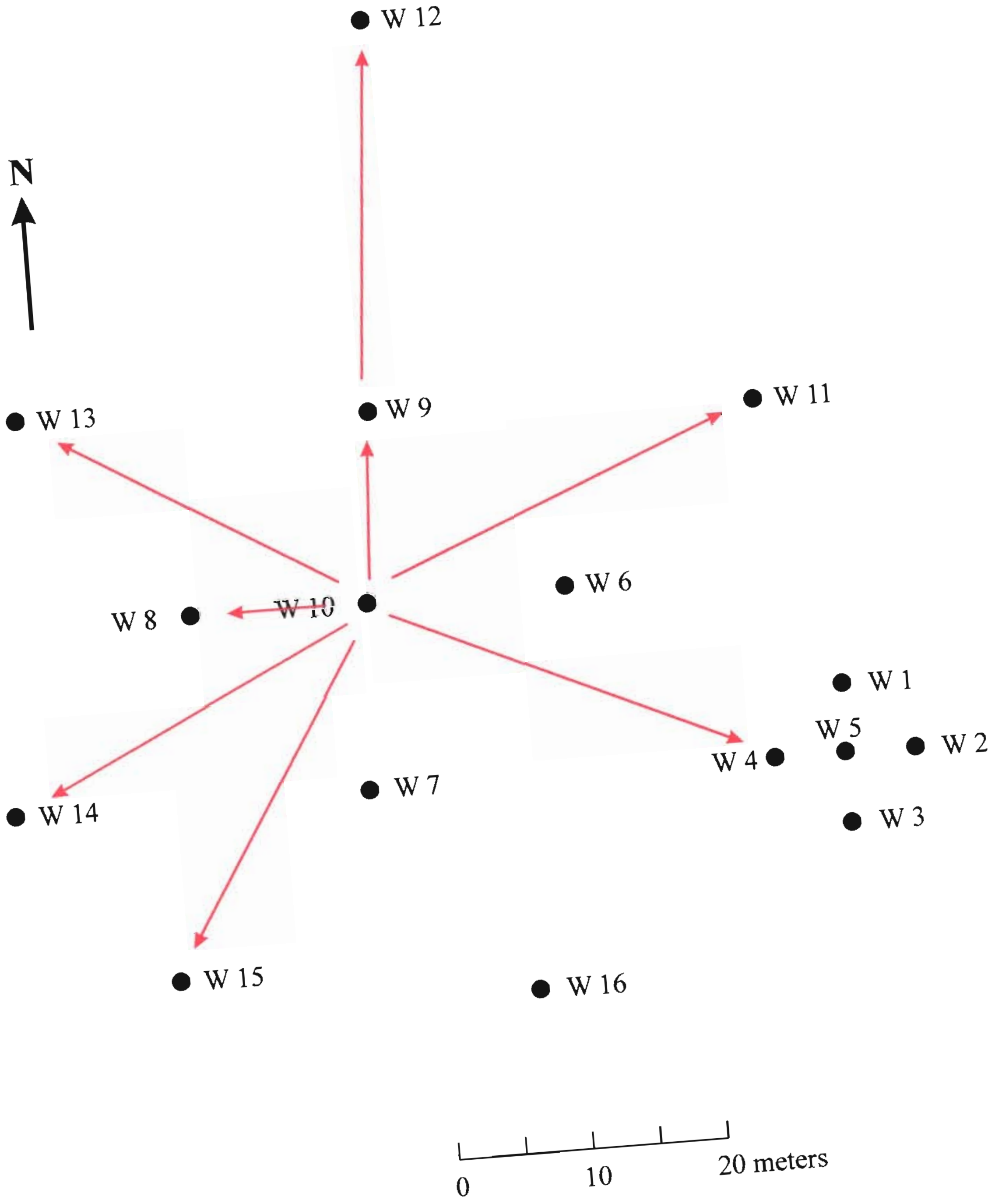
Appendix C.1.42
W10 - 18 June 2004 & 22 Aug 2004
PL 9 - -36.26 to -37.76 masl
4.40E-5 cm/s



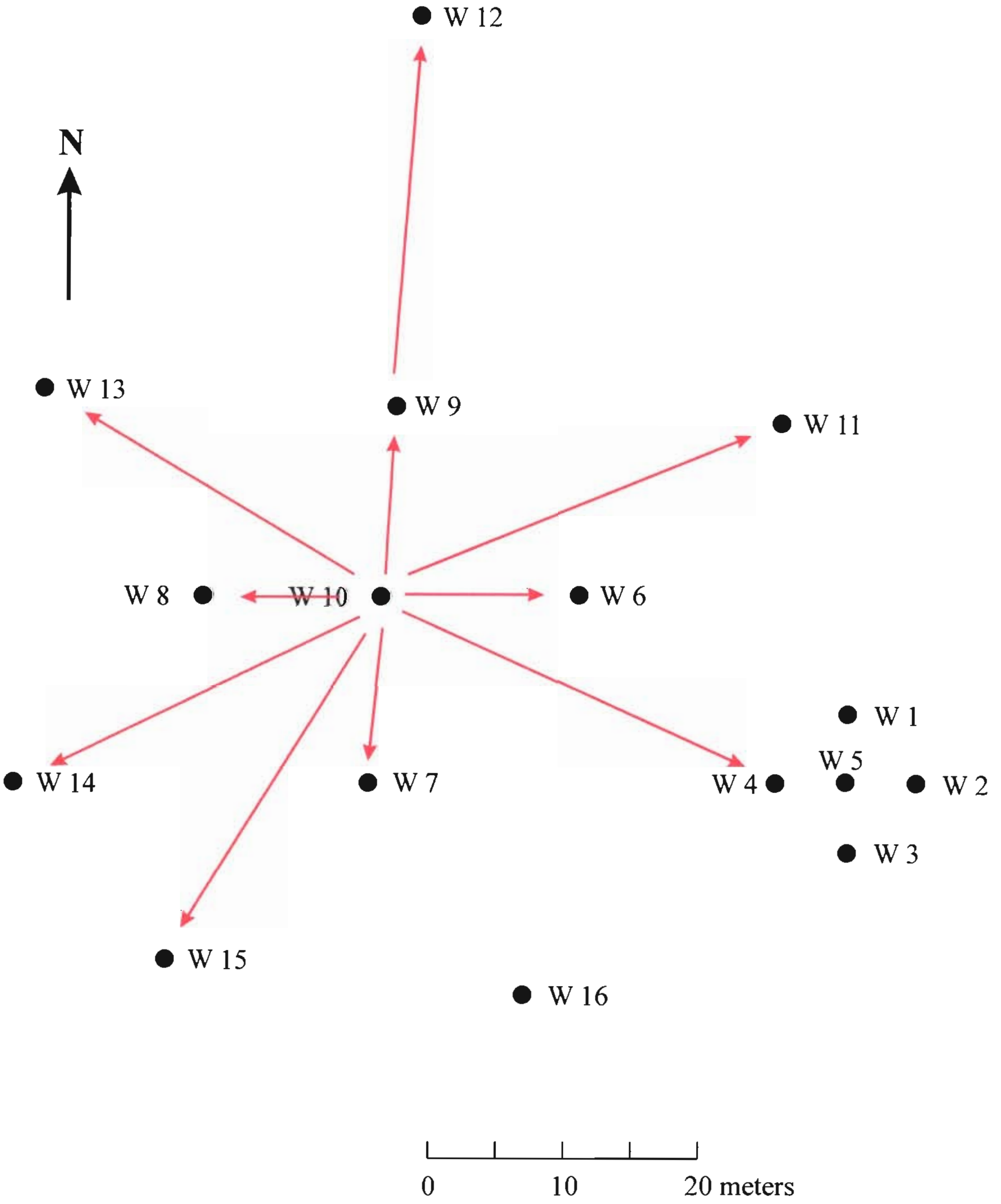
Appendix C.1.43
W10 - 18 June 2004 & 22 Aug 2004
PL 10 - -41.01 to -42.51 masl
2.07E-4 cm/s



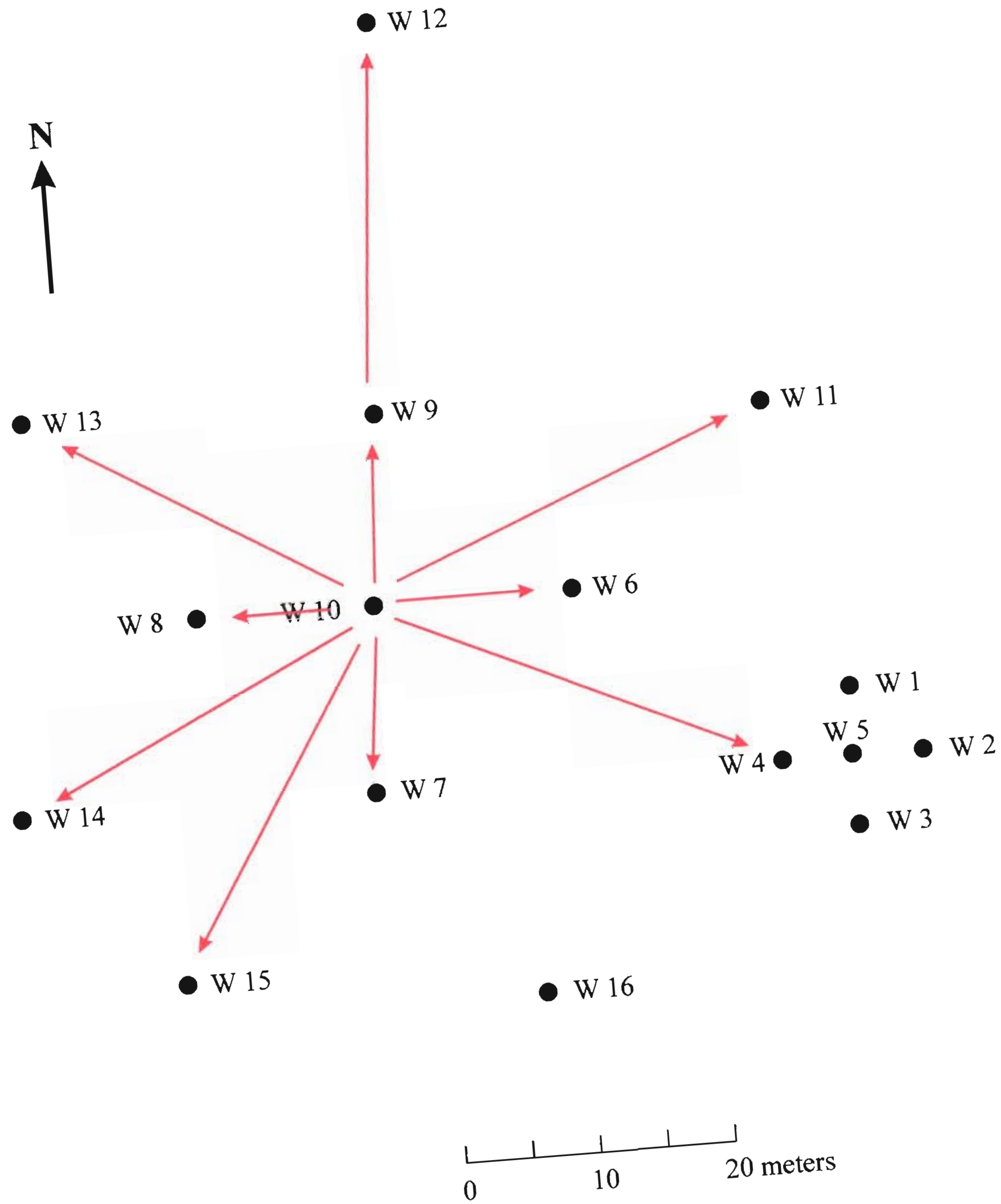
Appendix C.1.44
W10 - 18 June 2004 & 22 Aug 2004
PL 11 - -42.26 to -43.76 masl
2.07E-4 cm/s



Appendix C.1.45
W10 - 18 June 2004 & 22 Aug 2004
PL 12 - -44.01 to -45.51 masl
6.09E-6 cm/s



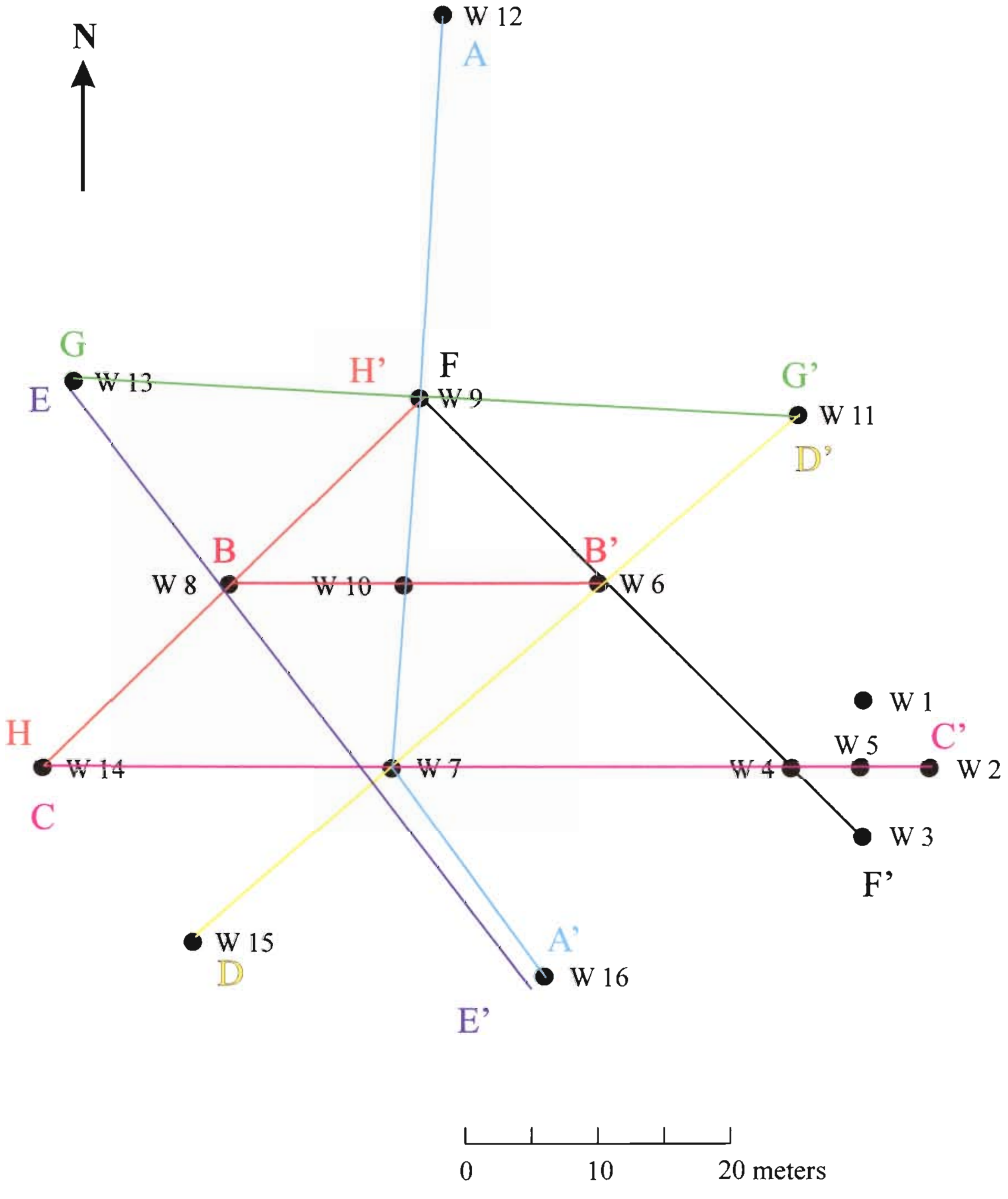
Appendix C.1.46
W10 - 18 June 2004 & 22 Aug 2004
PL 13 - -51.01 to -52.51 masl
2.73E-5 cm/s



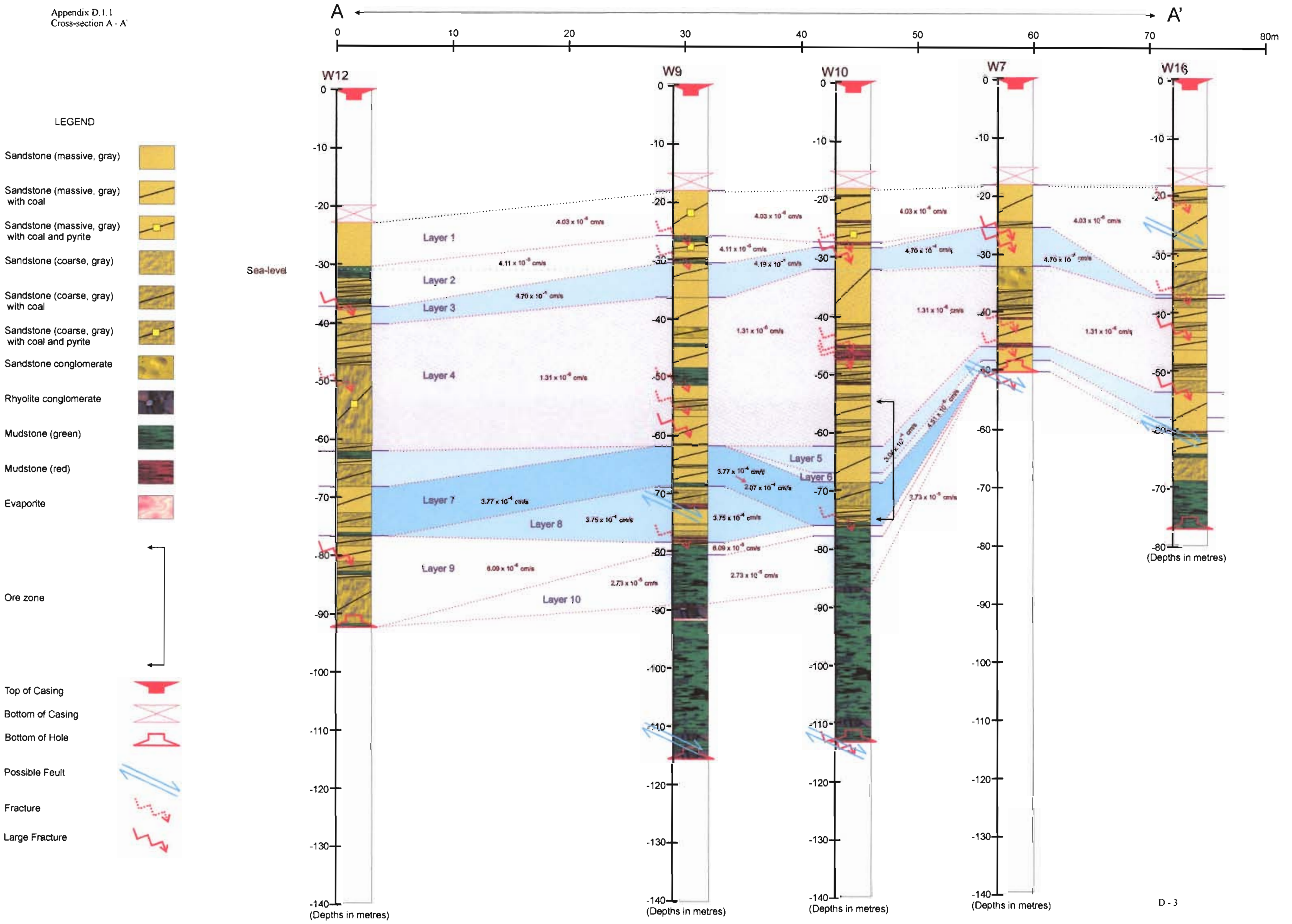
Hydraulic Conductivity Cross-sections

Hydraulic conductivity were cross-sectioned through the Yava Silver Mine Formation sandstones in order to layers of higher conductivity, assumed to contain fractures. Page D – 2 is a plan map showing the extents of the cross-sections listed as D.1.1 to D.1.8.

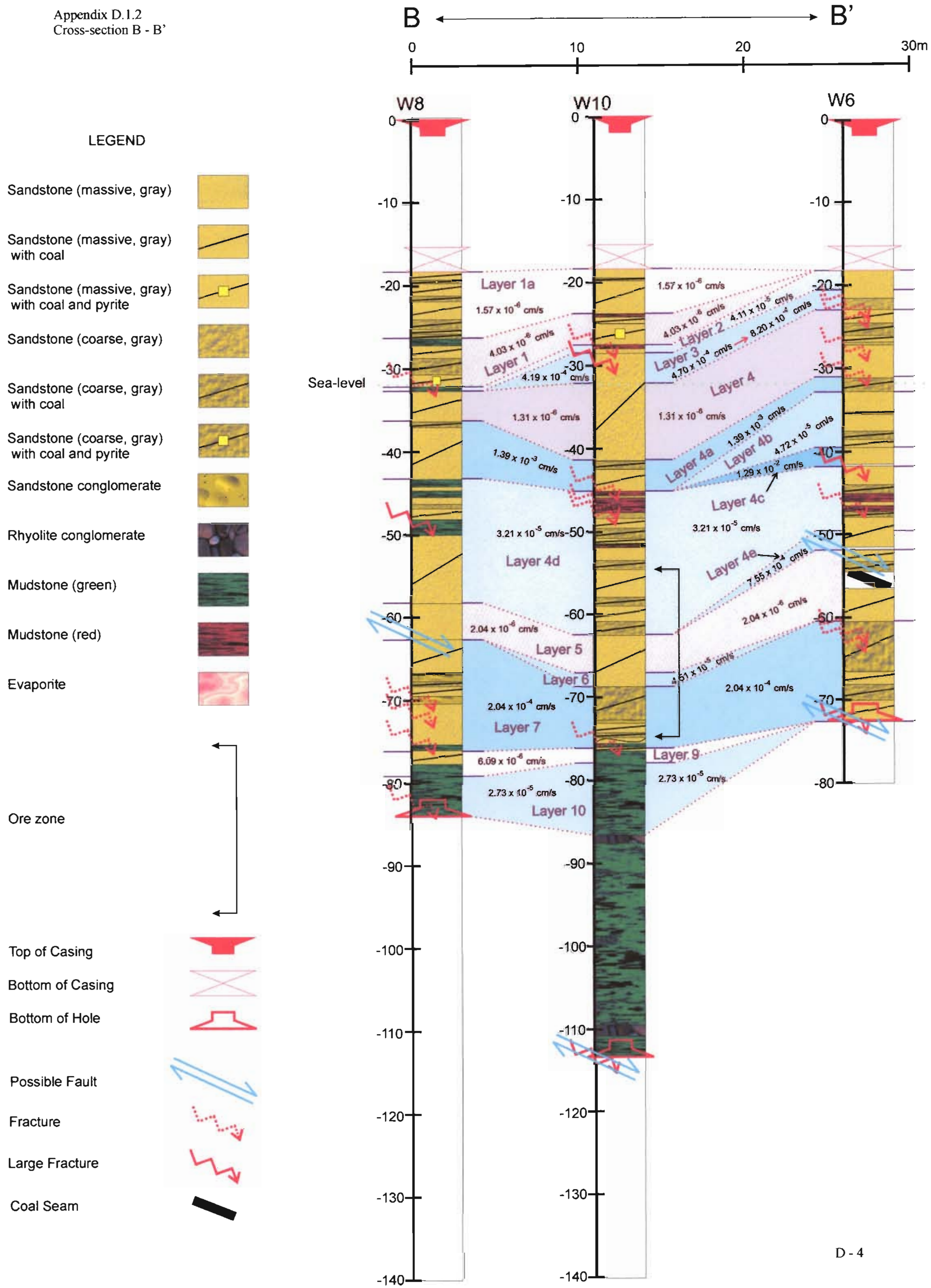
Appendix D.1
Cross-section Key Map



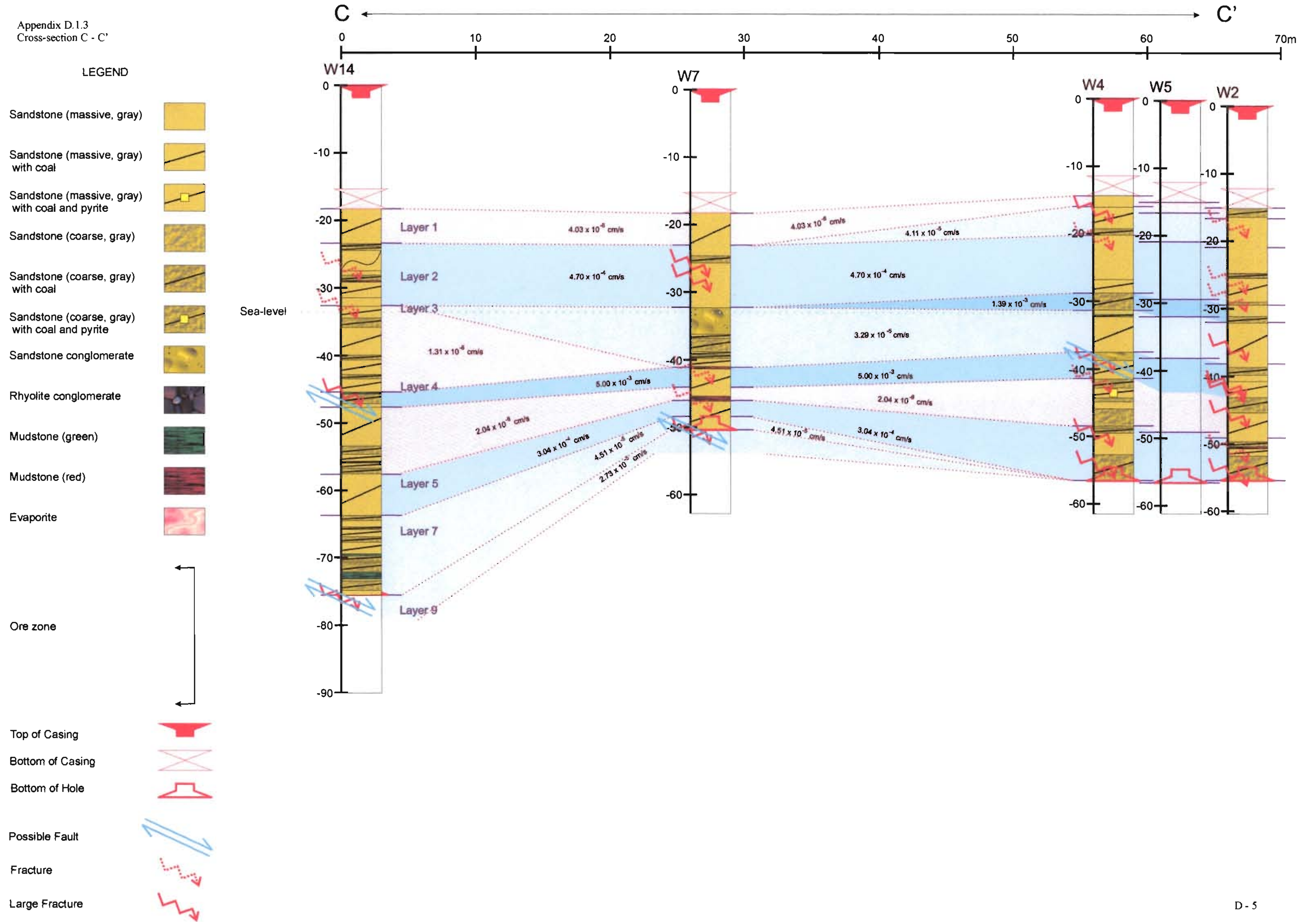
Appendix D.1.1
Cross-section A - A'



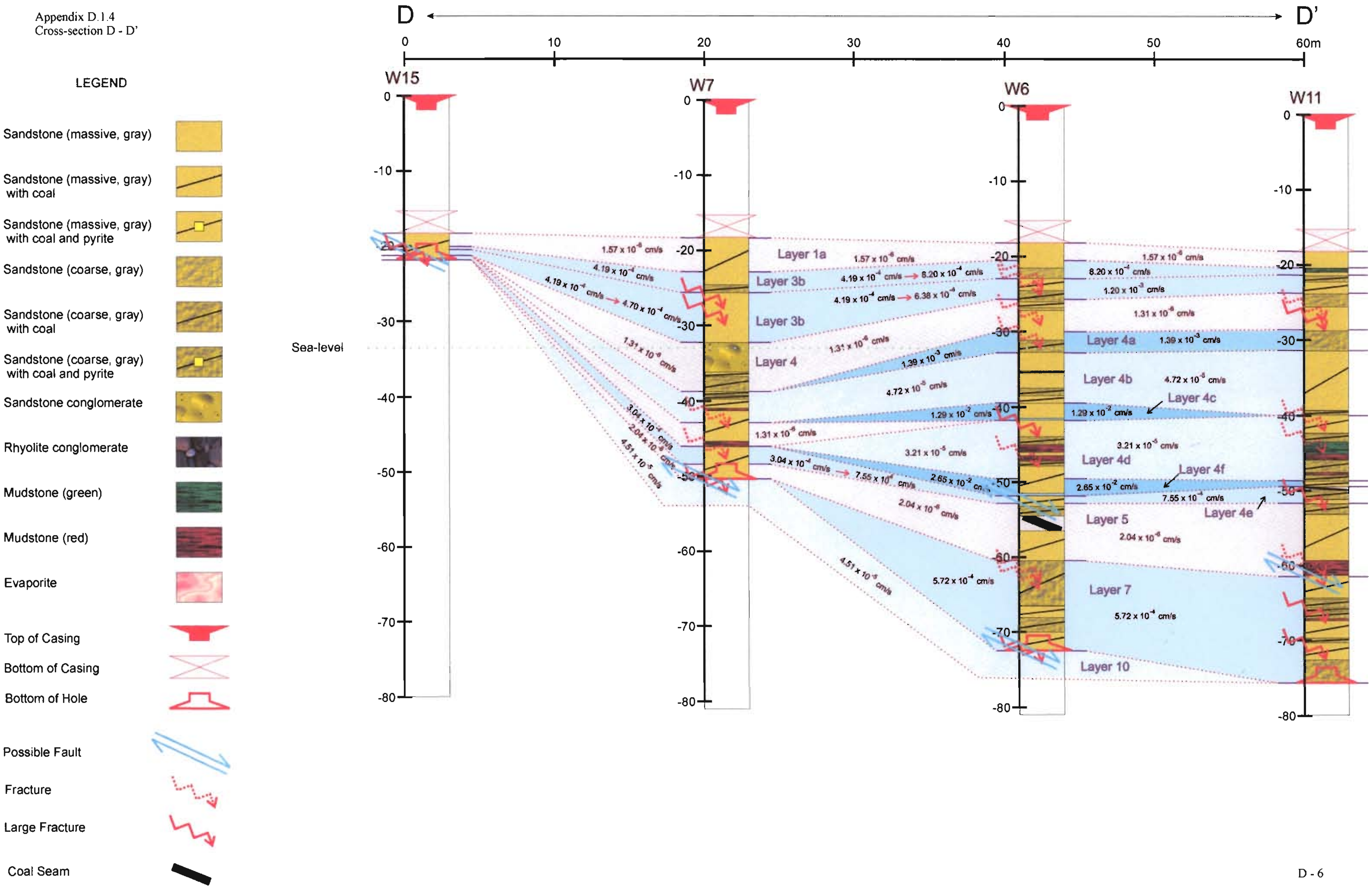
Appendix D.1.2
Cross-section B - B'



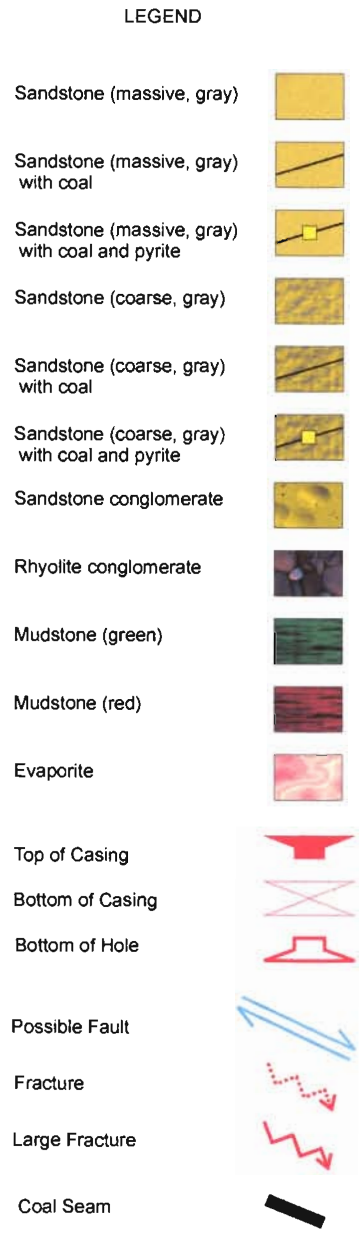
Appendix D.1.3
Cross-section C - C'

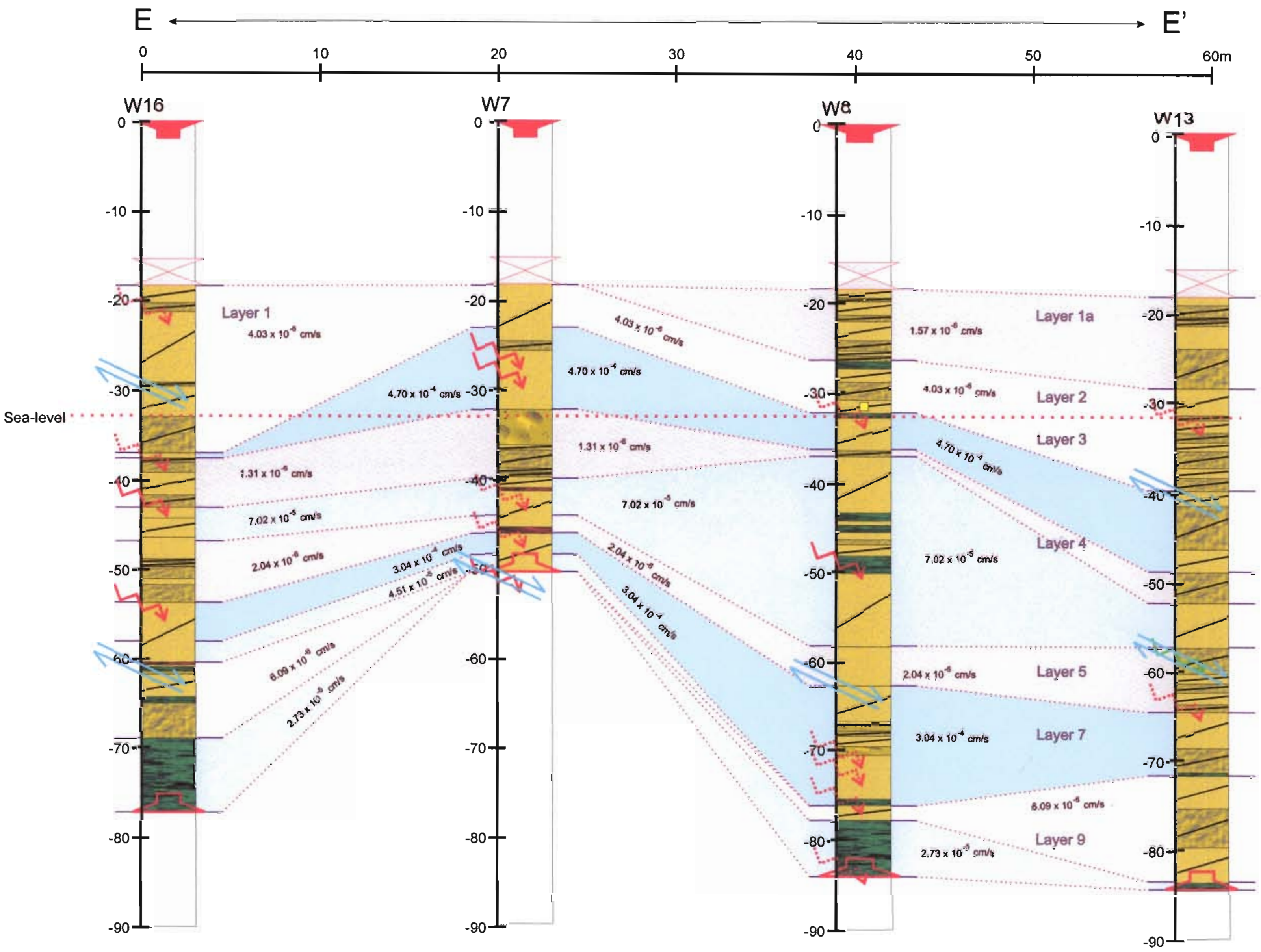


Appendix D.1.4
Cross-section D - D'

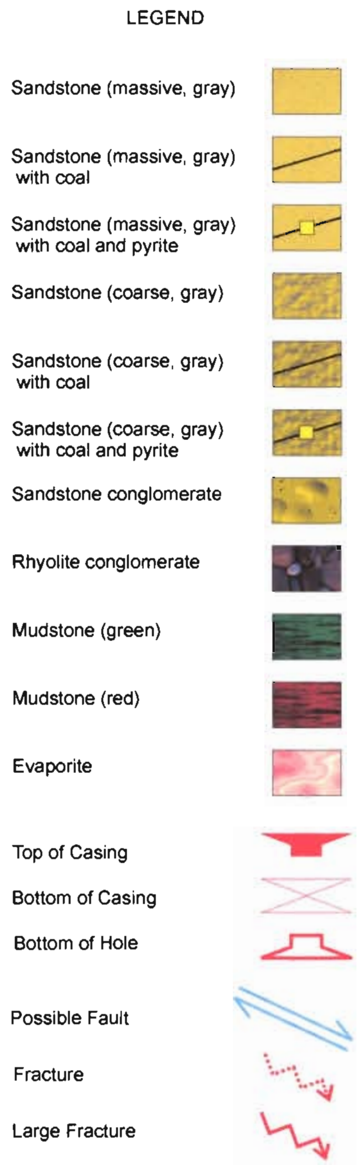


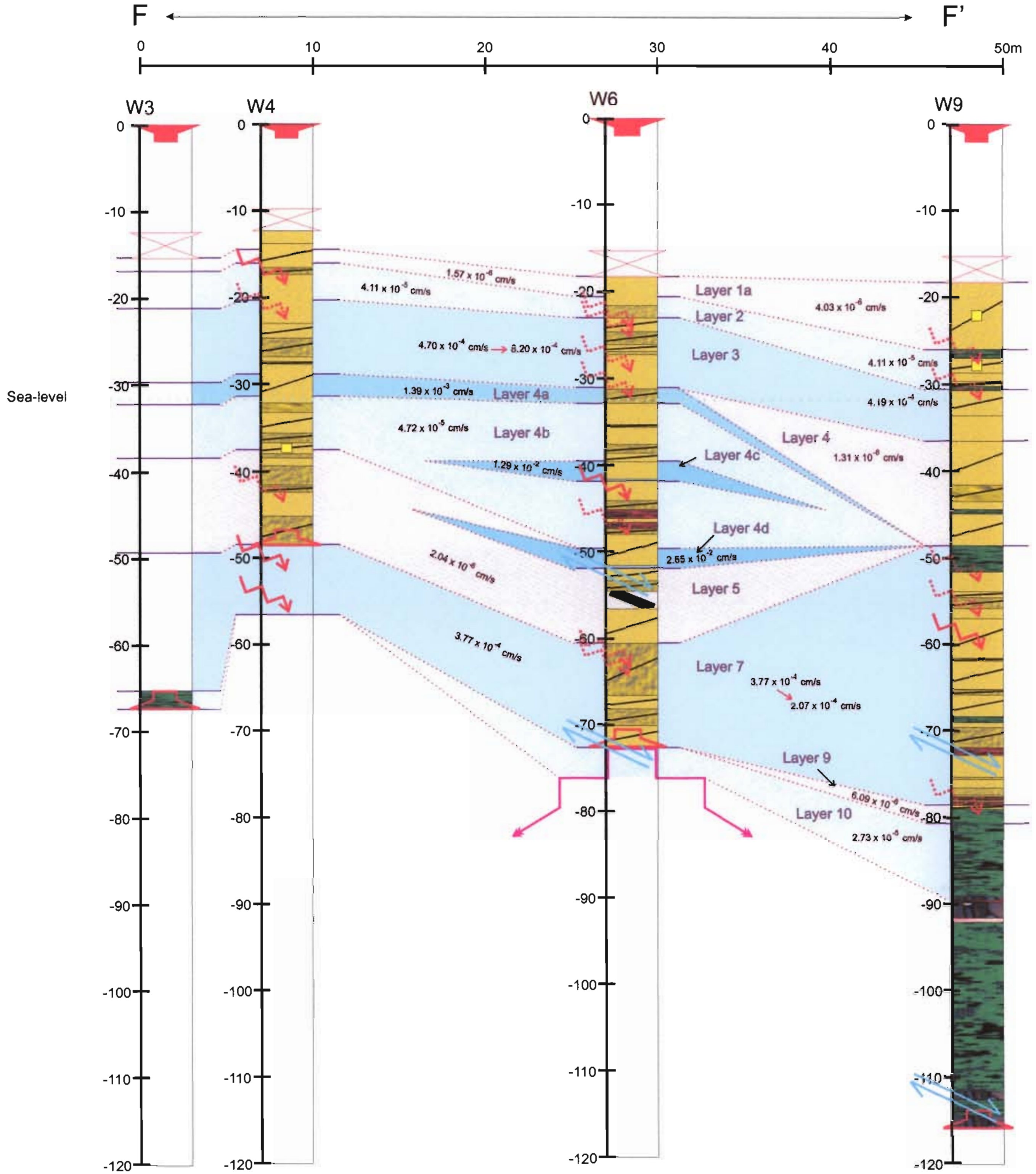
Appendix D.1.5
Cross-section E - E'



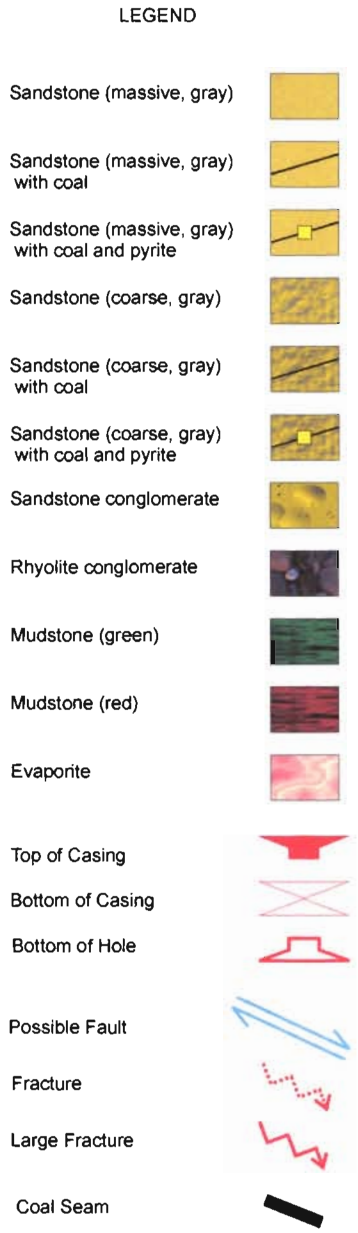


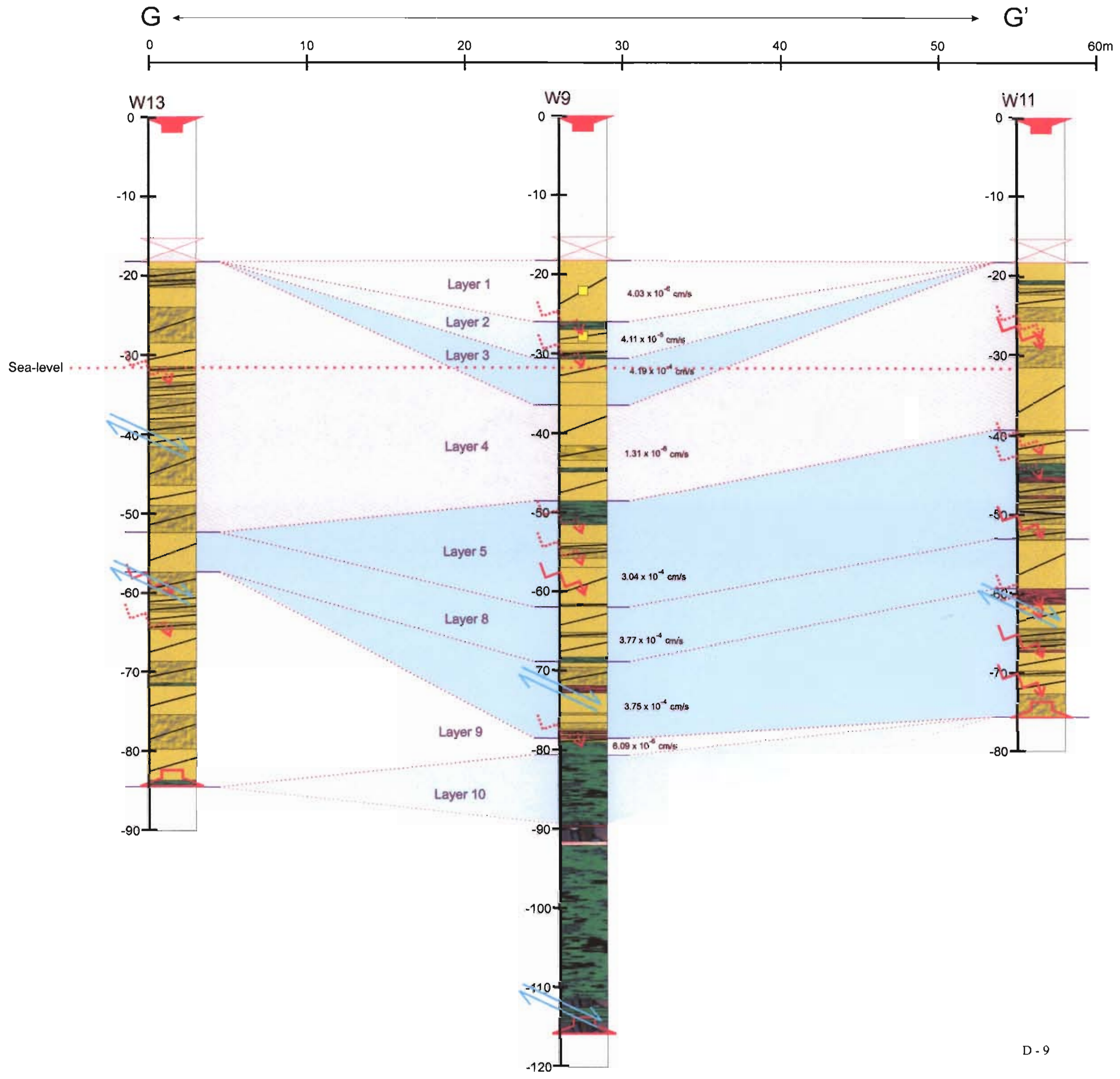
Appendix D.1.6
Cross-section F - F'



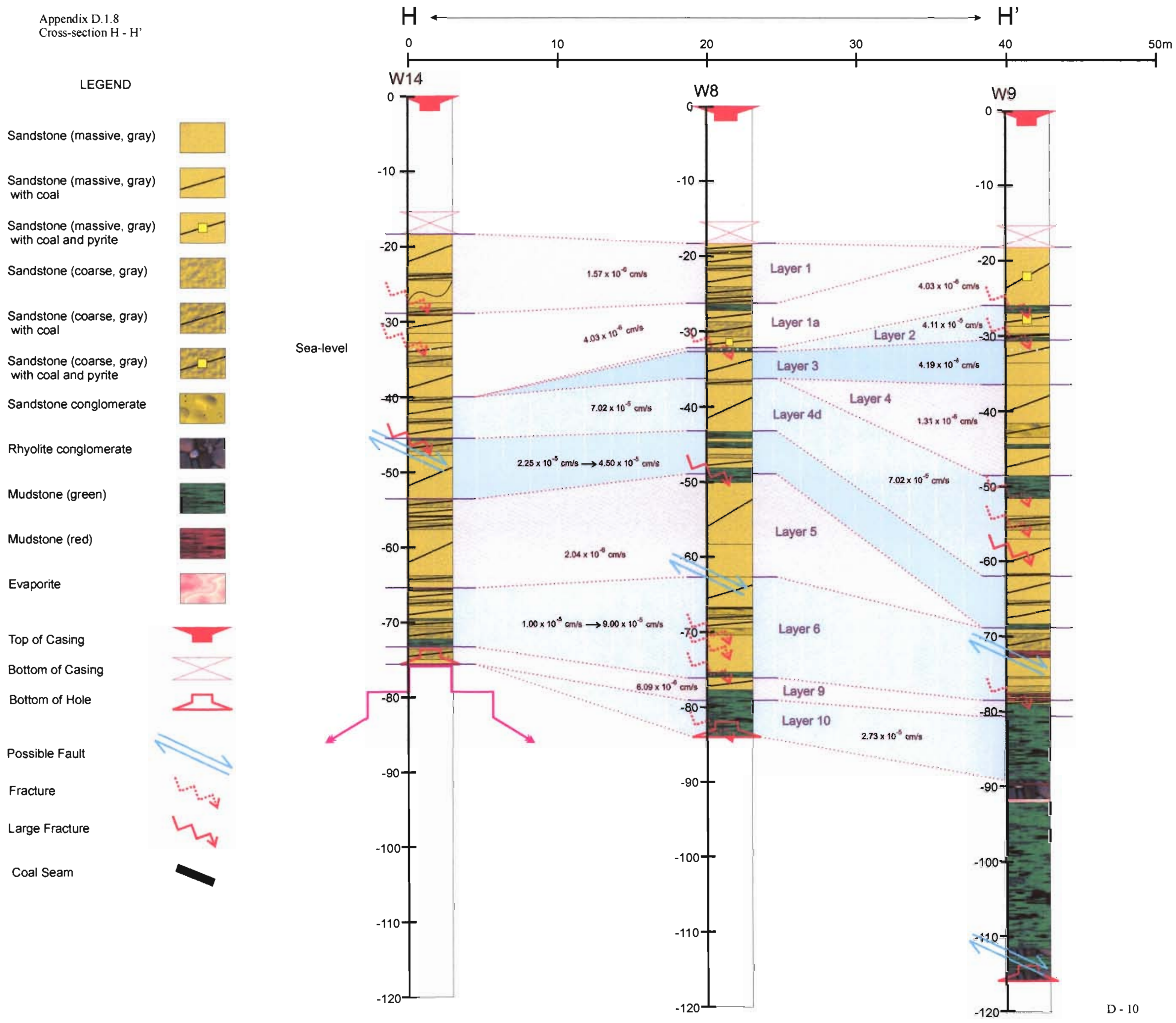


Appendix D.1.7
Cross-section G - G'





Appendix D.1.8
Cross-section H - H'



Appendix E
EXCEL 2000 Data Files for sandstone layering

Depths in metres \ Wells	*K (cm/s)*	W1	W2	W3	W4	W5	W6	W7	W8	W9	W10	W11	W12	W13	W14	W15	W16
Top of Casing		e+0.30	e+0.65	e+0.40	e+0.75	e+0.50	0	0	0	0	0	0	0	0	0	0	0
Bottom of Casing		14.33	15.09	15.24	14.17	14.72	18.29	18.29	18.29	18.29	18.29	18.29	23.16	18.29	18.29	18.29	18.29
Top of Aquifer		14.94	16.64	15.94	14.17	14.17	20.27	24.81	19.2	25.91	19.2	19.32	30.3	19.2	23.47	21.64	20.12
Bottom of Layer 1a	**1.57E-06**			16.75			po	no	26.5	no	23.75		no		21.75	20	no
Bottom of Layer 1	**4.03E-06**		18		16	17	po	25.8	32.25	26	27.25		30.5		40		37
Bottom of Layer 2	**4.11E-05**		21	21	20	20.75	20.5	po	32.75	30.75	28.25		37.25				no
Bottom of Layer 3a	**4.19E-04**															20.5	
Bottom of Layer 3b	**4.19E-04 - 4.7E-04**															21.25	
Bottom of Layer 3	**4.70E-04**		29.5	29.75	29	29.5	23	32.25	po	36.5	32		40.1				37.5
Bottom of Layer 4	**1.31E-06**						31	46.1	36.5	48.75	41.25	39.25	62.1	52.25			53.75
Bottom of Layer 4a	**1.39E-03**		32.5	32	31	31.5	32.5		43		44.5						
Bottom of Layer 4b	**4.72E-05**		39	38.25	37.5	38.5	39.5		no		po						
Bottom of Layer 4c	**1.29E-02**		44.5		41.5	43.5	41.75		no		po						
Bottom of Layer 4d	**3.21E-05**						49.5		58.5		62.25				45		
Bottom of Layer 4e	**7.55E-04**						52		po		no				53.25	21.5	
Bottom of Layer 5	**3.04E-04**		49	49.5	48.5	49	60.5	48.5	62.5	62	67	53	68.5		65		58
Bottom of Layer 6	**4.51E-05**		55.5		56.5	56	po	50.25	po	po	68.75		no		73.25		60.25
Bottom of Layer 7	**3.77E-04**			65.25			po	po	76	68.5	76		76.75				no
Bottom of Layer 8	**3.75E-04**							no		78.5	po	76	po	57.25			no
Bottom of Layer 9	**6.09E-06**						po	po	79	80.75	78		93	84.5	75.5		no
Bottom of Layer 10	**2.73E-05**	53.64	55.78	67.5			po	po	83.5	89.5	86.5		po				77.42

No Layer (no)
Pinch out (po)

Appendix F.1
Drawdown and hydraulic head calculations for Pumping Test data

Drawdown (m)

Date	**Start (s)**	**Finish (s)**	**PW**	**PR (gpm)**	**W1**	**W2**	**W3**	**W4**	**W5**	**W6**	**W7**	**W8**	**W9**	**W10**	**W11**	**W12**	**W13**	**W14**	**W15**	**W16**
9-Jun-04	0	14400	13	8	0.26	0.22	0.22	0.29	0.23	0.22	0.23	0.25	0.26	0.18	0.26	0.3	***14.89***	0.32	0.25	0.23
10-Jun-04	76380	79260	8	5	0.12	0.1	0.1	0.13	0.11	0.1	0.12	***32.46***	0.2	0.2	0.15	0.13	0.03	0.35	0.1	0.01
11-Jun-04	79260	141780			-0.02	-0.01	-0.01	-0.01	-0.01	0	-0.01	0	0.06	-0.03	-0.01	-0.01	0.01	0	-0.01	-0.01
12-Jun-04	227160	241500	7	40	4.6	4.7	4.6	4.6	4.6	4.5	***6.43***	4	3.9	4	4.4	3.6	1.7	3.8	4.9	4.7
14-Jun-04	404580	418980	3	40	5.1	5.1	***5.1***	5.1	5.1	4.6	4.8	3.8	4	4.2	4.6	3.7	1.6	3.6	4.5	4.7
15-Jun-04	490980	505380	12	40	3.3	3.3	3	3.3	3.2	3.6	3.2	4.1	4.8	3.4	3.7	***9.9***	1.5	3.2	3	3.2

Head (masl)

				W1	W2	W3	W4	W5	W6	W7	W8	W9	W10	W11	W12	W13	W14	W15	W16
		Elevation (masl)		***32.09***	***32.09***	***32.35***	***32.67***	***32.22***	***33.25***	***33.81***	***33.15***	***32.57***	***33.07***	***32.37***	***31.32***	***32.41***	***34.44***	***34.38***	***33.66***
Date		**Model Time (s)**		**W1**	**W2**	**W3**	**W4**	**W5**	**W6**	**W7**	**W8**	**W9**	**W10**	**W11**	**W12**	**W13**	**W14**	**W15**	**W16**
9-Jun-04	**8:30**	-18420	0	22.61	22.61	22.62	22.61	22.61	22.58	22.54	22.38	22.51	22.98	22.59	22.44	21.61	22.18	22.46	22.53
	13:37	0	8	22.61	22.61	22.62	22.61	22.61	22.58	22.54	22.38	22.51	22.98	22.59	22.44	21.61	22.18	22.46	22.53
	17:37	14400	0	22.35	22.39	22.40	22.32	22.38	22.36	22.31	22.13	22.25	22.80	22.33	22.14	***6.72***	21.86	22.21	22.30
10-Jun-04	9:30	73980	0	22.58	22.58	22.59	22.58	22.58	22.55	22.51	22.35	22.41	22.96	22.56	22.42	21.09	22.16	22.42	22.49
	10:50	76380	5	22.58	22.58	22.59	22.58	22.58	22.55	22.51	22.35	22.41	22.96	22.56	22.42	21.09	22.16	22.42	22.49
	11:38	79260	0	22.46	22.48	22.49	22.45	22.47	22.45	22.39	***-10.11***	22.21	22.76	22.41	22.29	21.06	21.81	22.32	22.48
11-Jun-04	9:00	121800	0	22.57	22.57	22.57	22.57	22.57	22.54	22.50	22.35	22.47	22.93	22.56	22.41	21.10	22.15	22.41	22.49
	14:33	141780	20 - 40	22.57	22.57	22.57	22.57	22.57	22.54	22.50	22.35	22.47	22.93	22.56	22.41	21.10	22.15	22.41	22.49
12-Jun-04	7:30	222780	0	22.49	22.50	22.50	22.51	22.50	22.47	22.44	22.95	22.41	22.86	22.49	22.34	21.11	22.09	22.36	22.42
	8:43	227160	40	22.49	22.50	22.50	22.51	22.50	22.47	22.44	22.95	22.41	22.86	22.49	22.34	21.11	22.09	22.36	22.42
	12:42	241500	0	17.89	17.80	17.90	17.91	17.90	17.97	***16.01***	18.95	18.51	18.86	18.09	18.74	19.41	18.29	17.46	17.72
14-Jun-04	7:40	396180	0	22.47	22.47	22.47	22.46	22.46	22.44	22.39	22.90	22.38	22.82	22.45	22.31	21.10	22.06	22.30	22.37
	10:00	404580	40	22.47	22.47	22.47	22.46	22.46	22.44	22.39	22.90	22.38	22.82	22.45	22.31	21.10	22.06	22.30	22.37
	14:00	418980	0	17.37	17.37	***17.37***	17.36	17.36	17.84	17.59	19.10	18.38	18.62	17.85	18.61	19.50	18.46	17.80	17.67
15-Jun-04	8:00	483780	0	22.35	22.34	22.32	22.31	22.30	22.31	22.27	22.77	22.25	22.69	22.31	22.19	21.05	21.93	22.17	22.28
	10:00	490980	40	22.35	22.34	22.32	22.31	22.30	22.31	22.27	22.77	22.25	22.69	22.31	22.19	21.05	21.93	22.17	22.28
	14:00	505380	0	19.05	19.04	19.32	19.01	19.10	18.71	19.07	18.67	17.45	19.29	18.61	***12.29***	19.55	18.73	19.17	19.08
16-Jun-04	14:30	593580	0	22.41	22.41	22.41	22.41	22.40	22.38	22.36	22.86	22.32	22.76	22.39	22.29	21.12	22.01	22.26	22.33
18-Jun-04	6:45	659280	0	22.45	22.45	22.45	22.45	22.45	22.42	22.40	22.90	22.36	22.81	22.44	22.28	21.14	22.05	22.30	22.37
	16:45	688080	PT	22.52	22.51	22.50	22.52	22.52	22.50	22.47	23.01	22.43	Packer	22.51	22.35	21.16	22.11	22.37	22.44

Appendix F.2
Hydraulic input data Head

x (m)	y (m)	z (masl)
35.818	13.953	22.61
30.795	9.373	22.61
25.920	14.249	22.62
30.647	18.976	22.61
30.499	14.101	22.74
44.829	34.784	22.58
29.317	48.818	22.54
43.500	64.182	22.38
59.159	48.818	22.51
43.795	48.670	22.98
59.159	20.158	22.59
85.456	47.045	22.44
58.569	74.080	21.6
28.431	77.478	22.18
16.612	62.410	22.46
15.283	38.477	22.53
0.214	15.726	23
1.544	11.885	23
6.419	7.157	23
12.771	4.351	23
19.272	2.430	23
26.363	1.248	23
36.852	0.657	23
51.034	99.934	21.5
52.955	94.911	21.5
55.466	88.558	21.5
60.637	82.649	21.5
65.955	80.138	21.5
73.046	79.694	21.5
79.547	82.206	21.5
86.786	85.456	21.5
93.729	89.002	21.5
99.491	93.581	21.5
59.455	99.786	21.45
60.785	95.797	21.45
64.773	91.218	21.45
70.830	87.967	21.45
79.547	89.445	21.45
86.933	91.218	21.45
93.434	95.206	21.45
99.638	98.309	21.45
69.058	99.786	21.4
72.899	96.240	21.4
79.251	95.059	21.4
87.081	95.797	21.4
95.945	99.786	21.4

Appendix F.3
Calculation of Specific Storage

Wells	**W1**	**W2**	**W3**	**W4**	**W5**	**W6**	**W7**	**W8**	**W9**	**W10**	**W11**	**W12**	**W13**	**W14**	**W15**	**W16**
Thickness of sandstone (b; [m],	62.75	60.00	60.00	62.75	61.50	70.50	68.00	72.00	76.00	71.75	71.25	80.90	78.00	67.00	64.00	62.25
Storativity (S) (Michel, 2004)																
9-Jun-04	3.34E-04	2.85E-04	2.85E-04	3.93E-04	3.12E-04	7.49E-04	9.87E-04	3.34E-03	2.37E-03	2.11E-03	4.95E-04	8.21E-04	1.80E-03	6.83E-04	5.95E-04	4.31E-04
12-Jun-04	8.29E-05	7.73E-05	9.09E-05	1.23E-04	8.48E-05	2.81E-04	2.08E-02	5.60E-04	2.42E-04	1.77E-03	8.48E-05	6.89E-05	4.06E-04	2.81E-04	1.90E-04	3.01E-04
14-Jun-04	8.19E-04	1.42E-03	6.88E-02	1.42E-03	2.84E-03	1.60E-04	7.47E-05	8.38E-05	8.38E-05	1.79E-04	1.03E-04	3.74E-05	1.33E-04	6.68E-05	4.30E-05	1.60E-04
15-Jun-04	6.62E-05	6.62E-05	4.82E-05	6.21E-05	6.07E-05	8.78E-05	8.38E-05	6.66E-05	8.00E-05	1.36E-04	8.98E-05	3.49E-04	4.50E-04	6.51E-05	5.29E-05	5.17E-05
1-Nov-04	4.70E-05	4.70E-05	4.70E-05	4.70E-05	1.45E-03	6.05E-07	1.35E-06	1.03E-06	1.39E-06	2.35E-06	4.80E-07	7.44E-07	2.10E-06	8.35E-07	7.44E-07	9.16E-07
Specific Storage (S/b; [1/m])																
9-Jun-04	5.32E-06	4.75E-06	4.75E-06	6.26E-06	5.07E-06	1.06E-05	1.45E-05	4.64E-05	3.12E-05	2.94E-05	6.95E-06	1.01E-05	2.31E-05	1.02E-05	9.30E-06	6.92E-06
12-Jun-04	1.32E-06	1.29E-06	1.52E-06	1.96E-06	1.38E-06	3.99E-06	3.06E-04	7.78E-06	3.16E-06	2.47E-05	1.19E-06	8.52E-07	5.21E-06	4.19E-06	2.97E-06	4.84E-06
14-Jun-04	1.31E-05	2.37E-05	1.15E-03	2.26E-05	4.62E-05	2.27E-06	1.10E-06	1.16E-06	1.10E-06	2.49E-06	1.45E-06	4.62E-07	1.71E-06	9.94E-07	6.72E-07	2.57E-06
15-Jun-04	1.05E-06	1.10E-06	8.03E-07	9.90E-07	9.87E-07	1.25E-06	1.23E-06	9.25E-07	1.05E-06	1.90E-06	1.26E-06	4.31E-06	5.77E-06	9.72E-07	8.27E-07	8.31E-07
1-Nov-04	7.49E-07	7.83E-07	7.83E-07	7.49E-07	2.36E-05	8.58E-09	1.99E-08	1.43E-08	1.83E-08	3.28E-08	6.74E-09	9.20E-09	2.69E-06	1.25E-08	1.16E-08	1.47E-08
Average 9, 12, 15 June	2.57E-06	2.38E-06	2.36E-06	3.07E-06	2.48E-06	5.29E-06	5.25E-06	1.84E-05	1.18E-05	1.87E-05	3.13E-06	5.10E-06	1.14E-05	5.12E-06	4.36E-06	4.20E-06

Note: Data from 14 June and 1 Nov discarded due to high or low readings

Appendix F.4
Specific Storage input Data

Wells	**x (m)**	**y (m)**	**Ss (1/m)**	**Sy**	**ep**	**tp**
1	35.9077	13.78428	*2.57E-06*	0.1	0.075	0.225
2	31.05873	9.692983	2.38E-06	0.1	0.075	0.225
3	25.9067	13.93582	2.36E-06	0.1	0.075	0.225
4	30.45262	18.93632	3.07E-06	0.1	0.075	0.225
5	30.45262	14.08735	2.48E-06	0.1	0.075	0.225
6	44.39338	35.15005	5.29E-06	0.1	0.075	0.225
7	29.24037	48.63623	5.25E-06	0.1	0.075	0.225
8	43.4842	64.5469	1.84E-05	0.1	0.075	0.225
9	59.39487	48.4847	1.18E-05	0.1	0.075	0.225
10	43.63573	48.93928	1.87E-05	0.1	0.075	0.225
11	59.5464	19.69397	3.13E-06	0.1	0.075	0.225
12	84.85193	46.66633	5.10E-06	0.1	0.075	0.225
13	58.33417	74.24483	1.14E-05	0.1	0.075	0.225
14	28.17965	77.42697	5.12E-06	0.1	0.075	0.225
15	16.96642	63.0316	4.36E-06	0.1	0.075	0.225
16	15.2996	38.33218	4.20E-06	0.1	0.075	0.225

Ss = Specific Storage
Sy = Specific Yeild
ep = estimated porosity
tp = total porosity

Calculation of well distances

Well distance was confirmed using a method after Jacob (Fetter, 2001).

$$S = \frac{2.25Tt_0}{r^2}$$

Where S is the storativity of the aquifer, T is the transmissivity (m /s) of the aquifer, t_0 is the time (s) and r^2 is the radial distance (m) from the pumping well. Setting up this equation for two different scenarios within the same zones of S and T gives two simultaneous equations that could be solved for r (a distance between wells), given the timing and distance of breakthrough curves (bt) and the time of drawdown (s) in a monitoring well. So the resulting equation would be:

$$\frac{2.25Tt_s}{r^2} = \frac{2.25Tt_{bt}}{r_{bt}}$$

$$\frac{t_s}{r^2} = \frac{t_{bt}}{r_{bt}^2}$$

$$or \quad r = \sqrt{\frac{t_s r_{bt}^2}{t_{bt}}}$$

Appendix C

Appendix C

To

GEOLOGIC AND HYDROGEOLOGIC INVESTIGATIONS
AT THE
SILVER MINE PROPERTY
CAPE BRETON COUNTY, NOVA SCOTIA

For the year ending 30 April 2006

MAPS INDICATING DDH CORE LOCATIONS, PROPERTY BOUNDARIES AND LOCATIONS OF CROSS-SECTIONS WHICH SHOW THE ORE ZONE IN RELATION TO HOST SANDSTONES AT THE YTI PROPERTY

SYMBOLS

Mine	⚒
Road	┈
Access Road to Processing Plant	═
Leachant line to Well-Field and Leachate line from Well-Field	═
Processing Plant	🟩
River, Lake	～
Bedding (inclined)	⁄
Geologic boundary (assumed)	— — —
Unconformity (approximate)	⊥⊥⊥
Fault (approximate)	— — —
Diamond Drill Hole	● ●
Former Yava Mine workings	┈
YTI claim boundary	━
Mineralized Zone	◌

LEGEND

PENNSYLVANIAN	6	**BIG BARREN FORMATION:** red conglomerate/sandstone and gray-green mudstone/sandstone.
	5	**SILVER MINE FORMATION:** gray-green/gray sandstone with coal; gray-green/gray mudstone.
MISSISSIPPIAN	4	**MacKEIGAN LAKE FORMATION:** gray/red shale, siltstone, gypsum/anhydrite.
	3	**UIST FORMATION:** Red siltstone, minor conglomerate intercalated with limestone/dolostone; minor gypsum/anhydrite; dolostone (7a).
Pre-SILURIAN	2	**SALMON RIVER RHYOLITE PORPHYRY:** Granitoid rocks; rhyolite intruding coarse grained sandstone/conglomerate (2a).
	1	**HADRYNIAN to ORDOVICIAN ROCKS:** Volcanic and sedimantary rocks, variably deformed and metamorphosed.

2006 - 4

YAVA TECHNOLOGIES INC. (YTI)

GEOLOGY OF THE YTI CLAIMS INDICATING LOCATION OF DIAMOND DRILL HOLES, DRILLED FROM 1962 TO 2003

SILVER MINE PROPERTY, CAPE BRETON COUNTY

NOVA SCOTIA

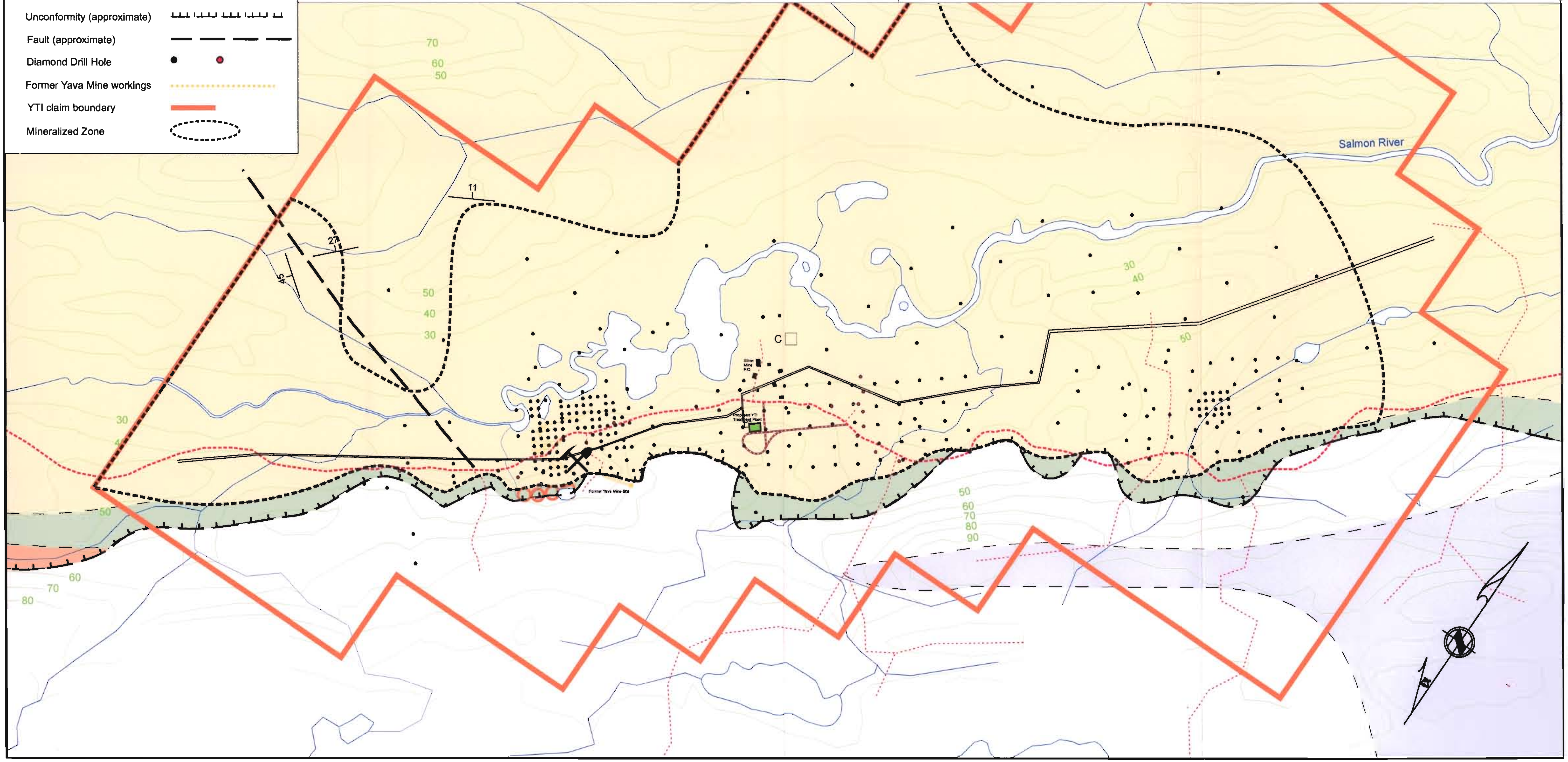
R. E. A. Boudreau

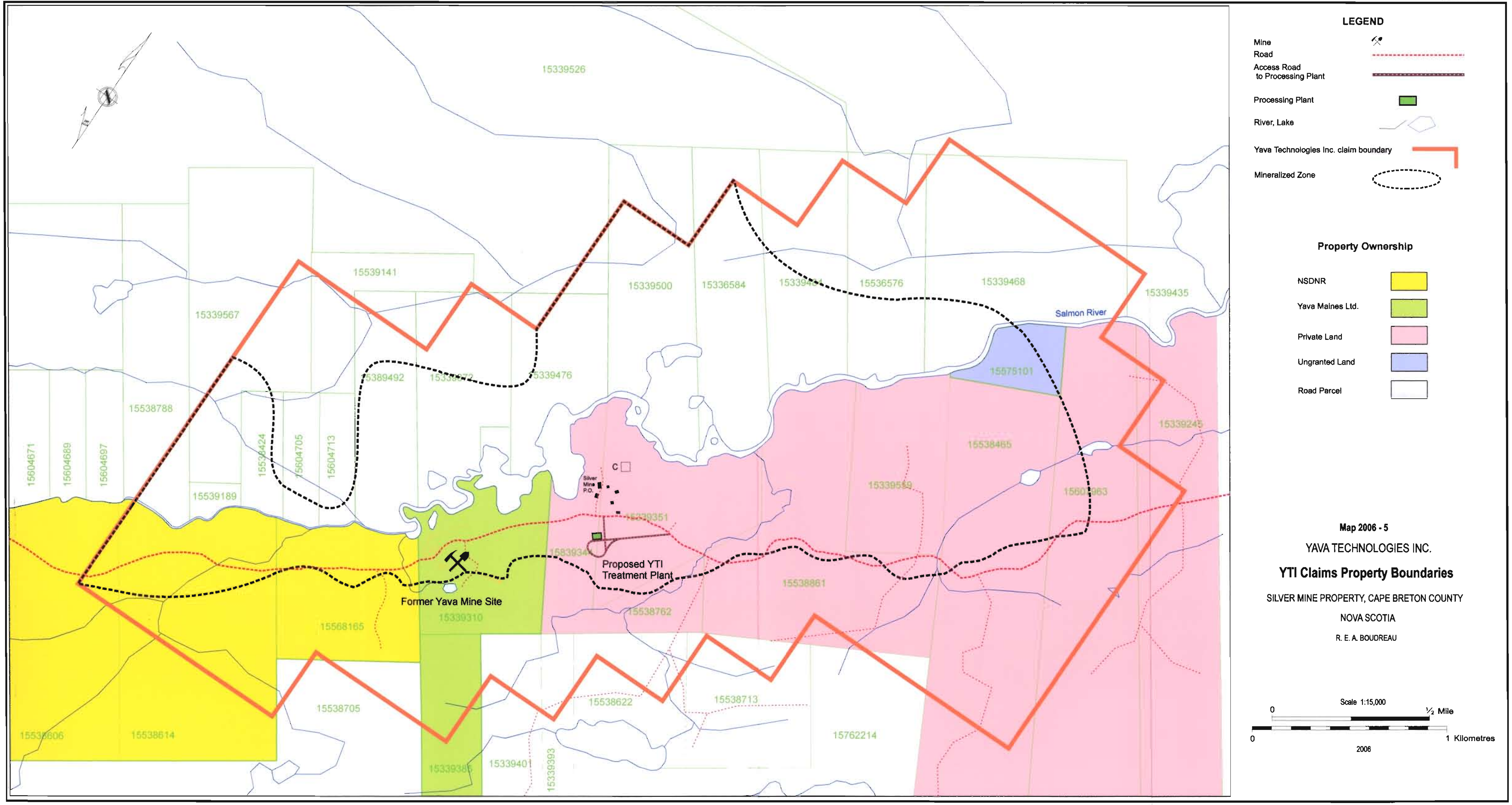
Scale 1:4550

0 1 Mile

0 1 Kilometre

2006




LEGEND
Mine
Road
Access Road to Processing Plant
Processing Plant
River, Lake
Yava Technologies Inc. claim boundary
Mineralized Zone
Property Ownership
NSDNR
Yava Maines Ltd.
Private Land
Ungranted Land
Road Parcel
Map 2006 - 5
YAVA TECHNOLOGIES INC.
YTI Claims Property Boundaries
SILVER MINE PROPERTY, CAPE BRETON COUNTY
NOVA SCOTIA
R. E. A. BOUDREAU
Scale 1:15,000
0
½ Mile
0
1 Kilometres
2006
Salmon River
Former Yava Mine Site
Proposed YTI Treatment Plant
Silver Mine P.O.
C
15339526
15539141
15339567
15339500
15336584
15536576
15339468
15339435
15575101
15338788
15339476
15538465
15339245
15604671
15604689
15604697
15604705
15604713
15539189
15339351
15601963
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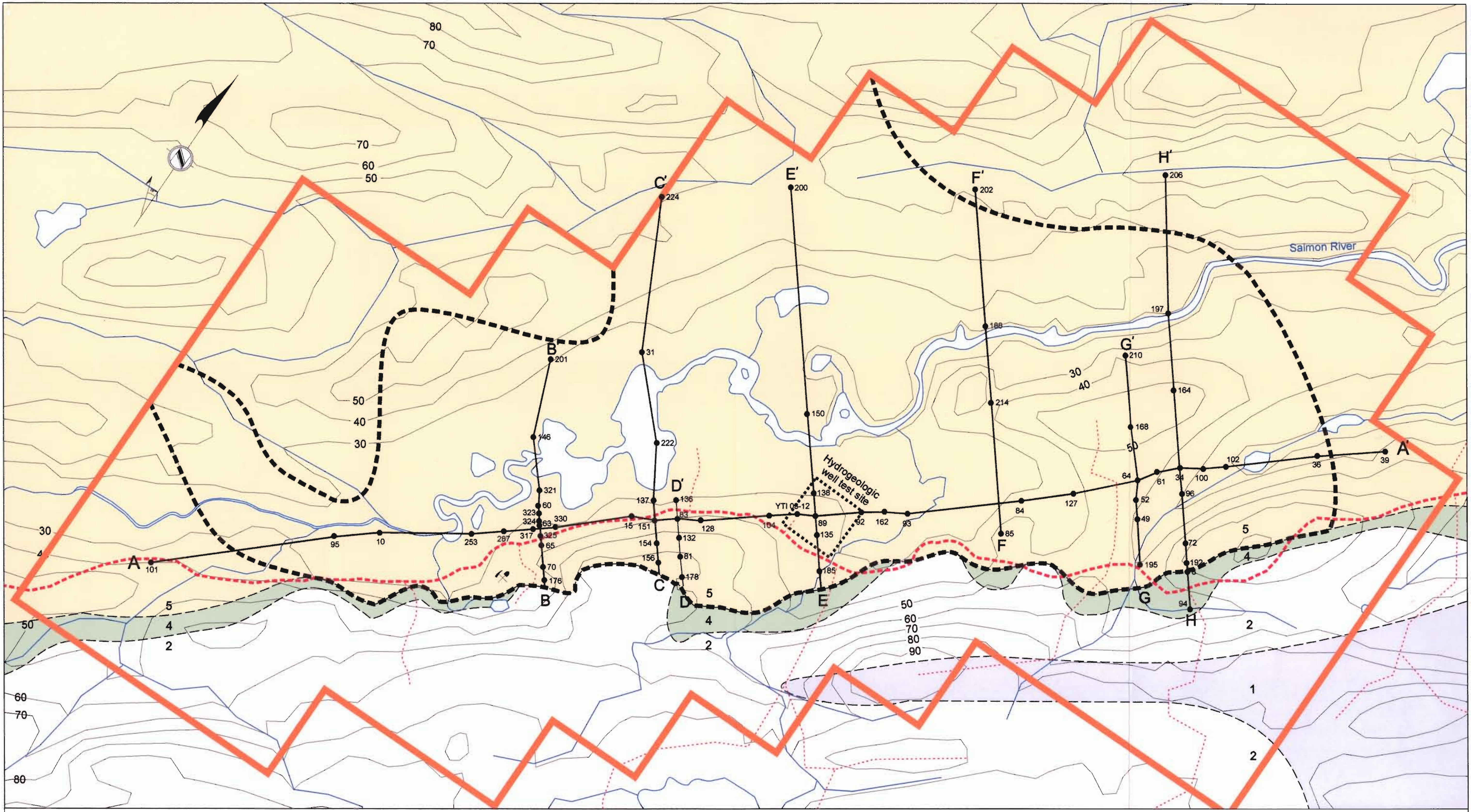
15538614
15339401
15339393



PENNSYLVANIAN	5	**SILVER MINE FORMATION:** gray-green/gray sandstone with coal; gray-green/gray mudstone.
MISSISSIPPIAN	4	**MacKEIGAN LAKE FORMATION:** gray/red shale, siltstone, gypsum/anhydrite.
Pre-SILURIAN	2	**SALMON RIVER RHYOLITE PORPHYRY:** Granitoid rocks; rhyolite intruding coarse grained sandstone/conglomerate (2a).
	1	**HADRYNIAN to ORDOVICIAN ROCKS:** Volcanic and sedimantary rocks, variably deformed and metamorphosed.

Explanation

Former mine	⚒
Gravel road, trail	▬ ▬ ▬ · · · · ·
Property boundary	▬▬▬
Mineralized Zone	▬ ▬ ▬ ▬
Geologic Boundary	— — — — —
Contour	—50—
Diamond drill hole (PD series, Yava Technologies Inc.)	● 64 ● YTI 03-12
Geologic section	A ●——● A′

Map 2006 - 6

YAVA TECHNOLOGIES INC.

Geology of the property

SILVER MINE PROPERTY, CAPE BRETON COUNTY

NOVA SCOTIA

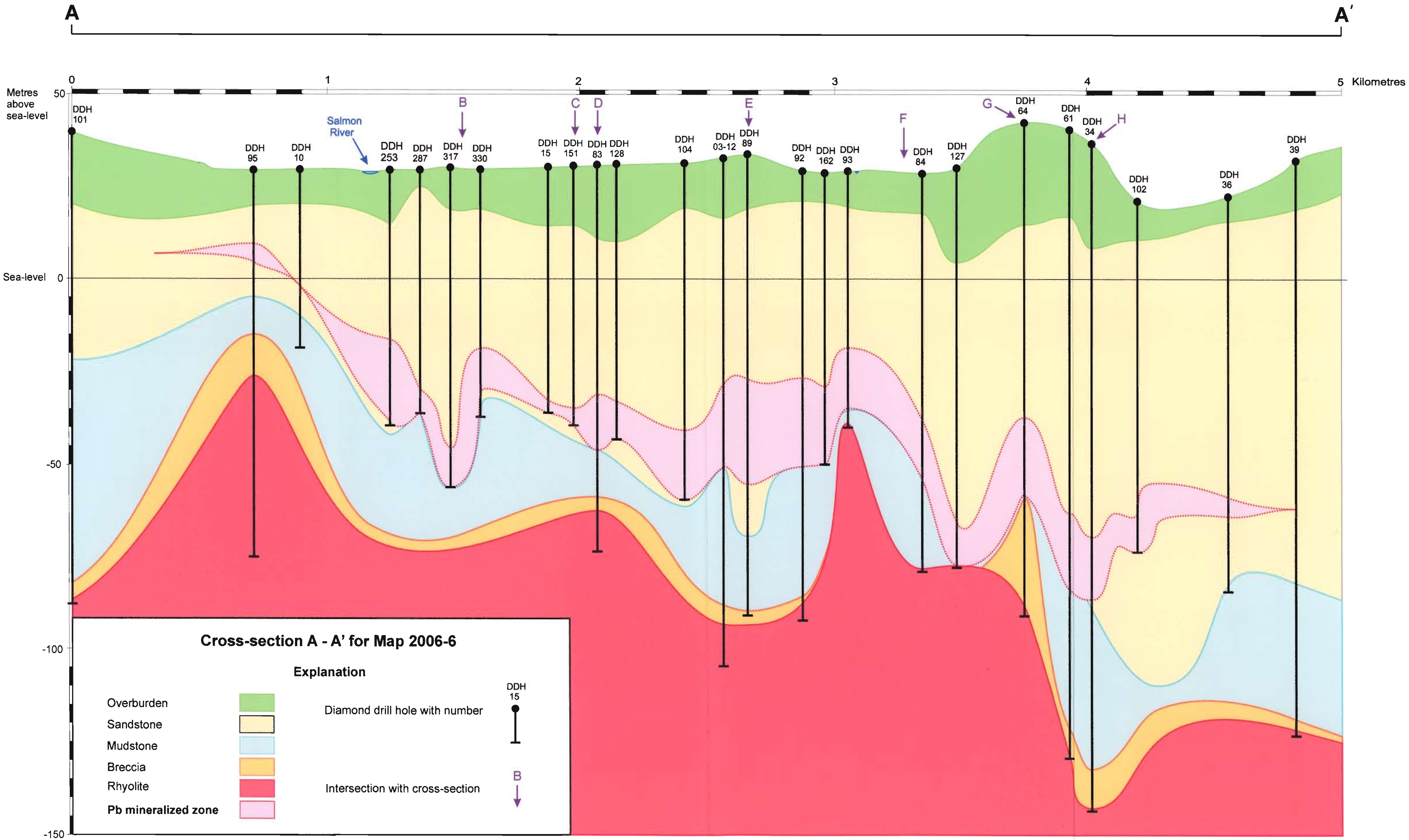
R. E. A. BOUDREAU

Scale 1:14,500

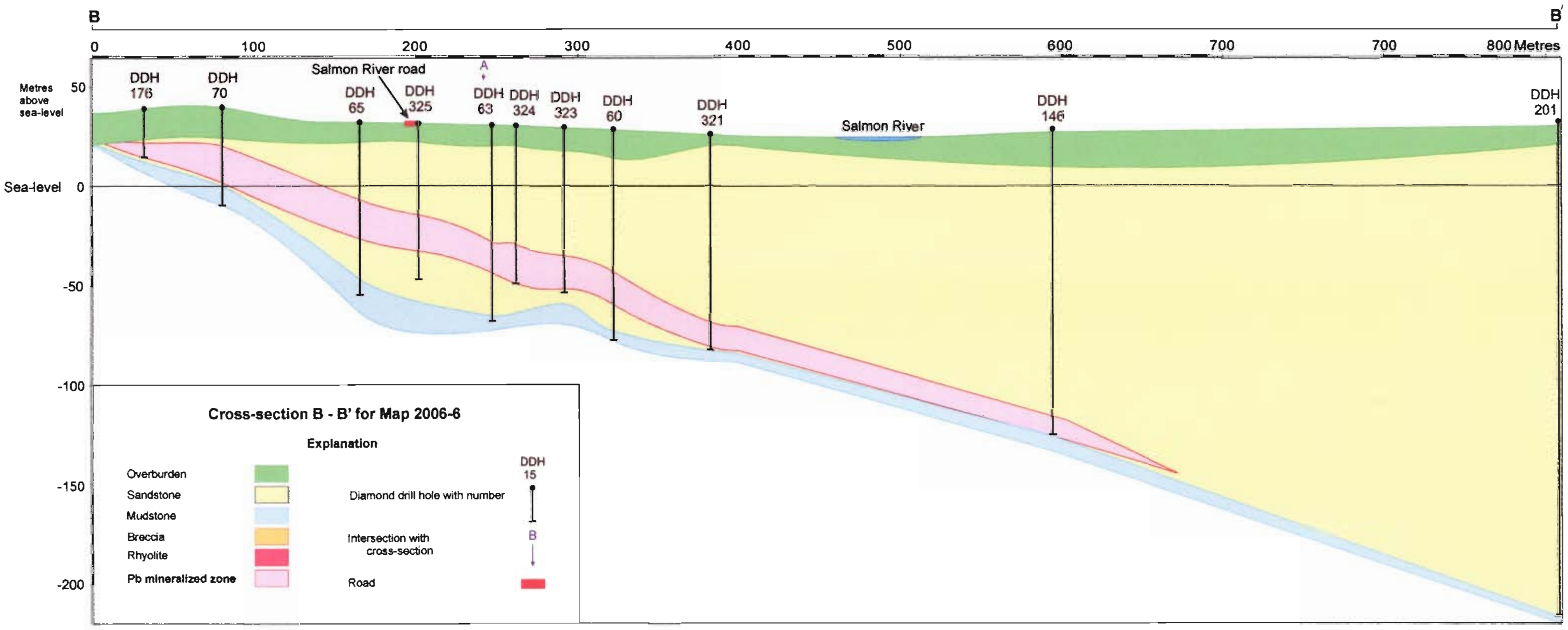
0 1 2 Kilometres

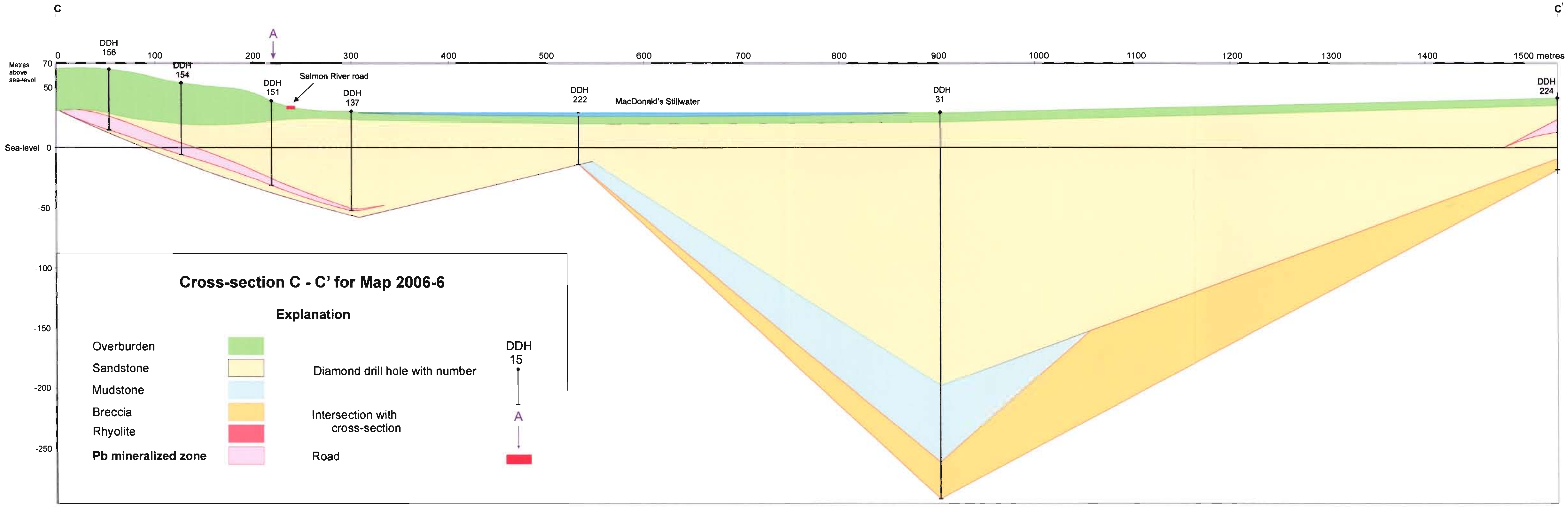
Contour interval in metres above sea-level

2006

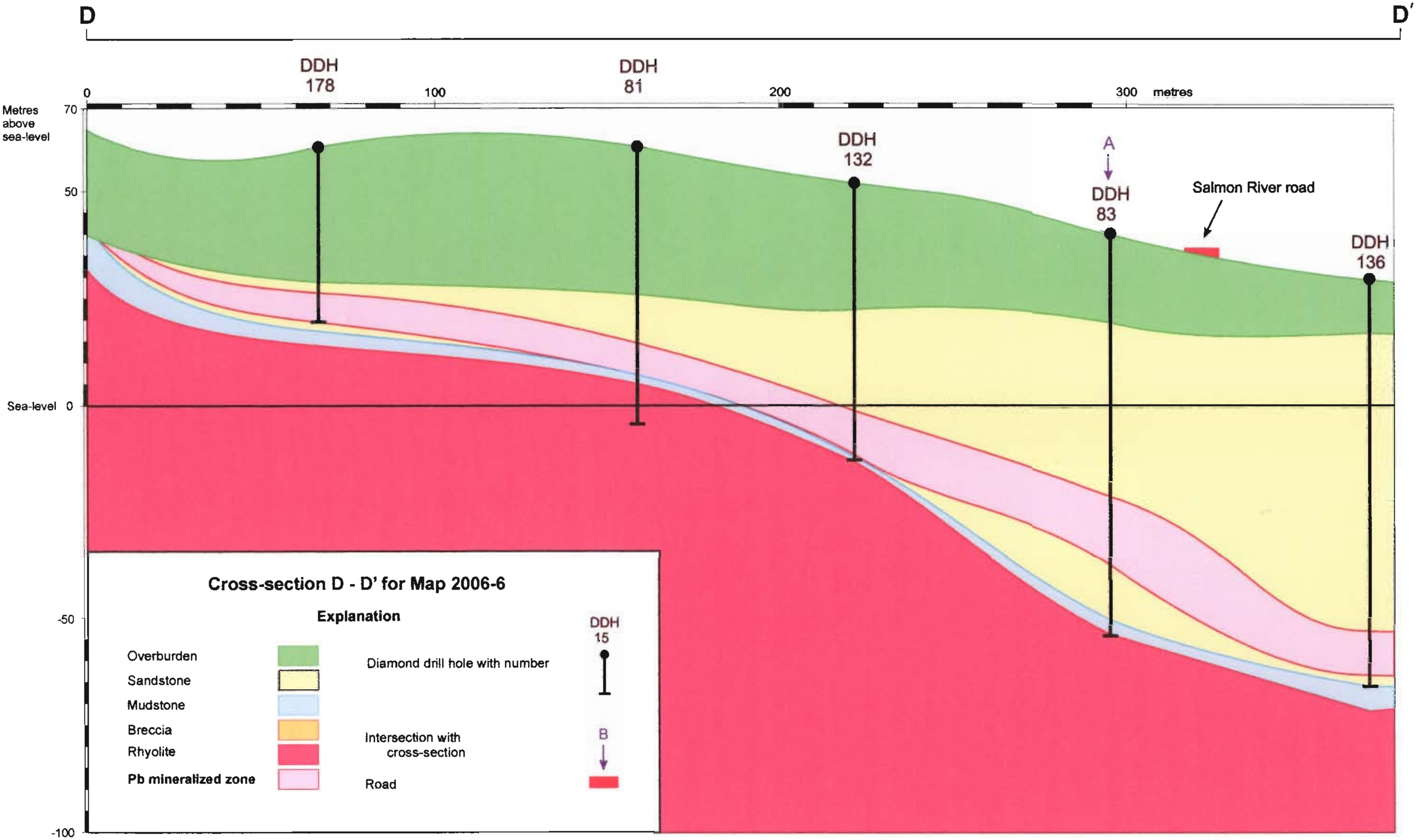


Cross-section A - A' for Map 2006-6

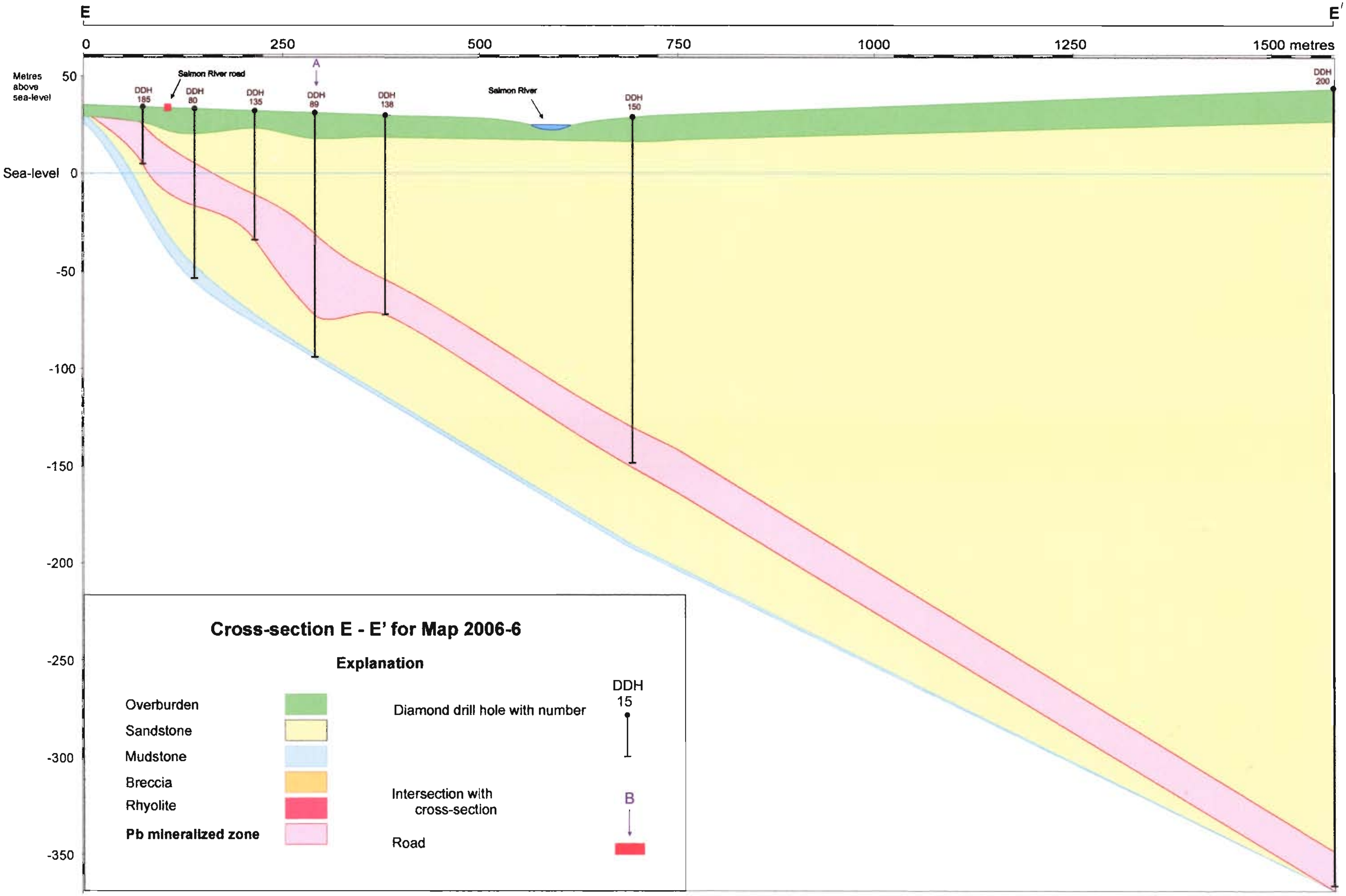

Cross-section B - B' for Map 2006-6



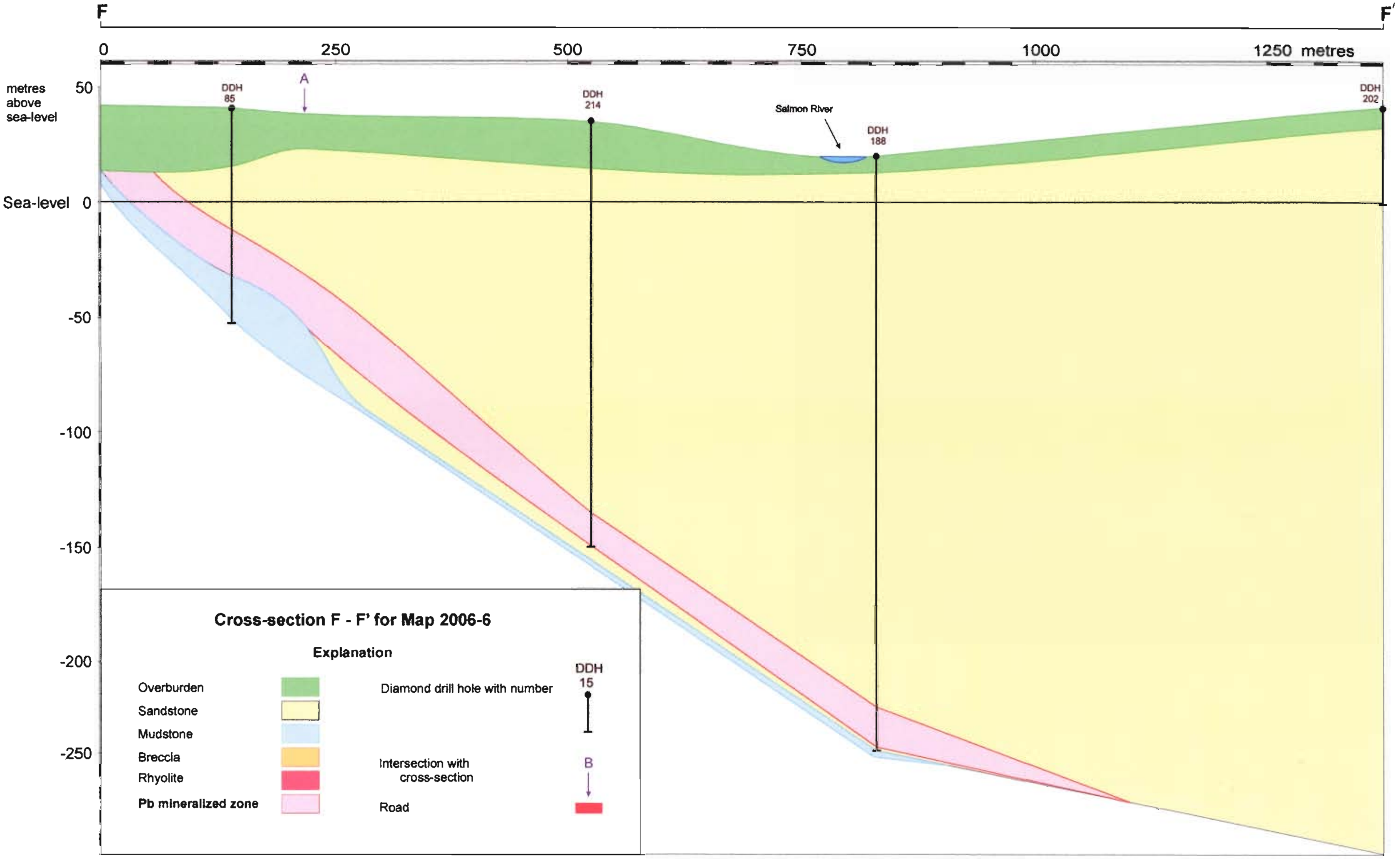
Cross-section C - C' for Map 2006-6



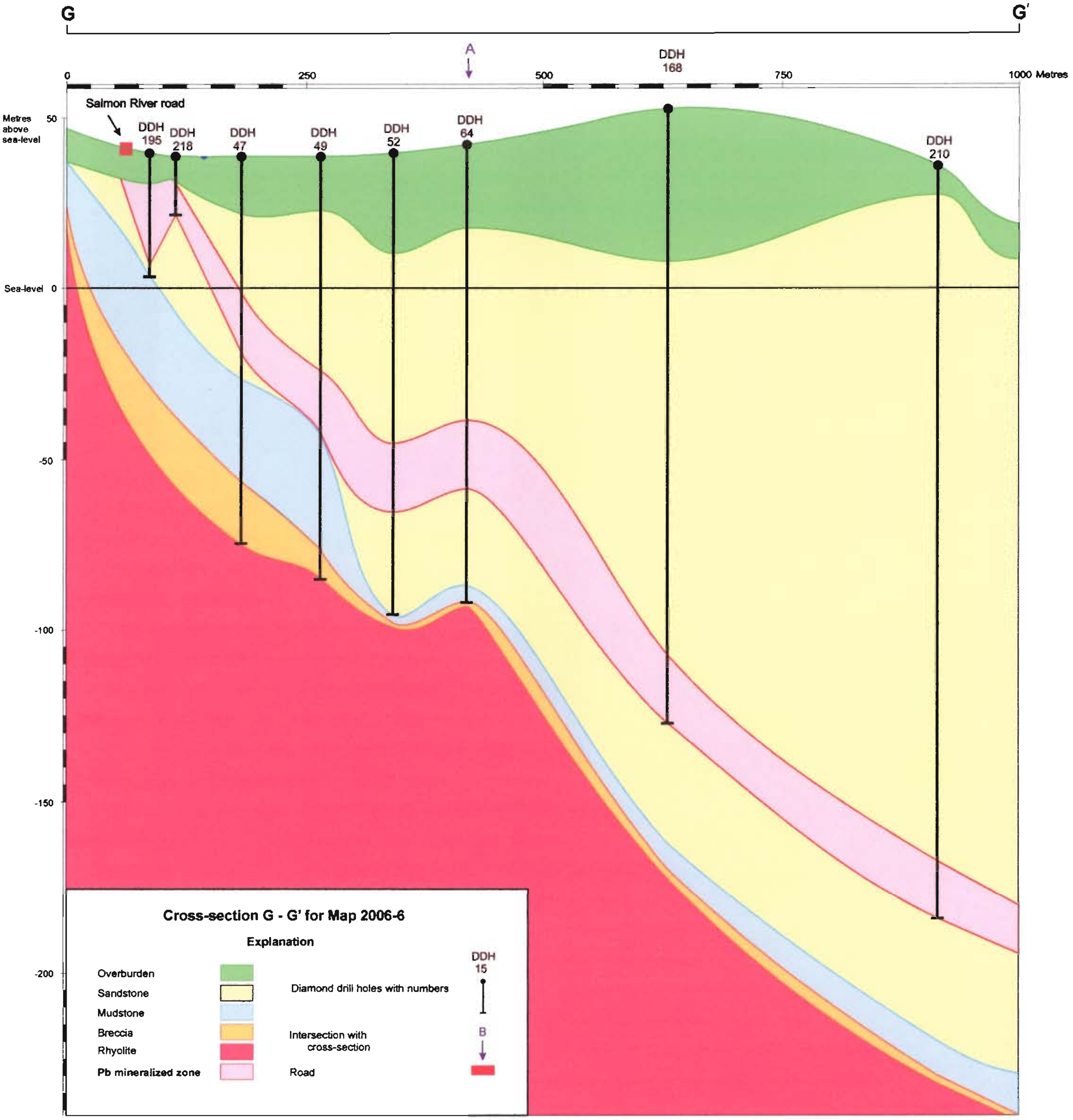
Cross-section D - D' for Map 2006-6



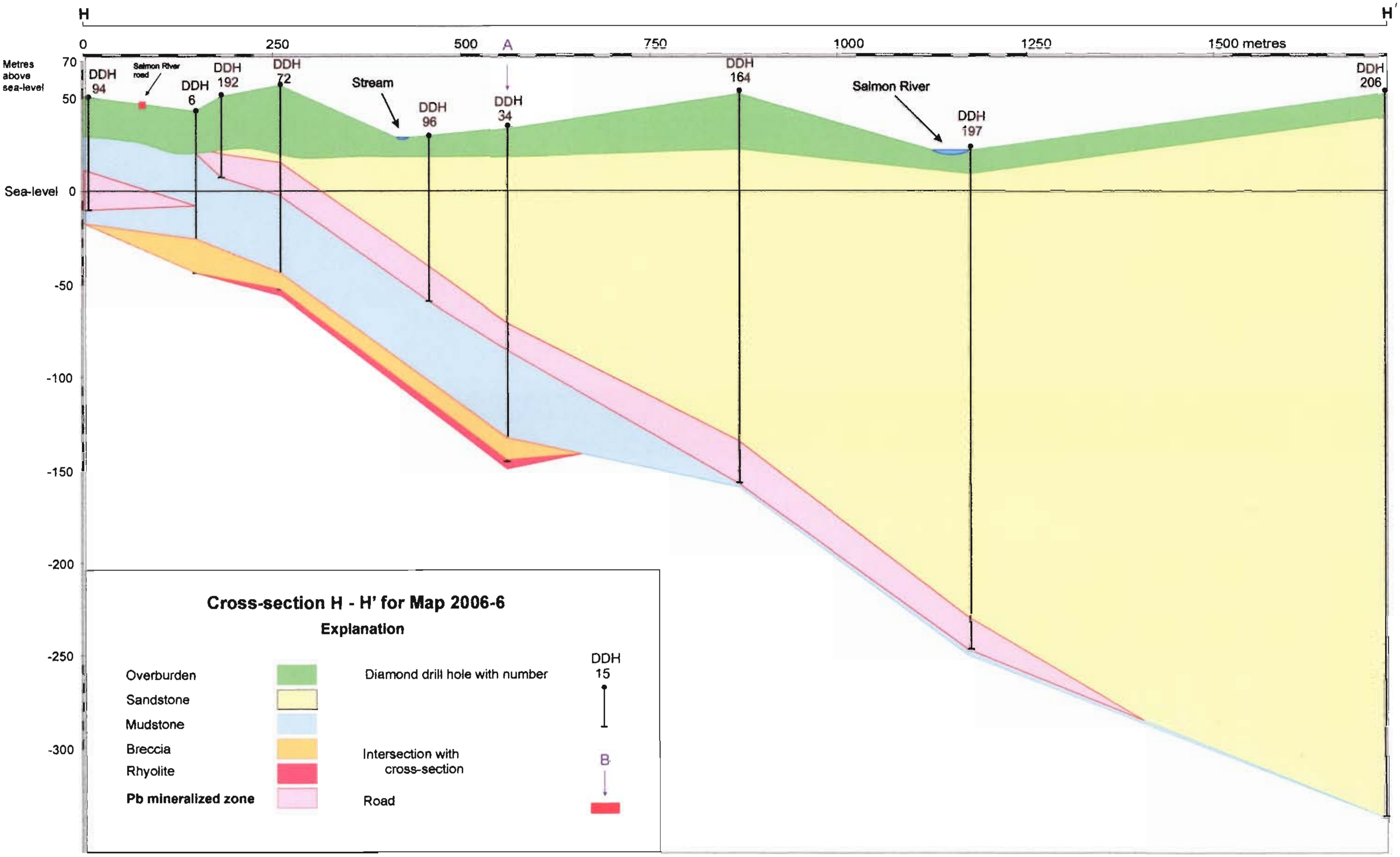
E
E'
0
250
500
750
1000
1250
1500 metres
Metres above sea-level
50
Sea-level 0
-50
-100
-150
-200
-250
-300
-350
Salmon River road
A
Salmon River
DDH 185
DDH 80
DDH 135
DDH 89
DDH 138
DDH 150
DDH 200
Cross-section E - E' for Map 2006-6
Explanation
Overburden
Sandstone
Mudstone
Breccia
Rhyolite
Pb mineralized zone
Diamond drill hole with number
DDH 15
Intersection with cross-section
B
Road



Cross-section F - F' for Map 2006-6



Cross-section G - G' for Map 2006-6



Cross-section H - H' for Map 2006-6

Explanation

Appendix D

Appendix D

To

GEOLOGIC AND HYDROGEOLOGIC INVESTIGATIONS
AT THE
SILVER MINE PROPERTY
CAPE BRETON COUNTY, NOVA SCOTIA

For the year ending 30 April 2006

ASSAY AND PERMEABILITY DATA
FROM CORE DDH 03-05

Appendix D
YTI DDH 03-05 Assay Values
25-Oct-05

ANALYTE	Ag	Al2O3	As	Au	B	Ba	Be	Bi	C	CaO	Cd	Ce	Cl
METHOD	**MERICP**	**MERXRF**	**MERAA**	**MERFA**	**MERICP**	**MERMS1**	**MERICP**	**MERMS1**	**MERLEC**	**MERXRF**	**MERMS1**	**MERMS1**	**MERCL**
DETECTION	**2**	**0.01**	**0.5**	**2**	**10**	**0.5**	**0.5**	**0.5**	**0.01**	**0.01**	**0.2**	**0.1**	**50**
UNITS	**PPM**	**%**	**PPM**	**PPB**	**PPM**	**PPM**	**PPM**	**PPM**	**%**	**%**	**PPM**	**PPM**	**PPM**
65751	<2	16.3	24.1	<2	120	397	4.2	<0.5	0.72	3.53	<0.2	93.9	160
65752	<2	12.6	24.1	<2	40	483	2.1	<0.5	1.13	5.62	<0.2	61.6	110
65753	<2	9.08	3.2	<2	90	1090	1.2	<0.5	1.96	9.74	<0.2	56.3	120
65754	<2	17.3	10.6	<2	80	561	3.2	<0.5	0.34	1.76	<0.2	92.3	140
65755	<2	20.3	14.3	<2	130	922	3.2	0.5	0.66	2.66	<0.2	175	150
65756	<2	21	12.8	<2	160	483	3.9	1.1	0.95	2.69	<0.2	100	190
65757	<2	12.9	4	<2	120	440	2.1	<0.5	0.87	3.88	<0.2	54.8	150
65758	<2	8.5	3.2	<2	130	2060	1.1	<0.5	3.63	17.8	0.2	39.2	120
65759	<2	8.18	3.1	<2	60	1710	1	<0.5	6.15	26.6	0.2	41.8	130
65760	<2	12.6	4.5	<2	100	2010	1.8	<0.5	2.3	9.04	<0.2	68.7	160
65761	<2	12.4	3.5	<2	90	1540	1.9	<0.5	3.38	11.3	<0.2	60.6	240
65762	<2	13.3	5.1	<2	140	2750	2	<0.5	3.4	11.4	<0.2	60.6	250
65763	<2	14.4	6.1	<2	90	1000	2.6	<0.5	3.08	11.1	<0.2	58	320
65764	<2	11.5	2.3	<2	50	>10000	1.9	<0.5	1.51	6.62	<0.2	25.5	260
65765	<2	11.2	2.7	3	90	>10000	2	<0.5	2.93	11.5	<0.2	36.8	270
65766	<2	15.4	6.1	<2	110	1520	3	<0.5	2.32	7.94	<0.2	59.1	380
65767	<2	12	4	3	90	749	1.8	<0.5	3.9	13.6	<0.2	53	370
65768	<2	16.4	4.8	<2	140	400	3	<0.5	2.14	6.82	<0.2	69.9	490
65769	<2	13.6	4.1	<2	120	392	2.2	0.8	3.01	10.9	<0.2	68.7	430
65770	<2	16	4.6	2	80	440	2.4	<0.5	2.17	7.46	<0.2	65.6	540
65771	4	13.6	42.7	2	100	486	1.9	<0.5	4.38	13.8	19	59.7	530
65772	<2	13.3	20.3	<2	120	3360	1.8	<0.5	2.9	10.1	2	58.3	520
65773	<2	10.9	17.7	<2	20	1940	1.4	<0.5	2.98	11	2.8	53.2	500
65774	<2	13.6	27.1	<2	80	547	1.9	<0.5	2.71	9.3	1.4	62	580
65775	<2	14	13.8	2	100	1070	2	<0.5	3.07	12.2	<0.2	62.6	640
65776	<2	12.6	32.7	2	110	2380	1.8	<0.5	3.6	13.8	0.4	58.1	650
65777	<2	10.7	11.5	2	110	2170	1.5	<0.5	2.5	15.7	<0.2	50	630
65778	2	11.2	14.2	<2	110	3440	1.5	<0.5	2	12.7	0.9	51.3	740
65779	4	12	27.6	<2	100	2010	1.8	<0.5	2.8	11.3	17.8	65.3	800
65780	2	13.2	15.1	<2	40	1310	2.3	<0.5	2.49	8.59	<0.2	69.1	960
65781	<2	12.9	4.8	<2	170	485	2.8	<0.5	2.73	9.45	<0.2	63.7	970
65782	<2	11	3.7	<2	70	1120	2.5	<0.5	2.41	8.9	<0.2	71.4	910
65783	3	11.8	14	<2	30	543	3	<0.5	2.28	7.75	0.6	72.9	920
65784	7	11.6	26	3	120	502	4.3	2.5	3.13	7.42	1.6	61.2	960
DUP-65751	<2	16.3	25	<2	100	379	4.4	<0.5	0.68	3.51	<0.2	90.3	160
DUP-65763	<2	14.4	5.5	<2	60	946	2.5	<0.5	3.1	11.1	<0.2	59.6	310
DUP-65775	<2	14	13.9	2	80	1020	2	<0.5	3	12.2	<0.2	59.3	660

ANALYTE	Co	Cr	Cr2O3	Cs	Cu	Dy	Er	Eu	Fe2O3	Ga	Gd	Ge	Hf
METHOD	MERICP	MERMS1	MERXRF	MERMS1	MERICP	MERMS1	MERMS1	MERMS1	MERXRF	MERMS1	MERMS1	MERMS1	MERMS1
DETECTION	1	2	0.01	0.5	0.5	0.05	0.05	0.05	0.01	1	0.05	1	0.2
UNITS	PPM	PPM	%	PPM	PPM	PPM	PPM	PPM	%	PPM	PPM	PPM	PPM
65751	20	100	0.01	18.4	23.3	5.31	3.59	1.28	5.95	18	6.2	2	8
65752	16	105	0.01	9.2	13.1	5.18	3.34	1.23	3.94	13	5.66	2	8.4
65753	8	159	0.02	5.2	9.3	5.19	3.3	1.23	3.73	9	5.73	2	10.1
65754	13	106	0.01	15.4	8.1	5.66	3.68	1.46	7.78	19	6.28	2	7.2
65755	15	191	0.01	35.6	9	9.92	6.14	2.59	9.03	43	11.3	5	8
65756	18	107	0.01	21.5	16.6	5.19	3.01	1.53	5.59	24	7.32	2	3.9
65757	8	64	<0.01	21.9	9.7	4.93	2.89	1.37	3.44	14	5.86	1	8.8
65758	4	70	<0.01	19.7	2.1	2.99	1.99	0.86	2.48	9	3.52	1	7.4
65759	6	77	<0.01	14.2	1.3	2.78	1.59	0.96	3.26	9	3.69	1	2.7
65760	8	75	<0.01	12.7	2.5	4.29	2.7	1.22	4.62	14	5.08	2	5.4
65761	8	80	<0.01	12.1	3.4	3.7	2.4	1.01	4.29	14	4.44	1	5
65762	9	75	<0.01	14	4.6	3.82	2.42	1.1	4.99	15	4.59	2	4.6
65763	10	81	<0.01	14.4	3.7	3.63	2.32	0.93	5.57	16	4.36	2	4.1
65764	3	77	<0.01	6.4	8.8	2.08	1.35	1.21	1.29	15	2.17	1	4.9
65765	8	96	0.01	8.7	7	2.55	1.66	1.47	2.38	13	2.89	1	4.2
65766	10	92	<0.01	16.5	6.8	3.38	2.06	1.04	6.35	18	4.19	2	3.8
65767	8	84	<0.01	10.8	7.5	3.67	2.19	0.93	4.55	13	4.13	1	3.7
65768	11	80	<0.01	15.2	6.5	3.74	2.29	1.14	6.79	18	4.64	2	3.5
65769	13	103	<0.01	12.1	8.5	4.29	2.65	1.14	4.54	15	4.85	2	4.1
65770	14	88	<0.01	14.6	34.8	3.63	2.24	1.1	5.56	18	4.77	2	3.5
65771	18	80	<0.01	9.6	84.3	3.62	2.22	1.05	4.42	16	4.51	2	3.1
65772	11	74	<0.01	9.1	25.2	3.67	2.47	1.33	4.17	14	4.46	2	5.4
65773	8	137	0.02	6.7	25.2	3.55	2.2	1.08	3.41	12	4.12	1	5.2
65774	12	87	<0.01	9.8	27.7	4.16	2.61	1.06	4.47	16	4.5	2	6
65775	10	89	<0.01	11	11.7	3.63	2.31	1.11	4.37	16	4.29	2	4.7
65776	11	71	<0.01	9	23.6	3.61	2.06	1.05	4.26	14	4.13	1	3.5
65777	7	61	<0.01	7	15.2	3.24	1.97	0.92	3.36	11	3.66	1	4.6
65778	9	77	<0.01	7.3	20.6	3.48	2.3	1.03	3.48	12	3.85	1	4.8
65779	14	78	<0.01	8.7	34.8	4.01	2.62	1.14	3.78	14	4.92	2	6
65780	13	89	<0.01	10.6	21.1	4.87	2.95	1.14	3.87	15	5.37	2	8.1
65781	8	66	<0.01	10.6	23.1	3.91	2.65	1.09	3.89	15	4.88	2	6.8
65782	6	78	<0.01	8.4	7.1	4.95	3.22	1.1	3.22	13	5.46	2	11.5
65783	11	108	0.01	9	22.4	4.72	3.09	1.17	3.63	14	5.66	2	9.2
65784	15	88	0.01	8	21.9	3.75	2.56	0.95	2.38	15	4.57	2	9.7
DUP-65751	21	95	0.01	17.8	22.5	5.44	3.48	1.37	5.94	18	5.89	2	7.7
DUP-65763	10	79	<0.01	15.1	4.2	3.65	2.32	1.02	5.59	16	4.5	2	4.2
DUP-65775	10	81	<0.01	10.6	11.5	3.58	2.29	0.97	4.37	15	4.34	1	3.9

ANALYTE	**Hg**	**Ho**	**In**	**Ir**	**K2O**	**La**	**Li**	**LOI**	**Lu**	**MgO**	**MnO**	**Mo**	**Na2O**
METHOD	**MERHG**	**MERMS1**	**MERMS1**	**FAM363**	**MERXRF**	**MERMS1**	**MERICP**	**MERXRF**	**MERMS1**	**MERXRF**	**MERXRF**	**MERICP**	**MERXRF**
DETECTION	**5**	**0.05**	**0.5**	**0.5**	**0.01**	**0.1**	**1**	**0.01**	**0.05**	**0.03**	**0.01**	**1**	**0.02**
UNITS	**PPB**	**PPM**	**PPM**	**PPB**	**%**	**PPM**	**PPM**	**%**	**PPM**	**%**	**%**	**PPM**	**%**
65751	25	1.2	<0.5	<0.5	3.69	46.2	61	6.1	0.54	2.35	0.16	2	0.25
65752	29	1.14	<0.5	<0.5	2.56	29	48	6.85	0.48	1.86	0.27	2	0.19
65753	<5	1.14	<0.5	<0.5	1.74	24.5	34	9.6	0.47	1.26	0.41	2	0.1
65754	<5	1.26	<0.5	<0.5	3.97	44.5	64	5.3	0.48	2.42	0.13	3	0.2
65755	<5	2.14	<0.5	<0.5	5.06	83	68	6.8	0.9	2.86	0.13	3	0.25
65756	6	1.09	<0.5	<0.5	5.66	47.8	63	7	0.44	2.72	0.12	8	0.24
65757	<5	1.04	<0.5	<0.5	3.39	24.5	36	5.95	0.4	1.71	0.16	2	0.18
65758	<5	0.66	<0.5	<0.5	2.4	18.8	18	15.7	0.26	1.22	0.31	2	0.18
65759	<5	0.59	<0.5	<0.5	2.2	19.7	22	24.8	0.2	3.13	0.58	2	0.2
65760	<5	0.93	<0.5	<0.5	3.39	32.9	32	11.5	0.36	3.48	0.5	3	0.22
65761	<5	0.81	<0.5	<0.5	3.51	28	33	15.2	0.31	5.45	0.83	2	0.23
65762	<5	0.83	<0.5	<0.5	3.76	29.3	36	16	0.34	5.8	0.89	2	0.22
65763	<5	0.79	<0.5	<0.5	4.27	27.9	39	14.8	0.36	5.1	0.53	3	0.27
65764	<5	0.46	<0.5	<0.5	3.77	11.1	22	7.15	0.19	1.32	0.31	<1	2.02
65765	<5	0.58	<0.5	<0.5	3.39	17.1	24	13.5	0.24	3.48	0.48	2	0.9
65766	<5	0.71	<0.5	<0.5	4.87	29.1	37	12.1	0.32	5.02	0.36	2	0.24
65767	6	0.78	<0.5	<0.5	3.73	24.3	34	17.5	0.32	5.92	0.54	2	0.3
65768	16	0.81	<0.5	<0.5	4.97	34.9	53	11.7	0.29	5.18	0.37	3	0.26
65769	16	0.95	<0.5	<0.5	4.13	31.9	46	14.2	0.37	4.84	0.49	2	0.32
65770	9	0.84	<0.5	<0.5	4.68	32.5	57	11.8	0.36	4.99	0.33	2	0.25
65771	21	0.77	<0.5	<0.5	3.35	27.8	62	15.8	0.32	5.21	0.75	8	0.07
65772	8	0.83	<0.5	<0.5	3.48	28.5	59	11.6	0.34	3.61	0.47	5	0.25
65773	8	0.78	<0.5	<0.5	2.6	25.1	49	11.5	0.29	3.03	0.56	4	0.26
65774	9	0.91	<0.5	<0.5	3.55	30.1	62	11.4	0.37	4.05	0.56	5	0.27
65775	6	0.81	<0.5	<0.5	3.67	29.8	64	13.7	0.35	4.27	0.6	3	0.3
65776	10	0.72	<0.5	<0.5	3.15	26.5	55	13.9	0.28	4.43	0.76	6	0.31
65777	6	0.63	<0.5	<0.5	2.59	24	47	10.5	0.27	3.37	0.51	<1	0.3
65778	<5	0.76	<0.5	<0.5	2.67	24.4	53	9.1	0.31	3.5	0.62	<1	0.4
65779	13	0.9	<0.5	<0.5	3.16	30.2	59	9.75	0.37	3.2	0.66	4	0.34
65780	15	1.02	<0.5	<0.5	3.9	32.3	66	10.9	0.4	3.87	0.76	4	0.57
65781	8	0.93	<0.5	<0.5	3.86	30.6	65	12.1	0.38	4.13	0.84	2	0.55
65782	9	1.15	<0.5	<0.5	3.08	32.6	54	11	0.6	3.76	0.76	2	0.51
65783	14	1.09	<0.5	<0.5	3.32	33.4	60	9.55	0.46	3.11	0.63	5	0.53
65784	16	0.88	<0.5	<0.5	3.53	28	59	10.1	0.37	3.66	0.9	3	0.67
DUP-65751	24	1.2	<0.5	<0.5	3.68	44.6	62	6.1	0.53	2.34	0.16	3	0.25
DUP-65763	<5	0.81	<0.5	<0.5	4.28	28.7	39	14.8	0.35	5.12	0.54	2	0.25
DUP-65775	6	0.8	<0.5	<0.5	3.66	28.5	61	13.7	0.3	4.26	0.6	2	0.3

ANALYTE	Nb	Nd	Ni	Os	P2O5	Pb	Pd	Pr	Pt	Rb	Re	Rh	Ru
METHOD	MERMS1	MERMS1	MERICP	FAM363	MERXRF	MERICP	FAM363	MERMS1	FAM363	MERMS1	FAM363	FAM363	FAM363
DETECTION	0.1	0.1	1	3	0.01	2	1	0.05	1	0.2	1	1	1
UNITS	PPM	PPM	PPM	PPB	%	PPM	PPB	PPM	PPB	PPM	PPB	PPB	PPB
65751	16.9	39.4	41	<3	0.14	17	1	10.1	1	136	<1	<1	<1
65752	13.9	30.6	33	<3	0.1	11	<1	7.15	<1	91.8	<1	<1	<1
65753	12	30.3	24	<3	0.09	7	<1	6.7	<1	59.4	<1	<1	<1
65754	16.2	40.3	42	<3	0.14	29	2	10.1	2	144	<1	<1	<1
65755	27.7	78.5	48	<3	0.18	61	1	19.4	2	335	<1	<1	<1
65756	13.2	44.3	48	<3	0.36	21	1	11	1	201	<1	<1	<1
65757	13.1	29.3	28	<3	0.26	14	1	6.58	2	134	<1	<1	<1
65758	9.4	19.4	17	<3	0.02	31	2	4.5	4	91.9	<1	<1	1
65759	7	20.7	22	<3	0.04	21	2	4.81	3	79.9	<1	<1	<1
65760	12.7	30.8	30	<3	0.15	6	<1	7.47	<1	116	<1	<1	<1
65761	11.1	26.4	32	<3	0.18	5	<1	6.6	<1	120	<1	<1	<1
65762	10.7	28.7	39	<3	0.21	6	1	6.91	<1	127	<1	<1	<1
65763	10.1	26.5	38	<3	0.21	5	<1	6.44	<1	136	<1	<1	<1
65764	13.1	12.8	11	<3	0.08	15	1	3.1	2	144	<1	<1	<1
65765	10.3	17.8	24	<3	0.16	7	<1	4.3	<1	113	<1	<1	<1
65766	11.2	26	41	<3	0.22	4	2	6.38	2	150	<1	<1	<1
65767	10.5	25	28	<3	0.13	9	2	6.01	<1	115	<1	<1	<1
65768	11.6	30.2	40	<3	0.22	7	<1	7.38	<1	151	<1	<1	<1
65769	11.5	30.3	40	<3	0.15	7	1	7.62	<1	132	<1	<1	<1
65770	11.4	29.9	38	<3	0.19	7	1	7.27	<1	148	<1	<1	<1
65771	10	27.8	37	<3	0.17	1560	2	6.52	2	112	5	<1	<1
65772	12.6	26.2	32	<3	0.17	452	<1	6.36	<1	109	4	<1	<1
65773	10.7	24.5	24	<3	0.13	563	1	5.89	<1	85.1	1	<1	<1
65774	12.6	27.8	33	<3	0.18	111	<1	6.87	<1	117	2	<1	<1
65775	11.3	28.3	31	<3	0.26	38	<1	6.91	<1	123	3	<1	<1
65776	9.7	26.7	30	<3	0.2	96	1	6.29	<1	101	2	<1	<1
65777	9.5	22.7	22	<3	0.16	61	<1	5.54	<1	85.2	<1	<1	<1
65778	10.9	23.8	25	<3	0.15	3160	<1	5.68	1	87.7	<1	<1	<1
65779	12	30.4	30	<3	0.16	8950	<1	7.4	<1	105	2	<1	<1
65780	14.4	32	30	<3	0.19	146	1	7.63	<1	122	2	<1	<1
65781	13.8	30.1	26	<3	0.16	78	<1	7.25	<1	125	<1	<1	<1
65782	16	32.9	24	<3	0.1	14	<1	8.19	<1	103	<1	<1	<1
65783	15.3	33.3	28	<3	0.14	4050	<1	8.06	<1	111	2	<1	<1
65784	15.4	28.2	27	<3	0.14	>10000	<1	6.7	<1	117	3	<1	<1
DUP-65751	16.2	39.3	43	<3	0.15	24	1	9.75	<1	135	<1	<1	<1
DUP-65763	10	27.5	39	<3	0.21	7	2	6.81	<1	140	<1	<1	<1
DUP-65775	10.5	27	30	<3	0.25	44	2	6.53	2	116	2	<1	<1

ANALYTE	S	Sb	Sc	Se	SiO2	Sm	Sn	Sr	SUM	Ta	Tb	Te	Th
METHOD	MERLEC	MERMS1	MERICP	MERSE	MERXRF	MERMS1	MERMS1	MERMS1	MERXRF	MERMS1	MERMS1	MERSE	MERMS1
DETECTION	0.01	0.5	0.5	0.5	0.01	0.1	2	0.1	0.01	0.5	0.05	0.5	0.1
UNITS	%	PPM	PPM	PPM	%	PPM	PPM	PPM	%	PPM	PPM	PPM	PPM
65751	0.23	1.2	15	<0.5	58.9	7.2	2	106	98.4	1.1	1.12	<0.5	11.9
65752	0.29	0.8	11.1	<0.5	64.1	6.7	<2	80.6	99	1	1.09	<0.5	9.6
65753	0.35	0.6	7.9	<0.5	63.1	6.7	<2	82.7	99.6	0.8	1.05	<0.5	8.1
65754	<0.01	1.4	16.8	<0.5	58.4	7.7	3	119	98.4	1.1	1.15	<0.5	12.5
65755	<0.01	3.4	19.8	<0.5	50.4	13.7	6	294	98.6	2	2.01	<0.5	23.4
65756	0.37	1	19.7	<0.5	52.5	8.4	3	181	98.6	0.9	1.15	<0.5	12.6
65757	<0.01	0.7	9.3	<0.5	66.9	6.3	<2	159	99.5	0.9	1.01	<0.5	10.4
65758	0.03	0.8	5.2	<0.5	50.7	3.9	<2	241	99.7	0.6	0.65	<0.5	7.7
65759	0.03	0.8	6.1	<0.5	30.6	4.2	<2	254	99.9	<0.5	0.61	<0.5	5.5
65760	0.06	0.8	9.6	<0.5	53	5.7	<2	177	99	0.9	0.88	<0.5	9.1
65761	0.03	0.7	9.7	<0.5	45.3	5.5	<2	167	99.3	0.7	0.74	<0.5	8.5
65762	0.07	1.1	10.6	<0.5	42.1	5.7	3	180	99.2	0.7	0.78	<0.5	9.3
65763	0.03	1.1	12.2	<0.5	43.2	4.9	<2	145	99.9	0.6	0.77	<0.5	8.6
65764	0.4	<0.5	3.5	<0.5	63.2	2.5	<2	799	97.5	1.1	0.39	<0.5	17.2
65765	0.47	0.5	6.4	<0.5	49.9	3.8	<2	527	97.3	0.8	0.56	<0.5	10.2
65766	0.05	1	13	<0.5	45.5	5.1	2	148	98.6	0.7	0.73	<0.5	9.6
65767	0.03	0.7	9.3	1.8	41.5	5	<2	153	100.2	0.7	0.74	<0.5	8.2
65768	0.02	0.7	15.1	<0.5	45.8	5.6	2	135	99.1	0.8	0.78	<0.5	10.3
65769	0.08	0.6	11.3	<0.5	46.3	5.8	2	156	100	0.8	0.92	<0.5	9.2
65770	0.03	0.5	14.3	<0.5	47.2	5.7	<2	139	99	0.8	0.8	<0.5	9.9
65771	1.2	2.1	11	<0.5	38	5.4	<2	182	95.7	0.7	0.75	<0.5	8.1
65772	0.99	0.9	10.6	0.8	48.7	4.8	<2	398	96.6	0.8	0.78	<0.5	8.6
65773	0.76	1	8	<0.5	53.1	4.6	<2	360	97.1	0.7	0.71	<0.5	8.1
65774	0.81	1.1	10.8	<0.5	49.4	5.5	3	185	97.5	0.8	0.83	<0.5	9.5
65775	1	0.9	11.4	0.8	42.5	5.5	<2	837	96.5	0.7	0.79	<0.5	9.2
65776	2.02	1.3	9.7	0.9	39.4	5.2	<2	1530	93.3	0.6	0.76	<0.5	7.9
65777	4.19	0.7	8.2	0.9	37	4.5	<2	2750	84.8	0.6	0.67	<0.5	7.3
65778	4.11	0.9	8.6	1.1	39.3	4.4	<2	3860	83.7	0.7	0.71	<0.5	8
65779	2.32	1.7	9.5	1	44.5	5.8	<2	987	89.5	0.8	0.82	<0.5	9.5
65780	0.36	0.9	10.3	<0.5	51.6	6.6	2	364	98.1	1	0.96	<0.5	11.7
65781	0.07	0.6	9.9	<0.5	50.5	5.5	<2	149	99	0.9	0.89	<0.5	11.6
65782	0.04	0.5	7.9	<0.5	56.1	6.3	2	205	99.2	1.1	1	<0.5	13.5
65783	0.27	1	8.5	1.5	56.1	6.5	<2	143	97.3	1.1	0.97	<0.5	12.4
65784	0.77	1	8	0.8	50.7	5.3	<2	146	91.6	1.1	0.78	<0.5	14
DUP-65751	0.26	1.1	16.2	<0.5	58.7	7	2	106	98.2	1.1	1.05	<0.5	11.8
DUP-65763	NVL	1	12.3	<0.5	43.2	5.4	2	146	100	0.7	0.77	<0.5	8.3
DUP-65775	1.06	0.8	10.9	0.8	42.3	5.2	2	775	96.3	0.7	0.74	<0.5	8.9

ANALYTE	TiO2	Tl	Tm	U	V	W	Y	Yb	Zn	Zr
METHOD	MERXRF	MERMS1	MERMS1	MERMS1	MERICP	MERMS1	MERMS1	MERMS1	MERICP	MERMS1
DETECTION	0.01	0.1	0.05	0.05	2	1	0.5	0.1	0.5	0.5
UNITS	%	PPM	PPM	PPM	PPM	PPM	PPM	PPM	PPM	PPM
65751	1.03	1.6	0.51	3.41	104	2	31.9	3.5	156	256
65752	0.88	1.1	0.49	2.92	75	2	31.7	3.3	118	270
65753	0.72	0.6	0.49	2.42	53	1	31.2	3.2	77.7	342
65754	0.97	1.5	0.52	3.09	116	2	31.7	3.4	146	231
65755	0.85	3.6	0.9	5.99	134	4	53.3	5.8	152	259
65756	0.77	2.4	0.46	6.03	131	1	29.4	3.1	150	123
65757	0.7	2.1	0.46	4.88	78	1	28.7	2.8	84.6	290
65758	0.45	1.9	0.26	2.42	47	1	18.8	1.9	44.2	257
65759	0.38	1.3	0.24	1.4	51	<1	16.3	1.5	41.2	82.4
65760	0.67	1.3	0.37	2.99	71	2	24.3	2.6	53.6	174
65761	0.63	1.3	0.34	2.71	74	1	22.2	2.2	56.4	156
65762	0.61	1.4	0.32	4.08	83	1	22.4	2.4	65	150
65763	0.6	1.4	0.33	2.42	84	2	20.7	2.1	77	137
65764	0.27	1.1	0.21	2.29	24	<1	14.5	1.4	34	109
65765	0.47	1	0.23	2.24	48	1	15.7	1.6	44.7	112
65766	0.63	1.5	0.3	2.76	91	1	18.4	2.1	59.2	123
65767	0.52	1	0.33	2.96	73	1	20.9	2	49.7	115
65768	0.71	1.4	0.34	2.98	100	1	20.5	2.2	68.6	108
65769	0.62	1.2	0.38	3.39	79	1	25.2	2.6	56.4	133
65770	0.68	1.4	0.35	3.1	98	2	21	2.3	67	110
65771	0.56	2.2	0.28	7.39	79	2	20.4	2	2610	98.5
65772	0.72	1.3	0.38	4.01	76	2	22	2.5	252	179
65773	0.61	1.1	0.32	3.19	55	1	20.5	2.1	385	169
65774	0.68	1.3	0.38	4.16	77	2	24.1	2.8	211	195
65775	0.6	1.2	0.32	3.89	82	2	21.7	2.1	73.1	141
65776	0.57	1.5	0.28	3.46	69	1	20	2	102	113
65777	0.54	0.9	0.3	2.94	57	2	18.5	1.9	56.5	153
65778	0.58	1.3	0.33	2.94	60	1	20.3	2.1	170	155
65779	0.62	2.4	0.38	3.83	65	1	24.5	2.5	1820	193
65780	0.69	1.4	0.42	4.98	74	1	27.5	2.9	78.6	263
65781	0.62	1	0.39	4.93	69	1	25	2.6	91.5	227
65782	0.7	0.8	0.49	4.62	52	1	31	3.4	100	374
65783	0.71	2.5	0.44	6.28	58	2	29.4	3	152	287
65784	0.55	3.6	0.37	12.1	51	1	25.4	2.8	264	308
DUP-65751	1.01	1.6	0.48	3.46	110	2	32.3	3.3	160	250
DUP-65763	0.58	1.4	0.33	2.57	85	2	21.2	2.3	77.1	143
DUP-65775	0.61	1.2	0.29	3.79	80	1	20.2	2.2	70.1	130

HYDRAULIC CONDUCTIVITY TEST

ASTM D 5084 (CONSTANT HEAD)

SAMPLE IDENTIFICATION

PROJECT NUMBER	06-1116-004	SAMPLE	-
PROJECT TITLE	Yava / Lab Testing / Toronto	SAMPLE DEPTH, m	139.5-139.595
DRILL HOLE	03-5	DATE	2/7/2006

SPECIMEN PROPERTIES AND DIMENSIONS (INITIAL)

SAMPLE HEIGHT, cm	4.67	UNIT WEIGHT, kN/m^3	24.71
SAMPLE DIAMETER, cm	3.62	DRY UNIT WEIGHT, kN/m^3	0.00
SAMPLE AREA, cm^2	10.29	SPECIFIC GRAVITY, assumed	2.70
SAMPLE VOLUME, cm^3	48.09	VOLUME OF SOLIDS, cm^3	0.00
TOTAL MASS, g	121.19	VOLUME OF VOIDS, cm^3	46.09
DRY MASS, g		VOID RATIO	#DIV/0!
WATER CONTENT, %	#DIV/0!		

SATURATION STAGE

CELL PRESSURE, kPa	-	EFFECTIVE CONFINING STRESS,	#VALUE!
HEAD PRESSURE, kPa	-	DURATION, min	-
BACK PRESSURE, kPa	-	B COEFFICIENT	-

CONSOLIDATION STAGE

CELL PRESSURE, kPa	-	EFFECTIVE CONFINING STRESS,	#VALUE!
HEAD PRESSURE, kPa	-	DURATION, min	
BACK PRESSURE, kPa	-	VOLUME CHANGE, cm^3	
		DRAINAGE	Top and Bottom

HYDRAULIC CONDUCTIVITY TEST

CELL PRESSURE, kPa	600	EFFECTIVE CONFINING STRESS,	500
HEAD PRESSURE, kPa	100	DURATION, min	38700
BACK PRESSURE, kPa	50	HYDRAULIC GRADIENT, i	109

SPECIMEN PROPERTIES AND DIMENSIONS (FINAL)

SAMPLE HEIGHT, cm	4.67	UNIT WEIGHT, kN/m^3	0.00
SAMPLE DIAMETER, cm	3.62	DRY UNIT WEIGHT, kN/m^3	0.00
SAMPLE AREA, cm^2	10.29	SPECIFIC GRAVITY, assumed	2.70
SAMPLE VOLUME, cm^3	48.09	VOLUME OF SOLIDS, cm^3	0.00
TOTAL MASS, g		VOLUME OF VOIDS, cm^3	46.09
DRY MASS, g		VOID RATIO	#DIV/0!
WATER CONTENT, %	#DIV/0!		

TEST RESULTS

ELAPSED TIME TO STEADY STATE FLOW (min)	00
DURATION OF STEADY STATE FLOW (min)	38700
INFLOW VOLUME UNDER STEADY STATE FLOW (cm^3)	6.0
OUTFLOW VOLUME UNDER STEADY STATE FLOW (cm^3)	-0.1
HYDRAULIC CONDUCTIVITY (INFLOW) (cm/s)	2.32E-09
HYDRAULIC CONDUCTIVITY (OUTFLOW) (cm/s)	-3.03E-11
HYDRAULIC CONDUCTIVITY, K, cm/s	**1.14E-09**

NOTES:

MIXING FLUID	
PERMEANT FLUID	tap water

HYDRAULIC CONDUCTIVITY TEST

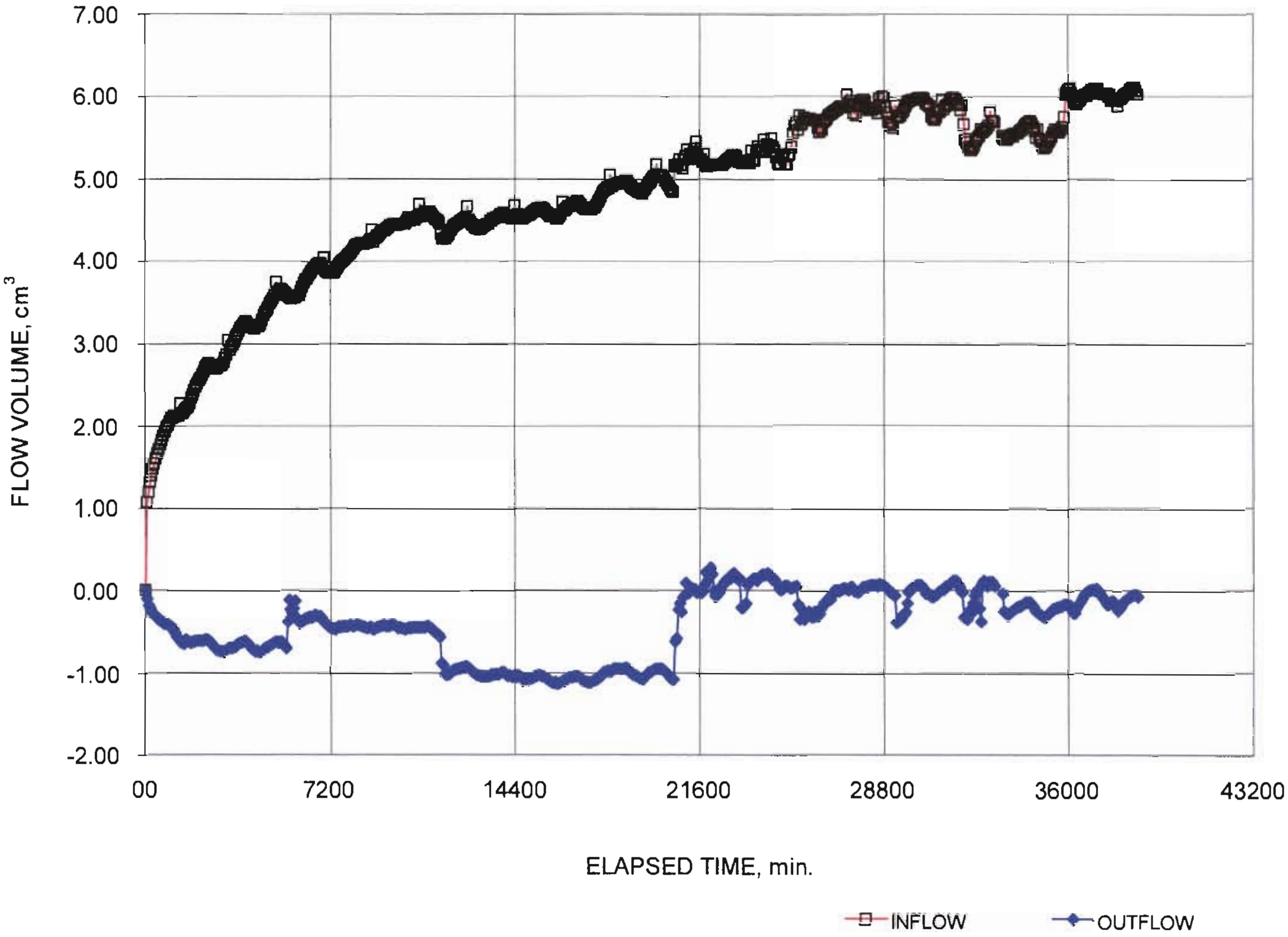
Golder Associates

Project No. 06-1116-004

Prepared By: LFG

Checked By: MM

Drill Hole 03-05 Sample Depth 139.5-139.595m



HYDRAULIC CONDUCTIVITY TEST

ASTM D 5084 (CONSTANT HEAD)

SAMPLE IDENTIFICATION

PROJECT NUMBER	06-1116-004	SAMPLE	-
PROJECT TITLE	Yava / Lab Testing / Toronto	SAMPLE DEPTH, m	139.5-139.595
DRILL HOLE	03-5	DATE	2/7/2006

SPECIMEN PROPERTIES AND DIMENSIONS (INITIAL)

SAMPLE HEIGHT, cm	4.67	UNIT WEIGHT, kN/m^3	24.71
SAMPLE DIAMETER, cm	3.62	DRY UNIT WEIGHT, kN/m^3	0.00
SAMPLE AREA, cm^2	10.29	SPECIFIC GRAVITY, assumed	2.70
SAMPLE VOLUME, cm^3	48.09	VOLUME OF SOLIDS, cm^3	0.00
TOTAL MASS, g	121.19	VOLUME OF VOIDS, cm^3	48.09
DRY MASS, g		VOID RATIO	#DIV/0!
WATER CONTENT, %	#DIV/0!		

SATURATION STAGE

CELL PRESSURE, kPa	-	EFFECTIVE CONFINING STRESS,	#VALUE!
HEAD PRESSURE, kPa	-	DURATION, min	-
BACK PRESSURE, kPa	-	*B* COEFFICIENT	-

CONSOLIDATION STAGE

CELL PRESSURE, kPa	-	EFFECTIVE CONFINING STRESS,	#VALUE!
HEAD PRESSURE, kPa	-	DURATION, min	
BACK PRESSURE, kPa	-	VOLUME CHANGE, cm^3	
		DRAINAGE	Top and Bottom

HYDRAULIC CONDUCTIVITY TEST

CELL PRESSURE, kPa	600	EFFECTIVE CONFINING STRESS,	500
HEAD PRESSURE, kPa	100	DURATION, min	18000
BACK PRESSURE, kPa	50	HYDRAULIC GRADIENT, i	109

SPECIMEN PROPERTIES AND DIMENSIONS (FINAL)

SAMPLE HEIGHT, cm	4.67	UNIT WEIGHT, kN/m^3	0.00
SAMPLE DIAMETER, cm	3.62	DRY UNIT WEIGHT, kN/m^3	0.00
SAMPLE AREA, cm^2	10.29	SPECIFIC GRAVITY, assumed	2.70
SAMPLE VOLUME, cm^3	48.09	VOLUME OF SOLIDS, cm^3	0.00
TOTAL MASS, g		VOLUME OF VOIDS, cm^3	48.09
DRY MASS, g		VOID RATIO	#DIV/0!
WATER CONTENT, %	#DIV/0!		

TEST RESULTS

ELAPSED TIME TO STEADY STATE FLOW (min)	00
DURATION OF STEADY STATE FLOW (min)	18000
INFLOW VOLUME UNDER STEADY STATE FLOW (cm^3)	0.9
OUTFLOW VOLUME UNDER STEADY STATE FLOW (cm^3)	0.5
HYDRAULIC CONDUCTIVITY (INFLOW) (cm/s)	7.26E-10
HYDRAULIC CONDUCTIVITY (OUTFLOW) (cm/s)	4.13E-10
HYDRAULIC CONDUCTIVITY, K, cm/s	**5.69E-10**

NOTES:

MIXING FLUID	
PERMEANT FLUID	tap water

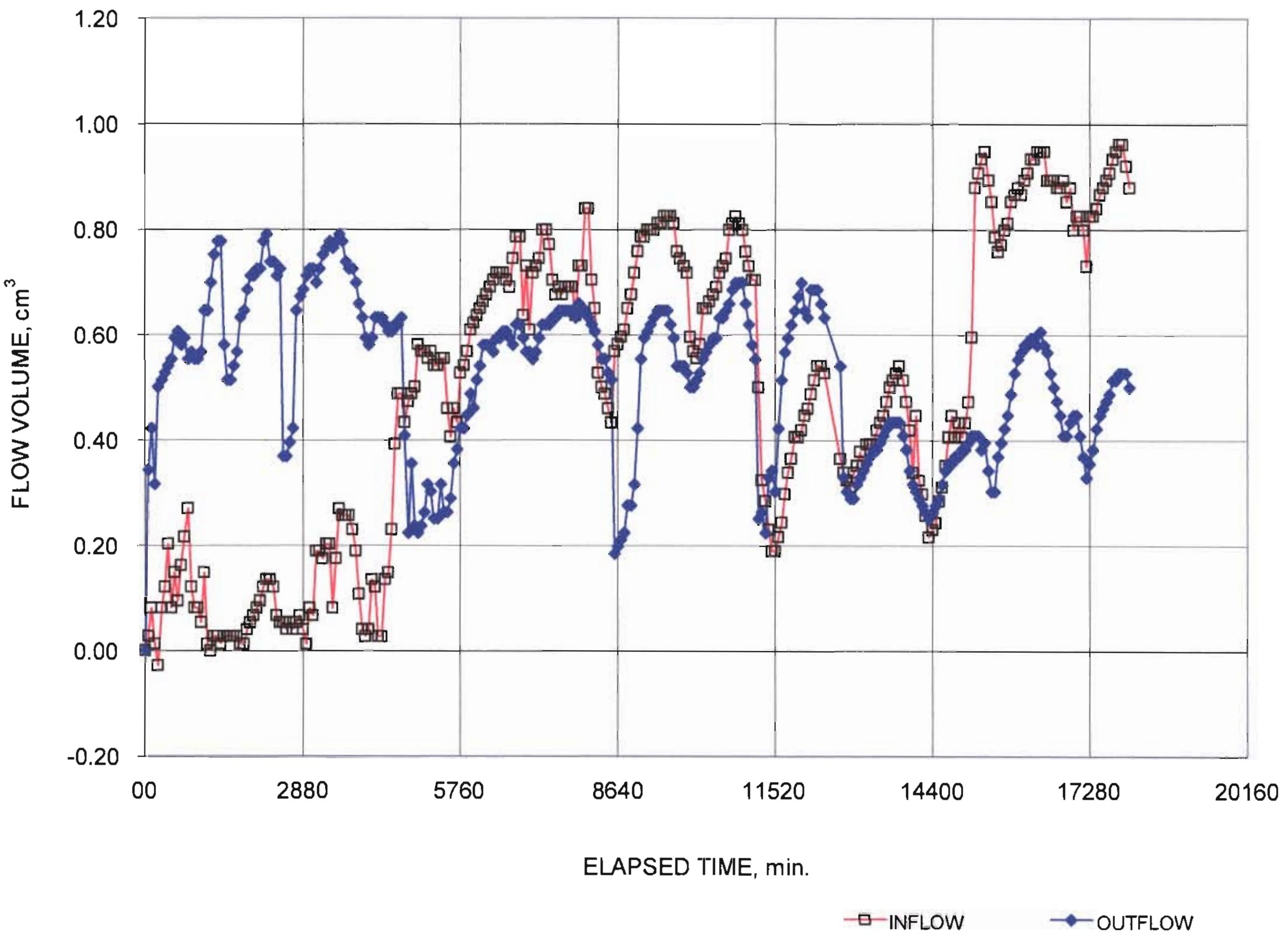
Project No. 06-1116-004
Prepared By: LFG

Golder Associates

Checked By: MM

HYDRAULIC CONDUCTIVITY TEST

Drill Hole 03-05   Sample Depth 139.5-139.595m



Appendix E

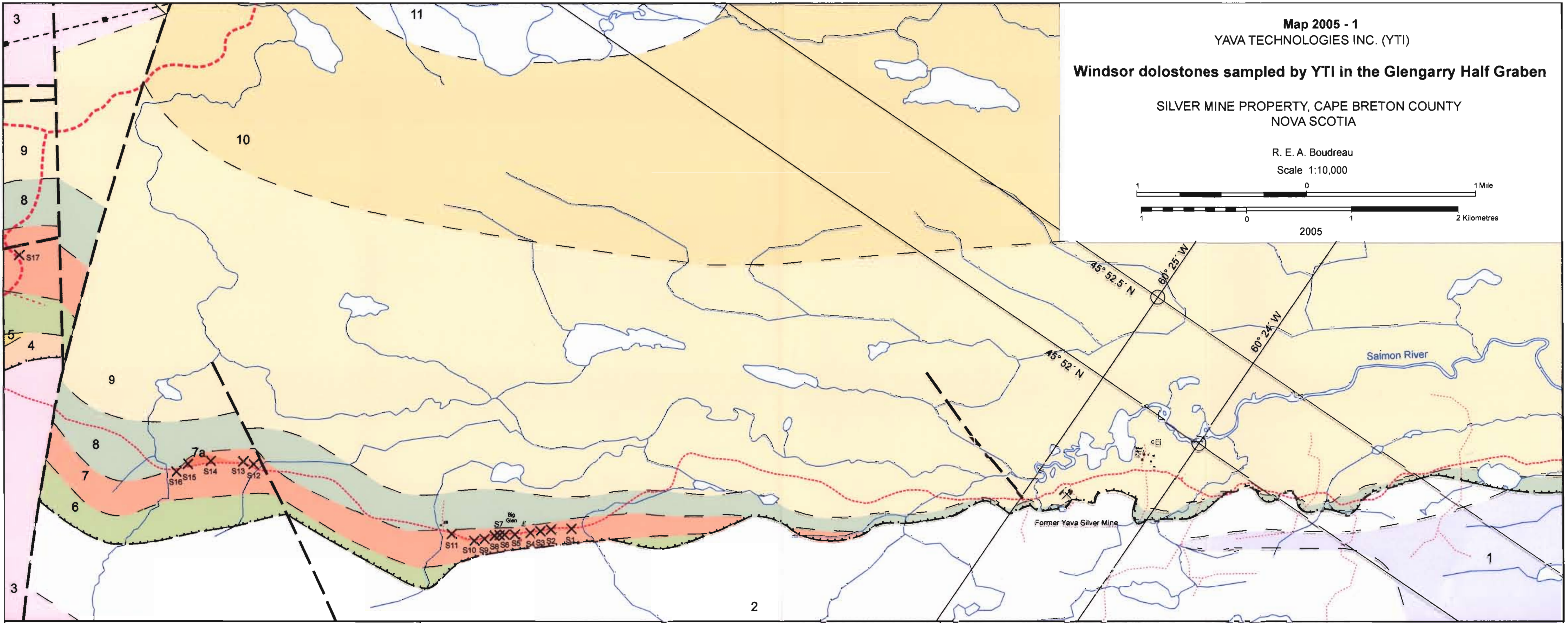
Appendix E

To

GEOLOGIC AND HYDROGEOLOGIC INVESTIGATIONS
AT THE
SILVER MINE PROPERTY
CAPE BRETON COUNTY, NOVA SCOTIA

For the year ending 30 April 2006

MAP SHOWING SAMPLE LOCATIONS AND TABLES OF ASSAY VALUES FROM WINDSOR FORMATION CARBONATE EXPLORATION DURING OCTOBER 2005



Map 2005 - 1

YAVA TECHNOLOGIES INC. (YTI)

Windsor dolostones sampled by YTI in the Glengarry Half Graben

SILVER MINE PROPERTY, CAPE BRETON COUNTY
NOVA SCOTIA

R. E. A. Boudreau

Scale 1:10,000

2005

LEGEND

PENNSYLVANIAN	11	**GLENGARRY VALLEY FORMATION:** gray-green/red sandstone/siltstone/shale.
	10	**BIG BARREN FORMATION:** red conglomerate/sandstone and gray-green mudstone/sandstone.
	9	**SILVER MINE FORMATION:** gray-green/gray sandstone with coal; gray-green/gray mudstone.
MISSISSIPPIAN	8	**MacKEIGAN LAKE FORMATION:** gray/red shale, siltstone, gypsum/anhydrite.
	7 7a	**UIST FORMATION:** Red siltstone, minor conglomerate intercalated with limestone/dolostone; minor gypsum/anhydrite; dolostone (7a).
	6	**LOCH LOMOND FORMATION:** gypsum, anhydrite and limestone/dolostone; red siltstone and sandstone/conglomerates.
	5	**ENON FORMATION:** gypsum, anhydrite and limestone/dolostone; red siltstone and sandstone/conglomerates.
	4	**UNDIVIDED BASAL CONGLOMERATE**
DEVONIAN	3	**LOCH LOMOND, IRISH COVE and HUNTINGTON MOUNTAIN PLUTONS:** Granitoid rocks
Pre-SILURIAN	2 2a	**SALMON RIVER RHYOLITE PORPHYRY:** Granitoid rocks; rhyolite intruding coarse grained sandstone/conglomerate (2a).
	1	**HADRYNIAN to ORDOVICIAN ROCKS:** Volcanic and sedimantary rocks, variably deformed and metamorphosed.

SYMBOLS

Dolostone outcrop	✕ S9
Geologic boundary (assumed)	— — — — — —
Unconformity, nonconformity (approximate)	┴┴┴ ┴┴┴ ┴┴┴
Fault (approximate)	▬ ▬ ▬ ▬ ▬ ▬
House	▪
Mine	⚒
River, Lake	⁄ ⬠
Road	········

Modified from Boehner and Prime, 1985, Loch Lomond Basin and Glengarry Half-Graben, Nova Scotia Department of Mines and Energy Map 85-2

Mineral Assay Values

Sample	Assays		(Detection LImit)	Sample	Assays		(Detection LImit)	Sample	Assays		(Detection LImit)
S1	Pb	>10,000	(2 ppm)	S7	Pb	5760	(2 ppm)	S13	Pb	751	(2 ppm)
	Mn	3.92	(0.01%)		Mn	2.75	(0.01%)		Mn	0.59	(0.01%)
	Ca	29.9	(0.01%)		Ca	32.2	(0.01%)		Ca	35.6	(0.01%)
S2	Pb	7130	(2 ppm)	S8	Pb	629	(2 ppm)	S14	Pb	99	(2 ppm)
	Mn	3.4	(0.01%)		Mn	1.79	(0.01%)		Mn	0.63	(0.01%)
	Ca	32.3	(0.01%)		Ca	29.6	(0.01%)		Ca	33	(0.01%)
S3	Pb	536	(2 ppm)	S9	Pb	5060	(2 ppm)	S15	Pb	2030	(2 ppm)
	Mn	2.19	(0.01%)		Mn	2.19	(0.01%)		Mn	1.35	(0.01%)
	Ca	38	(0.01%)		Ca	35.4	(0.01%)		Ca	32.3	(0.01%)
S4	Pb	708	(2 ppm)	S10	Pb	5710	(2 ppm)	S16	Pb	230	(2 ppm)
	Mn	1.52	(0.01%)		Mn	1.36	(0.01%)		Mn	0.62	(0.01%)
	Ca	50.1	(0.01%)		Ca	31	(0.01%)		Ca	32.2	(0.01%)
S5	Pb	854	(2 ppm)	S11	Pb	203	(2 ppm)	S17	Pb	187	(2 ppm)
	Mn	2.56	(0.01%)		Mn	1.25	(0.01%)		Mn	0.86	(0.01%)
	Ca	31	(0.01%)		Ca	31	(0.01%)		Ca	43.9	(0.01%)
S6	Pb	709	(2 ppm)	S12	Pb	479	(2 ppm)				
	Mn	2.38	(0.01%)		Mn	1.08	(0.01%)				
	Ca	30.1	(0.01%)		Ca	30.4	(0.01%)				

Appendix E
Windsor mudstone sample assays
25-Oct-05

ANALYTE	Ag	Al2O3	As	Au	B	Ba	Be	Bi	Br	C	CaO	Cd	Ce
METHOD	MERICP	MERXRF	MERAA	FAI313	MERICP	MERMS1	MERICP	MERMS1	MERNA	MERLEC	MERXRF	MERMS1	MERMS1
DETECTION	2	0.01	0.5	1	10	0.5	0.5	0.5	1	0.01	0.01	0.2	0.1
UNITS	PPM	%	PPM	PPB	PPM	PPM	PPM	PPM	PPM	%	%	PPM	PPM
65785	5	1.05	3.1	3	<10	26.6	<0.5	<0.5	3	12.5	29.9	15	3
65786	4	0.97	3.8	2	20	34.8	<0.5	<0.5	4	12.3	31.5	5.6	3.2
65787	3	0.87	2.2	1	<10	13.3	<0.5	<0.5	5	12.9	32.3	<0.2	1.6
65788	2	0.51	2	5	<10	183	<0.5	<0.5	5	12.6	38	1.4	3
65789	2	0.36	3.4	1	<10	79.1	<0.5	<0.5	3	12.6	50.1	2.3	3.6
65790	3	0.64	2.8	1	30	36.1	<0.5	<0.5	26	13	31	<0.2	3.5
65791	3	0.68	3.7	1	30	8.6	<0.5	<0.5	15	12.7	30.1	<0.2	2.8
65792	3	0.96	2.3	1	20	53.1	<0.5	<0.5	6	12.5	32.2	0.3	7.2
65793	3	0.91	1.7	2	20	32.3	<0.5	<0.5	28	12.6	29.6	0.9	10
65794	3	0.77	2.6	1	<10	825	<0.5	<0.5	5	12.4	35.4	0.3	5.1
65795	2	0.34	1.9	1	<10	15.8	<0.5	<0.5	10	13.1	31	3.5	1.5
65796	<2	0.29	1.4	1	<10	8.6	<0.5	<0.5	8	12.9	31	0.5	1.2
65797	2	0.54	2.2	1	<10	33.9	<0.5	<0.5	22	13	30.4	3.4	4.3
65798	<2	0.23	2.5	1	<10	10.8	<0.5	<0.5	5	13.1	35.6	5.9	2.1
65799	2	0.38	2.6	1	<10	11.9	<0.5	<0.5	5	13.1	33	3.1	2.3
65800	2	0.41	1.4	1	<10	7.6	<0.5	<0.5	4	13.1	32.3	5.3	1.5
65851	<2	0.84	1.8	1	<10	19.6	<0.5	<0.5	4	12.6	32.2	20.9	6.2
65852	<2	0.62	3.6	3	<10	20.6	<0.5	<0.5	3	12.2	43.9	2.2	6.8
65853	<2	8.85	1.9	2	<10	5230	0.7	<0.5	<1	0.09	0.3	<0.2	4.1
DUP-65785	5	1.06	3.5	1	<10	25.5	<0.5	<0.5	3	12.5	29.8	15.1	2.9
DUP-65797	2	0.54	2.7	1	<10	36.2	<0.5	<0.5	22	13.2	30.4	3.6	4.3

Appendix E
Windsor mudstone sample assays
25-Oct-05

ANALYTE	Cl	Co	Cr	Cr2O3	Cs	Cu	Dy	Er	Eu	Fe2O3	Ga	Gd	Ge
METHOD	MERCL	MERICP	MERMS1	MERXRF	MERMS1	MERICP	MERMS1	MERMS1	MERMS1	MERXRF	MERMS1	MERMS1	MERMS1
DETECTION	50	1	2	0.01	0.5	0.5	0.05	0.05	0.05	0.01	1	0.05	1
UNITS	PPM	PPM	PPM	%	PPM	PPM	PPM	PPM	PPM	%	PPM	PPM	PPM
65785	430	<1	8	<0.01	<0.5	1.8	0.24	0.13	<0.05	0.73	1	0.31	<1
65786	490	<1	6	<0.01	<0.5	1.3	0.23	0.15	<0.05	0.42	1	0.35	<1
65787	630	<1	6	<0.01	<0.5	0.9	0.13	0.07	<0.05	0.11	<1	0.13	<1
65788	500	<1	7	<0.01	<0.5	1.1	0.17	0.11	0.06	<0.01	<1	0.24	<1
65789	330	<1	5	<0.01	<0.5	2	0.23	0.12	0.07	<0.01	<1	0.34	<1
65790	1390	<1	5	<0.01	<0.5	1.2	0.25	0.14	0.09	0.61	<1	0.36	<1
65791	960	<1	7	<0.01	<0.5	1.6	0.21	0.12	0.06	0.64	<1	0.25	<1
65792	590	<1	7	<0.01	<0.5	1.3	0.6	0.27	0.17	0.48	<1	0.66	<1
65793	1450	<1	9	<0.01	<0.5	1.4	0.76	0.37	0.25	0.66	1	0.96	<1
65794	540	<1	8	<0.01	<0.5	1.7	0.39	0.18	0.15	0.57	<1	0.55	<1
65795	910	<1	7	<0.01	<0.5	1.9	0.14	0.07	<0.05	0.8	<1	0.19	<1
65796	690	<1	5	<0.01	<0.5	1.1	0.13	0.07	<0.05	0.69	<1	0.16	<1
65797	1130	<1	6	<0.01	<0.5	2.1	0.3	0.15	0.06	0.28	<1	0.35	<1
65798	500	<1	6	<0.01	<0.5	1.7	0.19	0.11	0.06	0.25	<1	0.23	<1
65799	610	4	6	<0.01	<0.5	1.5	0.25	0.13	0.08	0.42	<1	0.26	<1
65800	520	<1	6	<0.01	<0.5	1.6	0.18	0.1	<0.05	0.18	<1	0.16	<1
65851	590	<1	9	<0.01	<0.5	2.4	0.52	0.31	0.13	0.58	1	0.69	<1
65852	370	<1	7	<0.01	<0.5	2.9	0.54	0.25	0.14	0.59	<1	0.64	<1
65853	110	6	87	0.01	1.5	2.9	1.09	0.77	0.36	1.86	8	0.8	1
DUP-65785	440	<1	8	<0.01	<0.5	1.6	0.23	0.13	<0.05	0.73	1	0.31	<1
DUP-65797	1150	<1	8	<0.01	<0.5	1.9	0.35	0.16	0.09	0.28	<1	0.32	<1

Appendix E
Windsor mudstone sample assays
25-Oct-05

ANALYTE	Hf	Hg	Ho	In	K2O	La	Li	LOI	Lu	MgO	MnO	Mo	Na2O
METHOD	**MERMS1**	**MERHG**	**MERMS1**	**MERMS1**	**MERXRF**	**MERMS1**	**MERICP**	**MERXRF**	**MERMS1**	**MERXRF**	**MERXRF**	**MERICP**	**MERXRF**
DETECTION	**0.2**	**5**	**0.05**	**0.5**	**0.01**	**0.1**	**1**	**0.01**	**0.05**	**0.03**	**0.01**	**1**	**0.02**
UNITS	**PPM**	**PPB**	**PPM**	**PPM**	**%**	**PPM**	**PPM**	**%**	**PPM**	**%**	**%**	**PPM**	**%**
65785	<0.2	<5	<0.05	<0.5	0.05	1.6	3	43.9	<0.05	17.7	3.92	<1	<0.02
65786	<0.2	<5	0.05	<0.5	0.04	1.5	2	44.6	<0.05	18.3	3.74	1	<0.02
65787	<0.2	<5	<0.05	<0.5	0.01	0.8	2	45.5	<0.05	17.7	3.4	<1	0.02
65788	<0.2	<5	<0.05	<0.5	0.01	1.8	1	44.7	<0.05	14.3	2.19	<1	0.02
65789	<0.2	<5	0.05	<0.5	0.02	2.2	1	43.4	<0.05	3.87	1.52	1	0.05
65790	<0.2	<5	<0.05	<0.5	0.02	1.7	3	46.1	<0.05	19	2.56	<1	0.02
65791	<0.2	<5	<0.05	<0.5	0.03	1.5	3	45.4	0.05	20.7	2.38	<1	<0.02
65792	<0.2	<5	0.12	<0.5	0.07	3.3	4	44.9	<0.05	17.8	2.75	1	0.03
65793	0.2	<5	0.14	<0.5	0.07	4.2	4	45.4	<0.05	20.3	1.79	2	<0.02
65794	<0.2	<5	0.07	<0.5	0.05	2.4	2	44.2	<0.05	14.7	2.19	<1	0.04
65795	<0.2	8	<0.05	<0.5	<0.01	0.8	3	46.2	<0.05	20.3	1.36	1	<0.02
65796	<0.2	<5	<0.05	<0.5	0.01	0.7	1	46.5	<0.05	20.2	1.25	2	<0.02
65797	0.2	<5	0.07	<0.5	0.04	3.3	3	45.8	<0.05	20.2	1.08	2	<0.02
65798	<0.2	<5	<0.05	<0.5	0.02	1.4	2	45.9	<0.05	16.8	0.59	2	0.02
65799	0.2	<5	<0.05	<0.5	0.05	1.1	8	45.7	<0.05	18.9	0.63	3	<0.02
65800	<0.2	<5	<0.05	<0.5	0.03	0.8	2	45.8	<0.05	18.7	1.35	2	<0.02
65851	0.3	<5	0.11	<0.5	0.11	3.1	7	44.3	0.08	18.5	0.62	3	<0.02
65852	0.2	<5	0.09	<0.5	0.06	4	4	43.4	0.06	8.36	0.86	4	0.05
65853	4.4	<5	0.24	<0.5	2.49	2	17	1.25	0.17	0.71	0.03	<1	2.4
DUP-65785	<0.2	<5	<0.05	<0.5	0.05	1.3	3	44	<0.05	17.6	3.91	1	<0.02
DUP-65797	0.2	<5	0.07	<0.5	0.04	3.1	3	45.8	<0.05	20.1	1.08	2	<0.02

Appendix E
Windsor mudstone sample assays
25-Oct-05

ANALYTE	**Nb**	**Nd**	**Ni**	**P2O5**	Pb	**Pd**	**Pr**	**Pt**	**Rb**	S	**Sb**	**Sc**	**Se**
METHOD	**MERMS1**	**MERMS1**	**MERICP**	**MERXRF**	MERICP	**FAI313**	**MERMS1**	**FAI313**	**MERMS1**	MERLEC	**MERMS1**	**MERICP**	**MERSE**
DETECTION	**0.1**	**0.1**	**1**	**0.01**	2	**1**	**0.05**	**10**	**0.2**	0.01	**0.5**	**0.5**	**0.5**
UNITS	**PPM**	**PPM**	**PPM**	**%**	PPM	**PPB**	**PPM**	**PPB**	**PPM**	%	**PPM**	**PPM**	**PPM**
65785	0.2	1.2	4	<0.01	>10000	2	0.26	<10	1.3	0.37	<0.5	<0.5	<0.5
65786	0.3	1.3	4	<0.01	7130	2	0.27	<10	1.1	0.1	<0.5	<0.5	<0.5
65787	0.2	0.7	3	<0.01	536	2	0.16	<10	0.8	<0.01	<0.5	<0.5	<0.5
65788	0.6	1.2	5	<0.01	708	2	0.28	<10	0.7	<0.01	<0.5	<0.5	<0.5
65789	0.4	1.6	5	<0.01	854	2	0.34	<10	0.8	0.03	<0.5	<0.5	<0.5
65790	0.3	1.6	5	<0.01	709	2	0.36	<10	0.8	<0.01	<0.5	<0.5	<0.5
65791	0.2	1.3	5	<0.01	5760	2	0.29	<10	1.2	0.09	<0.5	<0.5	<0.5
65792	0.3	3.3	5	<0.01	629	2	0.76	<10	2.7	0.02	<0.5	<0.5	<0.5
65793	1.1	5.2	4	<0.01	5060	2	1.13	<10	4.1	0.03	<0.5	<0.5	<0.5
65794	0.4	2.6	6	<0.01	5710	2	0.59	<10	2.7	0.09	<0.5	<0.5	<0.5
65795	0.5	0.8	5	<0.01	203	2	0.18	<10	0.5	<0.01	2.2	<0.5	<0.5
65796	0.2	0.6	4	<0.01	479	2	0.13	<10	0.4	<0.01	<0.5	<0.5	<0.5
65797	0.5	1.8	5	<0.01	751	2	0.42	<10	2.3	0.03	<0.5	<0.5	<0.5
65798	1	1.1	4	<0.01	99	2	0.25	<10	0.9	0.03	<0.5	<0.5	<0.5
65799	0.5	1.2	11	<0.01	2030	2	0.28	<10	1.8	0.05	<0.5	<0.5	<0.5
65800	0.4	0.8	5	0.01	230	2	0.18	<10	0.9	0.03	<0.5	<0.5	<0.5
65851	1.4	3.4	6	<0.01	187	2	0.77	<10	4.9	0.21	<0.5	0.5	<0.5
65852	0.6	3	7	<0.01	192	2	0.73	<10	3.1	0.05	<0.5	<0.5	<0.5
65853	7.9	2.4	15	0.01	27	2	0.49	<10	69.2	0.09	<0.5	3.4	<0.5
DUP-65785	0.3	1.2	5	<0.01	>10000	2	0.25	<10	1.4	0.36	<0.5	<0.5	<0.5
DUP-65797	0.6	1.8	6	<0.01	739	2	0.43	<10	2.5	0.02	<0.5	<0.5	<0.5

Appendix E
Windsor mudstone sample assays
25-Oct-05

ANALYTE	**SiO2**	**Sm**	**Sn**	Sr	**Sr**	**SUM**	**Ta**	**Tb**	**Te**	**Th**	**TiO2**	**Tl**	**Tm**
METHOD	**MERXRF**	**MERMS1**	**MERMS1**	MERMS1	**MERICP**	**MERXRF**	**MERMS1**	**MERMS1**	**MERSE**	**MERMS1**	**MERXRF**	**MERMS1**	**MERMS1**
DETECTION	**0.01**	**0.1**	**2**	0.1	**0.5**	**0.01**	**0.5**	**0.05**	**0.5**	**0.1**	**0.01**	**0.1**	**0.05**
UNITS	**%**	**PPM**	**PPM**	PPM	**PPM**	**%**	**PPM**	**PPM**	**PPM**	**PPM**	**%**	**PPM**	**PPM**
65785	<0.01	0.2	<2	67.1	64	96.9	<0.5	<0.05	<0.5	0.2	0.01	0.2	<0.05
65786	0.09	0.3	<2	70.3	59.8	99.7	<0.5	0.05	<0.5	0.2	0.02	0.2	<0.05
65787	0.29	0.1	<2	64.3	61.4	100.2	<0.5	<0.05	<0.5	0.1	<0.01	<0.1	<0.05
65788	0.65	0.3	<2	89.4	81	100.4	<0.5	<0.05	<0.5	0.2	<0.01	<0.1	<0.05
65789	0.67	0.3	<2	312	271	99.7	<0.5	<0.05	<0.5	0.2	<0.01	<0.1	<0.05
65790	0.21	0.3	<2	70.5	63	100.2	<0.5	<0.05	<0.5	0.1	<0.01	<0.1	<0.05
65791	0.32	0.3	<2	67.9	63.5	100.3	<0.5	<0.05	<0.5	0.2	0.01	<0.1	<0.05
65792	1.01	0.7	<2	84.2	71.7	100.2	<0.5	0.11	<0.5	0.4	0.03	0.2	<0.05
65793	1.47	1	<2	52.7	49.5	100.2	<0.5	0.16	<0.5	0.8	0.03	0.1	<0.05
65794	0.79	0.6	<2	134	122	98.7	<0.5	0.08	<0.5	0.5	0.02	<0.1	<0.05
65795	0.34	0.2	<2	67.5	58.9	100.4	<0.5	<0.05	<0.5	0.2	<0.01	0.2	<0.05
65796	0.2	0.1	<2	59.1	55.6	100.1	<0.5	<0.05	<0.5	0.1	<0.01	<0.1	<0.05
65797	1.53	0.3	<2	49.3	43.7	99.8	<0.5	0.05	<0.5	0.3	0.03	0.1	<0.05
65798	0.73	0.2	<2	106	94.4	100.1	<0.5	<0.05	<0.5	0.3	0.01	<0.1	<0.05
65799	1.2	0.3	<2	95.1	84	100.2	<0.5	<0.05	<0.5	0.4	0.03	0.2	<0.05
65800	1.19	0.2	<2	80.5	68.5	100	<0.5	<0.05	<0.5	0.1	<0.01	0.1	<0.05
65851	2.93	0.7	<2	109	97.9	100.1	<0.5	0.11	<0.5	0.5	0.04	0.2	<0.05
65852	2.05	0.6	<2	181	167	99.9	<0.5	0.1	<0.5	0.4	0.03	0.1	<0.05
65853	80.8	0.7	<2	171	150	99.2	0.7	0.15	<0.5	6.1	0.45	0.4	0.11
DUP-65785	<0.01	0.3	<2	72.4	64.9	96.9	<0.5	<0.05	<0.5	0.2	0.01	0.2	<0.05
DUP-65797	1.52	0.3	<2	51.4	45.1	99.8	<0.5	0.06	<0.5	0.3	0.03	0.1	<0.05

Appendix E
Windsor mudstone sample assays
25-Oct-05

ANALYTE	U	V	W	Y	Yb	Zn	Zr
METHOD	**MERMS1**	**MERICP**	**MERMS1**	**MERMS1**	**MERMS1**	**MERICP**	**MERMS1**
DETECTION	**0.05**	**2**	**1**	**0.5**	**0.1**	**0.5**	**0.5**
UNITS	**PPM**	**PPM**	**PPM**	**PPM**	**PPM**	**PPM**	**PPM**
65785	0.43	<2	<1	1.8	<0.1	762	1.3
65786	0.53	2	<1	2.2	0.1	413	<0.5
65787	0.18	<2	<1	0.8	<0.1	16.2	<0.5
65788	0.84	<2	<1	1.3	<0.1	116	<0.5
65789	1.85	<2	<1	1.6	<0.1	319	<0.5
65790	0.46	<2	<1	1.9	0.1	14.1	<0.5
65791	0.44	<2	<1	1.5	<0.1	8.4	0.6
65792	0.36	<2	<1	3.2	0.2	92	1.5
65793	0.46	2	<1	4.1	0.3	83.2	4.3
65794	0.51	<2	<1	2	0.1	23.8	1.9
65795	1.07	<2	<1	1	<0.1	213	<0.5
65796	0.73	2	<1	0.9	<0.1	39.4	<0.5
65797	1.01	6	<1	1.8	0.1	101	6
65798	1.32	4	<1	1.1	0.1	186	1.6
65799	2	4	<1	1.3	0.1	154	7.3
65800	1.51	4	<1	1.1	<0.1	261	1
65851	1.86	7	<1	3	0.2	1080	8.6
55852	1.99	8	<1	3.1	0.2	159	6
65853	0.88	24	<1	6	0.9	22.1	155
DUP-65785	0.45	<2	<1	1.9	<0.1	756	0.7
DUP-65797	1.06	6	<1	1.9	0.2	99.4	6

AR2006-091

YAVA TECHNOLOGIES INC.

WORK PERFORMED ON IN-SITU LEACH MINING DEMONSTRATION PLANT TO RECOVER LEAD, ZINC, MANGANESE AND CALCIUM AS LEAD NITRATE/LEAD OXIDE/LEAD DIOXIDE, ZINC OXIDE, MANGANESE DIOXIDE AND PRECIPITATED CALCIUM CARBONATE FROM THE YAVA LEAD DEPOSIT, CAPE BRETON COUNTY, NOVA SCOTIA FOR THE YEAR ENDING APRIL 30, 2006

Prepared By:

Madhav P. Dahal Ph.D

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DUPLICATE AVAILABLE

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G. LIST OF REPORTS OF INVESTIGATIONS PERFORMED BY ELECTROSYNTHESIS INC.

1. PROGRESS REPORT NO 1……………………………………………………………

SUMMARY

Chemical Investigations

Yava's ongoing demonstration plant operation has performed extensive work on overall optimization of Yava's advanced chemical processing technology to recover lead, zinc, manganese and calcium as lead nitrate/lead oxide/lead dioxide, zinc oxide, manganese dioxide and precipitated calcium carbonate. The work performed during this period involved sand filtration, enhancement of loading and stripping kinetics and regeneration of ion exchange resins, handling of gelatinous precipitates, iron removal, heavy and rare earth metal sulfides precipitation and pilot-scale production of high purity commercial lead nitrate sample. The results of the investigations are presented in reports numbered 1 to 12 by Vizon SciTec Inc. attached hereto.

Electrochemical Investigations

Yava's closed loop in-situ leach mining operation regenerates most of the chemicals used in the leaching process by employing bipolar membrane salt-splitting technology. *Electrosynthesis Inc.* successfully performed electrochemical salt splitting tests with synthetic sodium nitrate ($NaNO_3$), sodium sulfate (Na_2SO_4) and sodium acetate (CH_3COONa) to produce sodium hydroxide/nitric acid ($NaOH/HNO_3$), sodium hydroxide/sulfuric acid (($NaOH/H_2SO_4$) and acetic acid/sodium hydroxide ($CH_3COOH/NaOH$). Relationships between the solution strengths, cell voltage, current efficiency and water transport were thoroughly examined. The results demonstrated that as product concentrations decline, current efficiency, cell voltage and water transport increase. Longer term stability of the system (over a period of 25 hours for the sodium nitrate system and 51 hours for the sodium sulfate system) was also investigated. There was some accumulation of a brownish colored precipitate in the flow channels of the feed compartments during the sodium sulfate experiments and this resulted in reduced flows and slightly higher voltage. When the flows were re-adjusted, the cell voltage returned to normal levels. Addition of a filter to the feed loop removed any insolubles in the solution and thus avoided this problem.

An extensive mathematical calculation was performed to determine the optimal operational parameters required for the salt splitting technology using reagent grade salt solutions. This will enable Yava to compare the effectiveness of bipolar membrane salt splitting technology efficiency between Yava's feed solution and theoretical reagent grade solution. This difference in performance directly equates to the effect of contaminants, maintenance and replacement frequency of electrodialysis units for Yava's operation. The results of the investigations by Electrosynthesis Inc are presented in Appendix B.

Appendix F

APPENDIX F

LIST OF REPORTS OF INVESTIGATIONS PERFORMED BY VIZON SCITEC INC.

Vizon Scitec Inc.
3650 Wesbrook Mall
Vancouver, BC
V6S 2L2

Project No. 4040751

Progress Report No 1

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC.

DURING MAY 2005

Prepared by

Norman Chow, Kenneth Seto (Vizon Scitec Inc)

and

Clive Brereton ,Alan Keylock (Noram Engineering and Constructors Ltd)

INTRODUCTION

Yava Technologies Inc ("Yava") has developed an innovative process involving in situ leach mining (ISLM) to extract lead, zinc, manganese and calcium from the Yava lead deposit on its mining property in Nova Scotia. The ISLM method was field tested on the deposit to confirm the character of the resulting leachate. Separation of the desired metals from the leachate to produce high purity metal ion solutions via solvent extraction and ion exchange was examined by SGS Lakefield Research Ltd (Lakefield). Yava is also in discussions with specialty producers of custom recovery systems offering high selectivity for its target metals that may be advantageously employed. In addition a number of processes required for the extraction from solution of some of the target metals as valuable chemicals have undergone successful testing at bench scale at Vizon Scitec Inc. (formerly BC Research Inc). Based on the work to date, a preliminary process flow diagram (PFD) for same has been produced (Appendix A).

In order to verify, and if necessary find appropriate replacements for the steps shown in the preliminary PFD significant work is required. The first part of this work is to test each of the steps sequentially using authentic leach solutions in order to establish partitioning of major components including contaminants. The steps should be performed at a scale large enough to generate sufficient intermediates and products to enable evaluation and to select appropriate commercial separation and reaction equipment. This pilot program will be managed by Noram Engineering and Constructors Ltd (Noram) with the necessary test work subcontracted to Vizon Scitec and other parties.

SUMMARY

1. Pilot plant preparation was completed entailing procurement and assembly of equipment, preparation of the spill containment berm, receipt of eight 45 gallon drums of synthetic leachate prepared by Lakefield and conducting an environmental and safety review.

2. The initial step of the process entailing sand filtration of the incoming leachate was pilot tested and shown to be effective in removing suspended solids to a relatively low amount without chemically altering the leachate. Sand filtration is required to remove insoluble particles that could otherwise plug the ion exchange resin bed proposed for the Yava process.

3. A trial of the proposed HAc + $H2O2$ ISLM leachant was conducted on Yava ore by varying the molar ratio of $H2O2$ to suphide minerals of the ore. Optimum Pb leaching under the conditions tested occurred at an $H2O2$ to suphide minerals molar ratio of 2:1. $O2$ gas was also tested as an alternative oxidant to $H2O2$. $O2$ under the conditions tested was significantly inferior to $H2O2$.

RESULTS AND DISCUSSION

Pilot Plant Preparation

Equipment and supplies were purchased in connection with the PFD provided in Appendix A and as outlined in Appendix B to the Pilot Plant Estimate submitted by Noram. The pilot plant space located at Vizon was prepared in order that its operation would conform to local environmental regulations set out by the Greater Vancouver Regional District (GVRD) and Vizon's internal Health and Safety Manual (which requires conformance with BC's WCB standards). Vizon's 131 page Health and Safety Manual is available on request.

A spill containment berm was assembled at the Vizon facility with an estimated capacity 1.5 times the volume of solution on hand for the Yava pilot plant. Solution storage tanks, stirred tank reactors, ion exchange columns and other relevant equipment were assembled in the pilot plant. Eight 45 gallon drums of Yava leachate prepared by Lakefield were received and placed in the pilot plant. A photograph of the plant is shown in Figure 1.

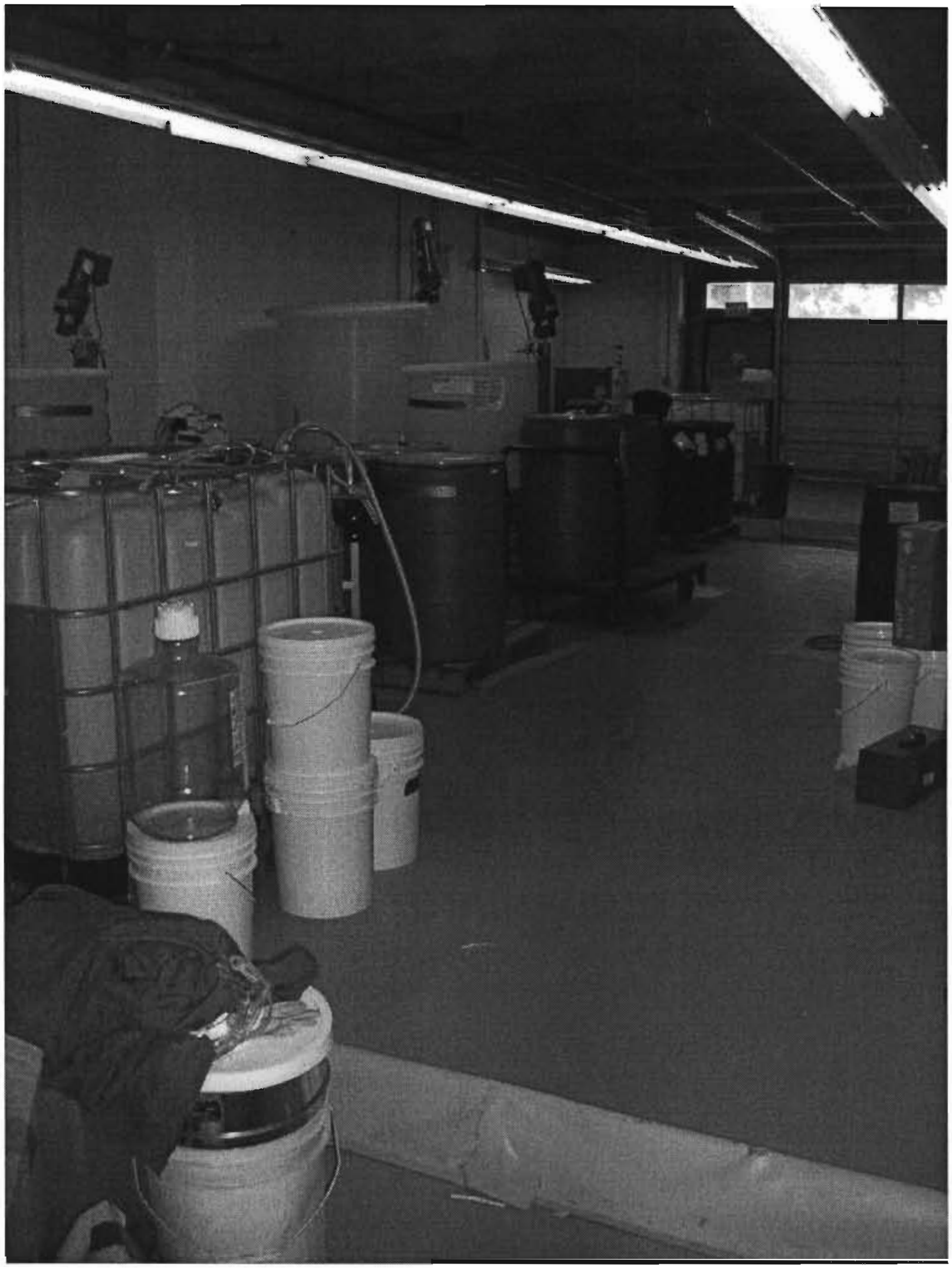


Figure 1. Yava Pilot Plant

Sand Filter Testing

Prior to ion exchange treatment, incoming leachate from ISLM must undergo filtration to remove fine particles of clay and other insoluble material picked up in the course of leachant travel through the ore body that can plug the ion exchange resin bed to prevent smooth ion exchange operation. Sand filtration was identified as one of the lowest cost filtration methods to treat the large volumes of solution that must be handled in the circumstances.

The basic operation of a sand filter entails:

1. Service flow (solution to be filtered) enters the top of the filter, flows through the media (sand) bed and discharges from the bottom. The interstices of the media bed will trap solid particles of equivalent minimum size present in the solution.

2. The pressures at the inlet and the outlet of the sand filter are monitored. A certain pressure difference between them indicates that the bed is plugged requiring .a fresh water backwash to clear the filter. This is usually happens automatically. Backwash discharge is either sent to disposal or recycled (eg elemental sulphur in the backwash could be retained for later use in the process to generate H2S to precipitate dissolved metals as sulphides.).

3. Following backwash, a rinse in the opposite direction is performed to settle and compact the bed. Its discharge goes to disposal or recycling as in (2).

4. Service flow proceeds and the process continually repeats itself.

The cross sectional area of the sand filter is generally sized according to the expected range of flow rate. Service flow is generally 3 to 5 US Gal per min per square feet of cross sectional area of the bed. Backwash flow is generally 12 to 25 US Gal per min per square feet of cross sectional area of the bed and is only conducted intermittently for a short period. The length of the bed is sized according to the frequency of backwashing and the capacity of the bed.

The particle sizes that can be removed by a sand filter depend on the type of media used. Due to the high service flow and small particle size required by the Yava leachate,a "Multi Media" sand was recommend by the supplier. . This is achieved by adding layers of several different media (usually 3 to 5) resulting in higher service flow rates and removal of smaller particle sizes.

The information on sand mixture employed is proprietary to the filter supplier; essentially the sand types used are added according to density and reverse grading. The densest sand with the smallest mesh size is loaded first and the least dense with the largest mesh size is loaded last with intervening media layered accordingly. Consequently larger particles are mainly removed first and successively smaller particles in succeeding layers.

Moreover by decreasing sand density from bottom to top the entire filter bed can readily settle to its original state following backwashing.

The most common media mix from top to bottom consists of: anthracite, filter sand, garnet 30x40, garnet 8x12, gravel 1/8in x 1/16in and gravel 1/4in x 1/8in.

Two pilot scale filtration tests were conducted at Vizon. One with the sand filter only and a second with a sand filter followed by a 10 micron polishing filter, and then a 5 micron polishing filter and then a 1 micron polishing filter.

The “filterability” of the leachate was determined by Total Suspended Solids (TSS) analysis. Solution was passed through a through a 1 micron filter and the solids collected per unit volume of solution (mg/L) were weighed.

The solutions used for filtration tests were obtained from separate as received 45 gallon drums of Yava leachate provided by Lakefield from leaching Yava ore with an HAc + H2O2 solution. The contents of each drum were well mixed prior to sampling and filtering. A significant difference in initial Total Suspended Solids (TSS) from drum to drum was observed by conducting TSS measurements after each sand filtration test was performed. Consequently it is unclear if a single representative TSS value would be found by on-site sampling or if instead a wide range of values would result. In any event the TSS value after filtration will show whether sand filtration alone or followed by polishing filtration is appropriate.

The results of the two filtration tests are set out below:

TEST #1

As received leachate TSS = 462 mg/L
Following sand filtration TSS = 15 mg/L

TEST #2

As received leachate TSS = 8 mg/L
Following sand filtration TSS = 6 mg/L
Following 10 micron polishing filter TSS = 4 mg/L
Following 5 micron polishing filter TSS = 3 mg/L
Following 1 micron polishing filter TSS = 2 mg/L

According to the sand filter supplier, conventional sand filtration removes all material to about 20 to 50 microns. Sand filtration with a Multi Media system has a particle size limit of about 10 microns.

ICP analysis of the leachate and the filtrate was conducted to determine if the type of filter media (i.e. sand) reacts chemically with the Yava leachate. The results (provided in Appendix C) show that the Pb, Zn, Mn and Ca concentrations are similar for the leachate and the filtrate. Consequently this type of filter media had no detrimental effect on the leachate.

ICP analysis (provided in Appendix C) was also conducted on the solids collected from the backwash to determine if elemental sulphur is present (which could be used to biologically produce H2S for use in the Yava process to precipitate heavy and rare earth metals). Only a small amount (0.01 wt%) of elemental sulphur was detected in the backflush solids. Sulphate, most likely as calcium sulphate (that could substitute for elemental sulphur) was also detected with the Al, Fe, plus small amounts of Cr, and trace As, Ba, Co and Cu.

Leaching

A bench scale study was conducted to examine leach efficiency of Yava ore using a lixiviant containing hydrogen peroxide (H2O2) and acetic acid (CH3COOH). The stoichiometric ratio of H2O2 to sulphide bearing minerals (i.e. lead sulphide (PbS) and zinc sulphide (ZnS)) was varied from 0:1 to 4:1. The stoichiometric ratio of CH3COOH to equivalent mole of target metals contained in the ore (i.e. lead (Pb), manganese (Mn), zinc (Zn) and calcium (Ca)) was fixed at 1.5:1 of the molar ratio required to convert these metals to metal acetates (i.e. lead acetate (Pb(CH3COO)2), manganese acetate (Mn(CH3COO)2, zinc acetate (Zn(CH3COO)2) and calcium acetate (Ca(CH3COO)2).

Preliminary tests were also conducted on substituting oxygen (O2) gas for H2O2. In addition, depleted ore residues following leaching were analyzed for secondary products (e.g. elemental sulphur (S^0) and possibly lead sulphate (PbSO4) precipitate) that may have formed during the dissolution of the minerals.

A detailed test procedure was provided by Yava (included as Appendix D to this report).

Yava Ore Sample

An ore sample (identified as #LDRM2) was provided by Yava for this test work. The assay of the ore (also provided by Yava) is summarized in Table 1.

Table 1: Assay of Ore Sample #LDRM2 Provided by Yava

Sample No.	Pb (wt%)	Zn (wt%)	Mn (wt%)	Fe (wt%)	Ca (wt%)	S (wt%)
LDRM2 – Yava	8.35	0.09	0.05	0.83	0.38	1.34

Water Saturation of Dry Ore

The amount of water required to saturate the dry ore was 3.8 wt%. This was determined by mixing 50 g of the as received crushed ore sample (#LDRM2) with 100 mL of de-ionized water in a Pyrex 125 mL Erlenmeyer flask. The flask containing the slurry was placed onto a variable speed orbital shake table and allowed to shake gently for 20 minutes. The slurry was filtered through a vacuum filtration system consisting of a ceramic Buchner funnel and dry filter paper. After filtration, the ceramic Buchner funnel containing the wet filter paper and wet filter cake were weighed and then placed into a low temperature oven heated to 55°C. The ceramic Buchner funnel containing the dry filter paper and dry filter cake were weighed the next day. To correct for moisture absorbed by the filter paper, the weight of a representative dry filter paper and wet filter paper that had been subjected to vacuum in a Buchner funnel were measured. The amount of water required to saturate the dry ore was determined by the difference in weight between the wet filter cake and the dry filter cake. The experimental data of the water saturation tests are summarized in Table 2.

Table 2: Water Saturation Results of the Yava Dry Ore Sample (Sample #LDRM2).

. Solids (g)	Buchner Funnel + Wet Filter Paper + Wet Filter Cake (g)	Buchner Funnel + Dry Filter Paper + Dry Filter Cake (g)	Dry Filter Paper (g)	Wet Filter Paper (g)	Water in Filter Cake (g)	Water in Saturated Ore (%)
50.00	394.50	390.16	0.43	2.86	1.91	3.8

Ore Leaching with H2O2 and CH3COOH

The stoichiometric amount of H2O2 was calculated on the basis that each mole of H2O2 would be required to oxidize one mole of sulphide bearing mineral (viz PbS and ZnS) according to Equations 1 and 2.

$$PbS(s) + H_2O_2 \rightarrow PbO(s) + H_2O + S^{o}(s) \qquad (1)$$

$$ZnS(s) + H_2O_2 \rightarrow ZnO(s) + H_2O + S^{o}(s) \qquad (2)$$

The stoichiometric amount of CH3COOH was calculated on the basis that each mole of CH3COOH would be required to convert each equivalent mole of the target metals (Pb, Mn, Zn and Ca) to metal acetates (Pb(CH3COO)2, Mn(CH3COO)2, Zn(CH3COO)2 and Ca(CH3COO)2) as described in Equations 3 to 6.

$$PbO + 2\ CH3COOH \rightarrow Pb(CH3COO)2 + H2O \qquad (3)$$

$$MnCO3 + 2\ CH3COOH \rightarrow Mn(CH3COO)2 + H2O + CO2 \qquad (4)$$

$$ZnO + 2\ CH3COOH \rightarrow Zn(CH3COO)2 + H2O \qquad (5)$$

$$CaCO3 + 2\ CH3COOH \rightarrow Ca(CH3COO)2 + H2O + CO2 \qquad (6)$$

The theoretical weights of concentrated H2O2 and CH3COOH required to leach 50 g of Yava ore sample #LDRM2 are summarized in Table 3.

Table 3: Theoretical Stoichiometric Amount of H2O2 and CH3COOH Required to Leach 50 g Yava Ore Sample #LDRM2.

Wt. Ore (g)	Wt. Conc. H_2O_2 Required (g)	Wt. Conc. CH_3COOH Required (g)
50.00	0.708	3.126

The leach solutions used in this test work were prepared from ACS reagent grade 30 wt% H2O2 and ACS reagent grade CH3COOH specified as greater than or equal to 99.7 wt% concentration. For the purpose of this test work, calculations of the required amount of CH3COOH were based on a concentration of 99.7 wt%. The test conditions and calculated weights of ACS reagent grade 30 wt% H2O2 and ACS reagent grade 99.7 wt% CH3COOH required for each test are summarized in Table 4. To facilitate adequate mixing of the ore with the leach solution, deionized water was added (amount for each test summarized in Table 4) to each leach trial to target a 65 % pulp density (i.e. 65 wt% of solids).

Table 4: Test Conditions for Leaching 50 g of Yava Ore (Sample #LDRM2). Weights of CH3COOH are Based on 150% of the Stoichiometric Amount Required to Convert Target Metals to Metal Acetates.

Test No.	H2O2: Sulphide Minerals Molar Ratio	Wt. Of 30wt% H_2O_2 in Lixiviant (g)	$Wt._{99.7\%}$ CH_3COOH in Lixiviant (g)	Wt. H_2O Added in Leach (g)
A-1	0.10:1	0.24	4.70	21.98
A-2	0.25:1	0.59	4.70	21.63
A-3	0.50:1	1.18	4.70	21.04
A-4	0.75:1	1.77	4.70	20.45
A-5	1:1	2.36	4.70	19.86
A-6	2:1	4.72	4.70	17.50
A-7	3:1	7.09	4.70	15.13
A-8	4:1	9.45	4.70	12.77
A-9	0:1	0.00	4.70	22.22
B-1	0:1	O2 Used in Place of H2O2 Leach with CH3COOH First Followed by O2	4.70	22.22
C-1	0:1	O2 Used in Place of H2O2 O2 Exposure First Followed by Treatment with CH3COOH	4.70	22.22

Each lixiviant given in Table 4 was prepared by adding the required ACS reagent grade 30 wt% H2O2 into a 125-mL polypropylene container, followed by the addition of the required ACS reagent grade CH3COOH. The sample container was capped and gently swirled to ensure good mixing. Each lixiviant solution was then placed in a dark corner until required – no more than 10 minutes.

The results of the leaching trials conducted in accordance to the procedures in Appendix D are summarized in **Table 5**.

The leach results show that under the conditions tested, the maximum leach efficiency for Pb was 71.8% at a H2O2 to sulphide minerals (Pb + Zn) molar ratio of 2:1. With respect to Zn, Mn and Ca leach efficiency, low concentrations in the intial ore sample (0.08%, 0.04% and 0.32%) respectively resulted in fluctuating calculated leach efficiency due to the practical limits of the assays. With respect to elemental sulphur, the maximum S^o

increase (resulting from the oxidation of sulphides) was at a H_2O_2 to sulphide minerals ratio of 1:1. Above a H_2O_2 to sulphide minerals molar ratio of 1:1, sulphate (caused by over oxidation of the sulphur species) begins to increase (which consumes H_2O_2 without any useful benefit). O_2 as an oxidant was significantly inferior to H_2O_2 under the conditions tested. Leach time was not examined as a variable in this study. This can be examined at a later date if required.

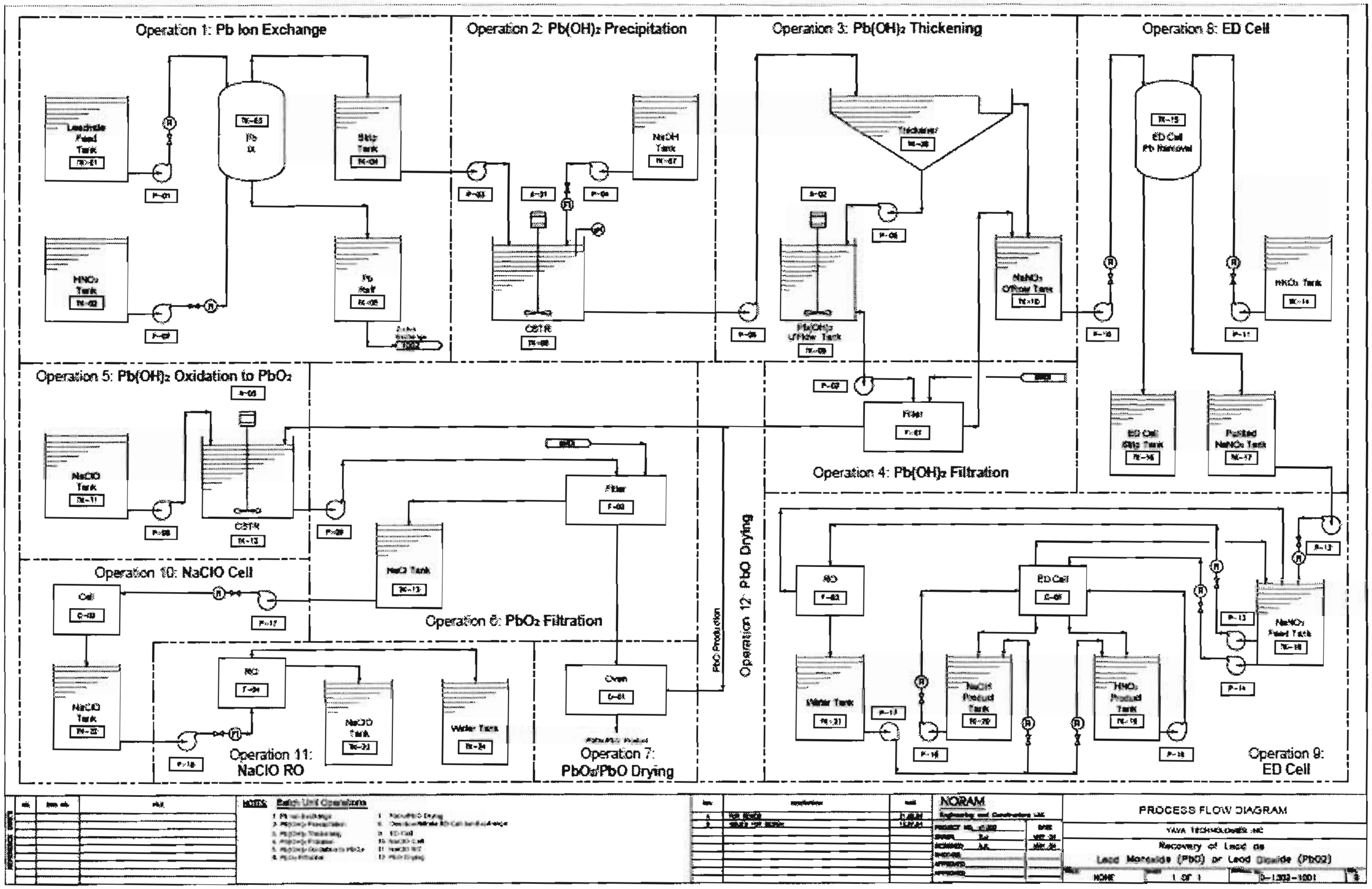
PDF

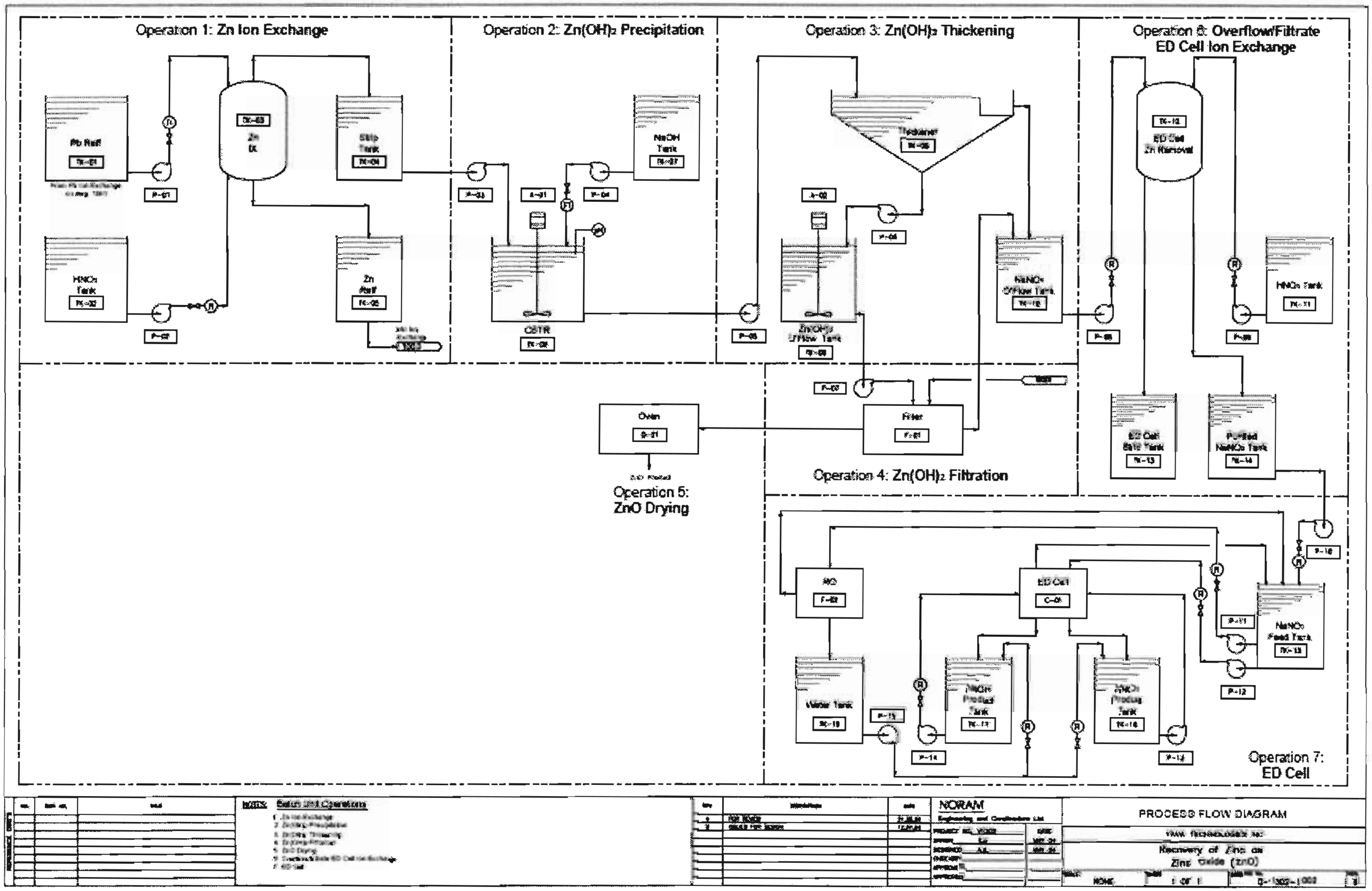
Table 5. Analyses for Ore Leach Study

Test No.	H2O2:(Pb+Zn) Molar Ratio	Sample	Specific Gravity	Weight (g)	pH	Assay Units	Assay Results								Leach Efficiency (%)				S^o increase after leaching (wt %)	SO_4 increase after leaching (wt%)	Oxidation of S2- (wt%)
							Pb	Zn	Mn	Ca	S^o	SO_4	$^*S^{2-}$	S_T	Pb	Zn	Mn	Ca			
		Feed Solid		**50.00**		**%**	**8.04**	**0.08**	**0.04**	**0.32**	**0.21**	**0.28**	**1.23**	**1.53**							
A-1	0.10 1	PLS	1.04	27.80	3.2	mg/L	22020	9 9	279	959											
		Wash Soln.	1.00	447.59	3.9																
		Residue		48.98		%	7.53	0.05	0.04	0.32	0.27	0.25	1.20	1.55	8.3%	33.9%	13.2%	2.0%	26%	-13%	4%
			Wt. Loss	**2.0%**																	
A-2	0.25.1	PLS	1.10	28.33	3.2	mg/L	42299	11 7	297	935											
		Wash Soln.	1.00	434.84	4 1																
		Residue		48.41		%	5.67	0.06	0.03	0.26	0 36	0.27	1.04	1.49	31.7%	28.6%	34%	21%	66%	-7%	18%
			Wt. Loss	**3.2%**																	
A-3	0.50:1	PLS	1.10	28.82	3.3	mg/L	54374	15.3	307	968											
		Wash Soln.	1.00	461.69	4.2																
		Residue		47.92		%	4 26	0.06	0 03	0.23	0.56	0.27	0.92	1.57	49.2%	32 9%	43.4%	31.1%	156%	-8%	28%
			Wt. Loss	**4.2%**																	
A-4	0 75.1	PLS	1.08	28.81	3.2	mg/L	50948	13 3	281	778											
		Wash Soln.	1.00	484.52	4.0																
		Residue		47.96		%	4.67	0 06	0.04	0 32	0.67	0.30	0 71	1.48	44.3%	32.9%	23.7%	4.1%	206%	3%	44%
			Wt. Loss	**4.1%**																	
A-5	1:1	PLS	1.12	29.39	3.2	mg/L	81971	18.3	305	898											
		Wash Soln.	1 00	468.05	4.4																
		Residue		47.42		%	3.58	0.06	0.03	0.26	0.84	0.32	0.70	1.65	57.8%	32.4%	33%	23%	279%	8%	46%
			Wt. Loss	**5.2%**																	
A-6	2 1	PLS	1.12	29.39	3.2	mg/L	98364	25.0	315	976											
		Wash Soln.	1.00	457.41	4.4																
		Residue		47.21		%	2.40	0.06	0.03	0.29	0.70	0.45	0.68	1.53	71.8%	35.1%	29.2%	14.4%	215%	52%	48%
			Wt. Loss	**5.6%**																	
A-7	3.1	PLS	1.12	28.66	3.1	mg/L	88695	29.3	322	967											
		Wash Soln.	1.00	434.93	4.3																
		Residue		47.64		%	2 75	0.05	0 03	0.26	0.50	0.51	0.69	1.36	67 4%	44.0%	33%	23%	127%	74%	46%
			Wt. Loss	**4.7%**																	
A-8	4.1	PLS	1 12	27 82	3 1	mg/L	80193	44 3	366	1504											
		Wash Soln.	1.00	474.54	4.3																
		Residue		47.89		%	3.53	0 05	0.03	0.31	0 62	0.56	0.57	1 38	57 9%	43.7%	26.0%	7 2%	183%	92%	55%
			Wt. Loss	**4.2%**																	
A-9	CH_3COOH Only	PLS	1.04	27.69	3.0	mg/L	16978	9.1	296	1477											
		Wash Soln.	1.00	434.33	3 9																
		Residue		49.12		%	6 79	0.06	0 03	0.42	0 19	0.28	1.25	1.53	17 0%	28.8%	28.6%	0% est	-11%	-2%	0%
			Wt. Loss	**1.8%**																	
B-1	CH_3COOH Followed by O_2	PLS	1.04	26.91	3.2	mg/L	16862	8.2	216	1096											
		Wash Soln.	1.00	449.14	3.9																
		Residue		49.18		%	7 31	0 04	0 04	0 32	0.01	0.21	1.32	1.40	10 6%	50 8%	10.6%	1 6%	-95%	-26%	0% est
			Wt. Loss	**1.6%**																	
C-1	O_2 Followed by CH_3COOH	PLS	1.03	18.88	3.1	mg/L	13510	6.6	119	795					6.2%	0.3%	**9.9%**	**9.1%**			
		Wash Soln.	1.00	421.39	4.0																
		Residue		49.13		%	6 87	0.05	0.03	0.23	0 0	0.19	1.29	1 35	16 0%	43.5%	26.3%	29 4%	-100%	-33%	0% est
			Wt. Loss	**1.7%**																	

Note: * S^{2-} Calculated by Difference

APPENDIX A – Yava Preliminary Process Flow Diagrams

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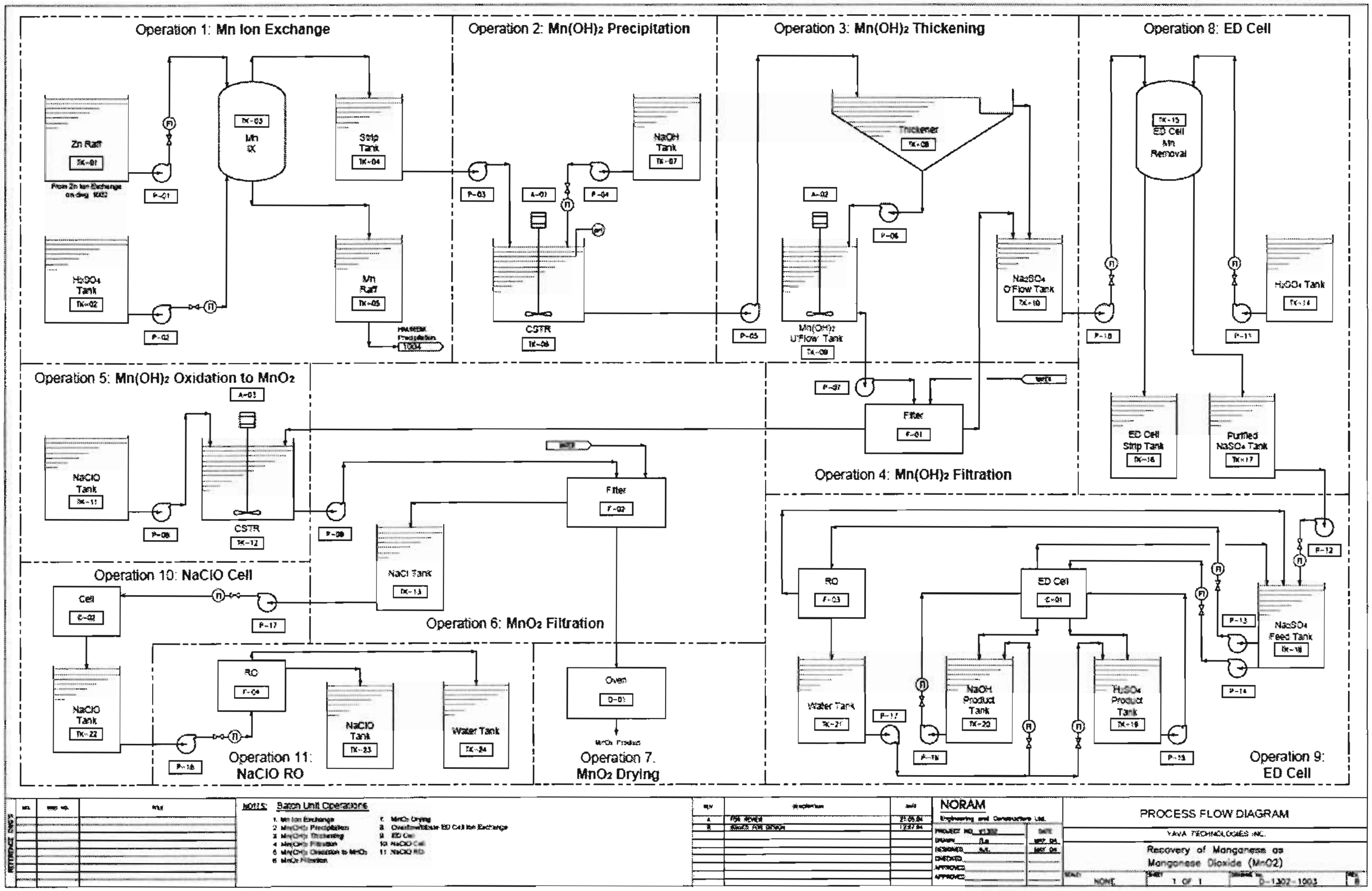
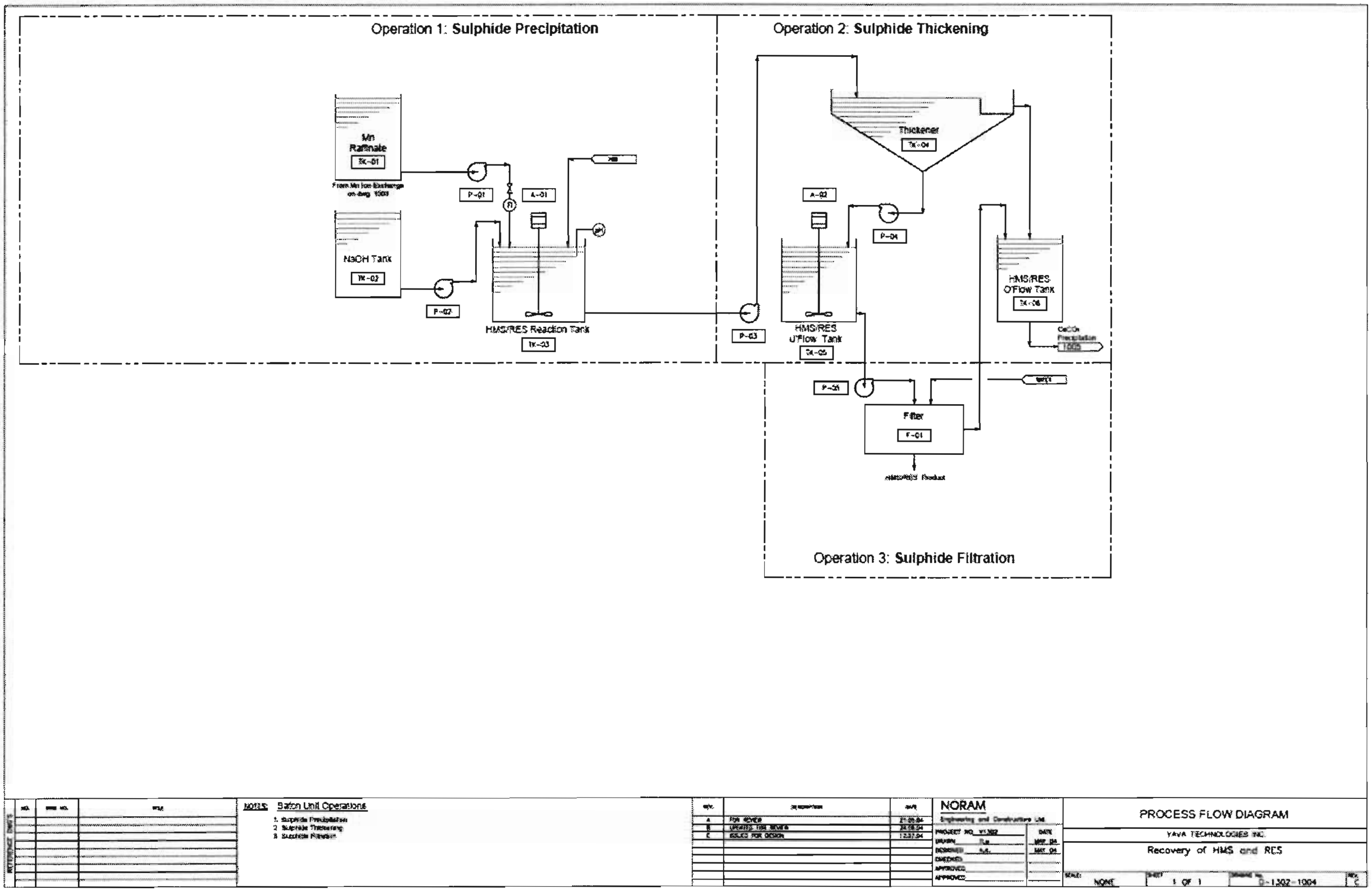
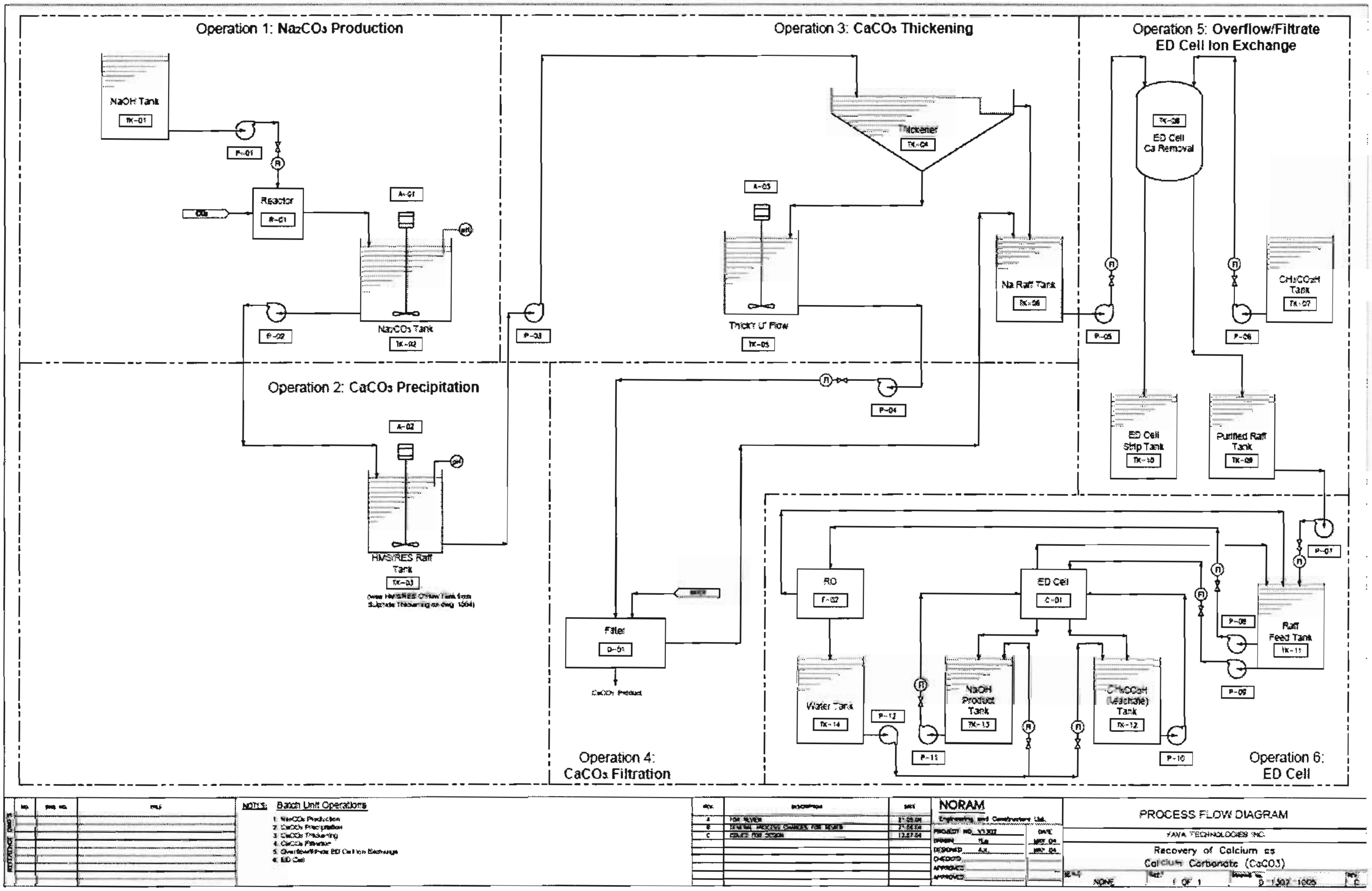
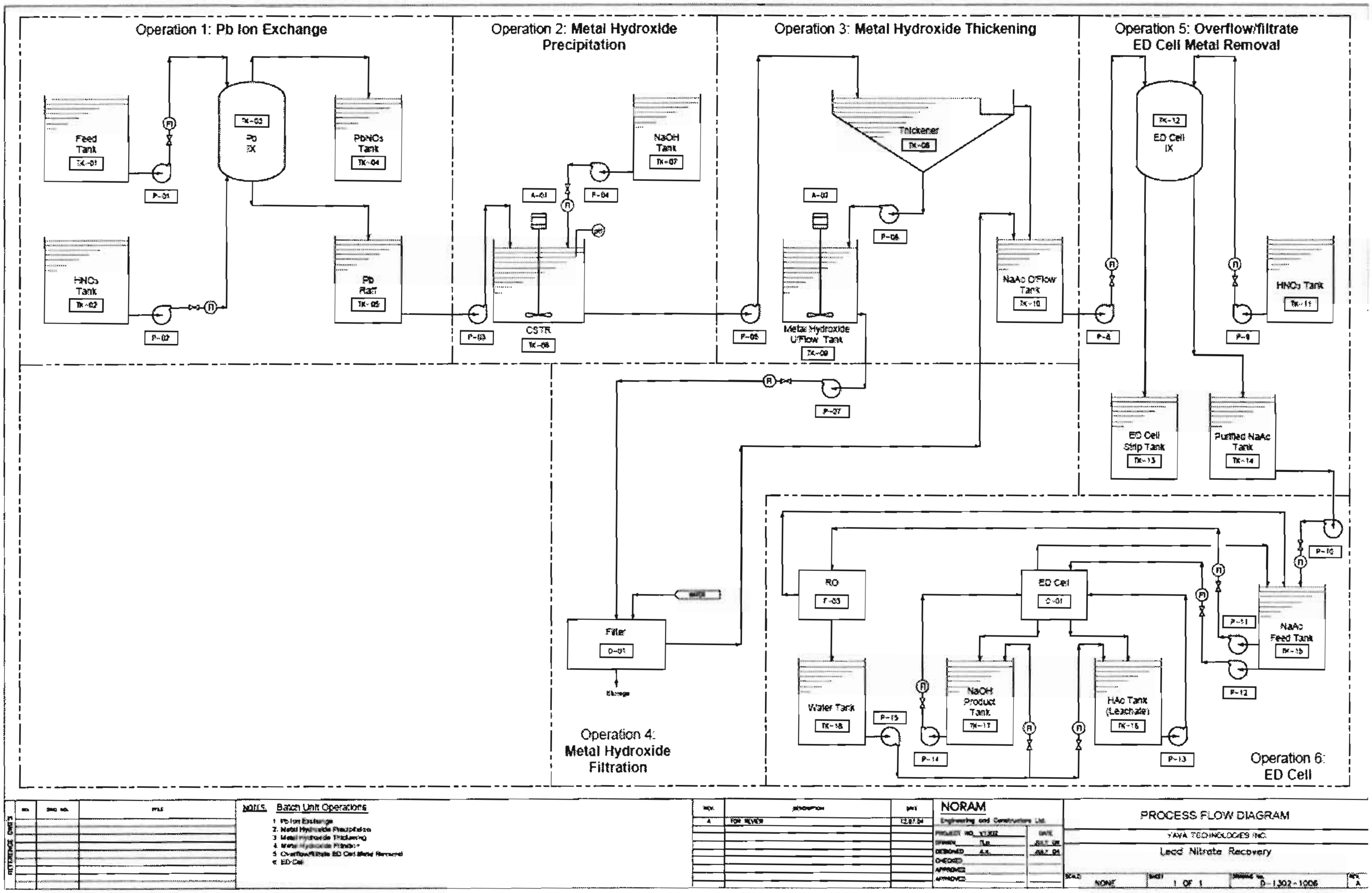
Pb Raff
P-01
HNO3 Tank
P-02
Zn IX
Strip Tank
Zn Raff
Operation 2: Zn(OH)2 Precipitation
NaOH Tank
A-01
CSTR
Operation 3: Zn(OH)2 Thickening
Thickener
A-02
Zn(OH)2 O'Flow Tank
NaNO3 O'Flow Tank
Operation 6: Overflow/Filtrate ED Cell Ion Exchange
ED Cell Zn Removal
HNO3 Tank
Oven
ZnO Product
Operation 5: ZnO Drying
Filter
P-07
Operation 4: Zn(OH)2 Filtration
ED Cell Strip Tank
Purified NaNO3 Tank
TK-13
TK-14
RO
F-02
ED Cell
C-01
Water Tank
NaNO3 Product Tank
HNO3 Product Tank
NaNO3 Feed Tank
P-11
P-12
P-14
Operation 7: ED Cell
NOTES: Batch Unit Operations
NORAM
Engineering and Constructors Ltd
PROCESS FLOW DIAGRAM
Recovery of Zinc as Zinc Oxide (ZnO)
NONE
1 OF 1


Operation 1: Mn Ion Exchange
Zn Raff
TK-01
From Zn Ion Exchange on dwg. 1002
P-01
TK-03
Mn IX
Strip Tank
TK-04
H₂SO₄ Tank
TK-02
P-02
Mn Raff
TK-05
Mn(OH)₂ Precipitation 1004
Operation 2: Mn(OH)₂ Precipitation
P-03
A-01
P-04
NaOH Tank
TK-07
CSTR
TK-06
Operation 3: Mn(OH)₂ Thickening
Thickener
TK-08
A-02
P-06
P-05
Mn(OH)₂ U'Flow Tank
TK-09
Na₂SO₄ O'Flow Tank
TK-10
Operation 8: ED Cell
TK-15
ED Cell Mn Removal
H₂SO₄ Tank
TK-14
P-10
P-11
ED Cell Strip Tank
TK-16
Purified Na₂SO₄ Tank
TK-17
Operation 5: Mn(OH)₂ Oxidation to MnO₂
A-03
NaClO Tank
TK-11
P-08
CSTR
TK-12
P-07
Water
Filter
F-01
Operation 4: Mn(OH)₂ Filtration
Water
Filter
F-02
P-09
NaCl Tank
TK-13
Operation 6: MnO₂ Filtration
Operation 10: NaClO Cell
Cell
C-02
P-17
NaClO Tank
TK-22
P-18
RO
F-04
NaClO Tank
TK-23
Water Tank
TK-24
Operation 11: NaClO RO
Oven
D-01
MnO₂ Product
Operation 7. MnO₂ Drying
RO
F-03
Water Tank
TK-21
ED Cell
C-01
NaOH Product Tank
TK-20
H₂SO₄ Product Tank
TK-19
Na₂SO₄ Feed Tank
TK-18
P-12
P-13
P-14
P-15
P-16
P-17
Operation 9: ED Cell
REFERENCE DWG'S
NO.
DWG. NO.
TITLE
NOTES: Batch Unit Operations
1. Mn Ion Exchange
2. Mn(OH)₂ Precipitation
3. Mn(OH)₂ Thickening
4. Mn(OH)₂ Filtration
5. Mn(OH)₂ Oxidation to MnO₂
6. MnO₂ Filtration
7. MnO₂ Drying
8. Overflow/Bleed ED Cell Ion Exchange
9. ED Cell
10. NaClO Cell
11. NaClO RO
REV
DESCRIPTION
DATE
A
FOR REVIEW
B
ISSUED FOR DESIGN
NORAM
Engineering and Constructors Ltd.
PROJECT NO.
DRAWN
DESIGNED
CHECKED
APPROVED
APPROVED
DATE
PROCESS FLOW DIAGRAM
YAVA TECHNOLOGIES INC.
Recovery of Manganese as Manganese Dioxide (MnO2)
SCALE NONE
SHEET 1 OF 1
DRAWING No. D-1302-1003
REV. B


Operation 1: Sulphide Precipitation
Operation 2: Sulphide Thickening
Operation 3: Sulphide Filtration
Mn Raffinate
TK-01
From Mn Ion-Exchange on dwg. 1003
NaOH Tank
TK-02
P-01
P-02
A-01
FI
pH
HMS/RES Reaction Tank
TK-03
P-03
Thickener
TK-04
A-02
P-04
HMS/RES U'Flow Tank
TK-05
HMS/RES O'Flow Tank
TK-06
CaCO3 Precipitation
1005
P-05
Filter
F-01
HMS/RES Product
NOTES: Batch Unit Operations
1. Sulphide Precipitation
2. Sulphide Thickening
3. Sulphide Filtration
A
B
C
FOR REVIEW
UPDATED FOR REVIEW
ISSUED FOR DESIGN
NORAM
Engineering and Construction Ltd.
PROJECT NO.
DRAWN
DESIGNED
CHECKED
APPROVED
APPROVED
DATE
PROCESS FLOW DIAGRAM
YAVA TECHNOLOGIES INC.
Recovery of HMS and RES
SCALE: NONE
SHEET 1 OF 1
D-1302-1004
C


Operation 1: Na2CO3 Production
NaOH Tank
TK-01
P-01
Reactor
R-01
CO2
A-01
pH
Na2CO3 Tank
TK-02
P-02
Operation 2: CaCO3 Precipitation
A-02
pH
HMS/RES Raff Tank
TK-03
Operation 3: CaCO3 Thickening
Thickener
TK-04
A-03
Thick'r U' Flow
TK-05
P-03
Na Raff Tank
TK-06
P-04
Operation 4: CaCO3 Filtration
Filter
D-01
CaCO3 Product
Operation 5: Overflow/Filtrate ED Cell Ion Exchange
TK-08
ED Cell Ca Removal
CH3CO2H Tank
TK-07
P-05
P-06
ED Cell Strip Tank
TK-10
Purified Raff Tank
TK-09
Operation 6: ED Cell
RO
F-02
ED Cell
C-01
Raff Feed Tank
TK-11
P-07
P-08
P-09
Water Tank
TK-14
P-12
NaOH Product Tank
TK-13
P-11
CH3CO2H (Leachate) Tank
TK-12
P-10
NOTES: Batch Unit Operations
1. Na2CO3 Production
2. CaCO3 Precipitation
3. CaCO3 Thickening
4. CaCO3 Filtration
5. Overflow/Filtrate ED Cell Ion Exchange
6. ED Cell
REFERENCE DWG'S
A
B
C
NORAM
Engineering and Constructors Ltd.
PROCESS FLOW DIAGRAM
Recovery of Calcium as Calcium Carbonate (CaCO3)
NONE
1 OF 1
D-1302-1005
C


Operation 1: Pb Ion Exchange
Feed Tank TK-01
P-01
Pb IX TK-03
PbNO₃ Tank TK-04
HNO₃ Tank TK-02
P-02
Pb Raff TK-05
Operation 2: Metal Hydroxide Precipitation
NaOH Tank TK-07
P-04
A-01
pH
CSTR TK-06
P-03
Operation 3: Metal Hydroxide Thickening
Thickener TK-08
A-02
P-06
Metal Hydroxide U'Flow Tank TK-09
P-05
NaAc O'Flow Tank TK-10
P-07
Operation 4: Metal Hydroxide Filtration
Filter D-01
Storage
Operation 5: Overflow/filtrate ED Cell Metal Removal
ED Cell IX TK-12
HNO₃ Tank TK-11
P-8
P-9
ED Cell Strip Tank TK-13
Purified NaAc Tank TK-14
Operation 6: ED Cell
RO F-03
ED Cell C-01
NaAc Feed Tank TK-15
P-10
P-11
P-12
Water Tank TK-18
P-15
NaOH Product Tank TK-17
P-14
HAc Tank (Leachate) TK-16
P-13
NO. DWG. NO. TITLE
REFERENCE DWG'S
NOTES: Batch Unit Operations
1. Pb Ion Exchange
2. Metal Hydroxide Precipitation
3. Metal Hydroxide Thickening
4. Metal Hydroxide Filtration
5. Overflow/Filtrate ED Cell Metal Removal
6. ED Cell
REV. DESCRIPTION DATE
A FOR REVIEW
NORAM
Engineering and Construction Ltd.
PROJECT NO.
DRAWN
DESIGNED
CHECKED
APPROVED
APPROVED
DATE
PROCESS FLOW DIAGRAM
YAYA TECHNOLOGIES INC.
Lead Nitrate Recovery
SCALE NONE
SHEET 1 OF 1
DRAWING No. D-1302-1006
REV. A

50	ft	3/4"ID x 1"-OD Clear Vinyl Tubing
40	ft	1/2"-ID x 3/4"-OD VINYLITE Reinforced Vinyl Tubing
20	each	3/8"-Nominal Dia. G5M Type 316 Stainless Micro Gear Clamp
20	each	5/8"-Nominal Dia. G5 Type 316 Stainless Micro Gear Clamp
5	each	1"-Nominal Dia. G5 Type 316 Stainless Micro Gear Clamp
1	each	1/4"-Hose x 1/4"-NPT(M) Polyurethane Recoil Air Hose, (10 ft-L)
1	each	Rubber Tipped Blow Gun
1	each	Hose Cutter
1	pkg.	Hose Cutter Replacement Blades
1	each	1/2"-ID x 50-ft G311 Industrial Water Hose, Rubber
1	each	1/4" x 6" PTFE Coated Stir Rods
2	case	600-mL Pyrex Brand 1000 Beakers (6 each/case)
2	each	500-mL Low Form Beakers with Handle, PP
2	each	1000-mL Low Form Beakers with Handle, PP
2	each	2000-mL Low Form Beakers with Handle, PP
2	each	250-mL Pyrex Brand 3025 Graduated Cylinders, Double Metric
2	each	1000-mL Pyrex Brand 3025 Graduated Cylinders, Double Metric
2	each	500-mL Economical Graduated Cylinder, PP
2	each	1000-ml Economical Graduated Cylinders, PP
1	pkg.	750-mL Nalgene Wide Mouth Unitary Wash Bottle, LDPE (4 each/pkg.)
1	each	4" - Parafilm Wrap and Dispenser (125-ft roll)
2	pkg.	500-mL Pyrex Brand 5340 Flasks (6 each/pkg.)
1	pkg.	290-mL Powder Funnel, PP (6 each/pkg.)
10	each	90-mm Plastic Buchner Funnel, PP
1	each	4000-mL Pyrex Brand 5340 Flasks
1	pkg.	Size #12 Solid Laboratory Rubber Stopper (Black),
1	each	Standard Duty Dry Vacuum Piston Pump
4	box	Whatman 5, Qualitative Filter Paper, 9-cm dia. (2.5-micron)
1	pkg.	25-mm PTFE Syringe Filters, 0.45 micron, (100 pcs./pkg.)
1	pkg.	46-mm OD Top x 29-mm OD Bottom Fisherbrand Filter Flask Adapter, Neoprene, (12 pcs./pkg.)
1	each	400-mL CoorsTek Mortar, Porcelain
1	each	194-mm CoorsTek Pestle, Porcelain
1	pkg	"Reagent Digger" Spatula, 304SS, 6-1/2" L, (12 each/pkg.)
1	each	175-mL Scoop, SS
1	each	1550-mL Scoop, SS
1	pkg	High-Heat Spatulas, PP/Rubber, 16-1/2" L, (6 each/pkg.)
1	each	pH/mV/°C Accumet Portable Meter Kit
1	each	Epoxy-Body Combination pH Electrode
1	each	Flexible Electrode Holder
1	each	pH 1.68, Oakton pH Buffer Solution, 500-mL
1	each	Oakton Buffer Solution Pack, buffers 4, 7 and 10, 500-mL/each
2	each	Electrode Saver Bottle
1	each	Thermo Orion ORP Standards (475-mL)
1	pkg.	1/2"-ID Kynar Straight Barbed Connectors (10 pcs./pkg.)
1	pkg.	3/8"-ID Kynar Straight Barbed Connectors (10 pcs./pkg.)

1	pkg.	1/4"-ID Kynar Straight Barbed Connectors (10 pcs./pkg.)
1	pkg.	1/2"-MPT x 1/4"-ID Barbed Kynar Connector, (10 pcs./pkg.)
1	pkg.	1/2"-MPT x 3/8"-ID Barber Kynar Connector, (10 pcs./pkg.)
1	pkg.	1/2"-MPT x 1/2"-ID Barbed Kynar Connector, (10 pcs./pkg.)
2	each	100-US Gallon Polypropylene Open Top Conical Bottom Tank, (32"-dia. X 38"-H) complete with Removable Cover, 1/2"-PP Half Coupling / Outlet, one 1/2"-PP MPT Nipple / Outlet, 1/2"-PP Ball Valve w/Viton Gasket, Epoxy Coated Steel Stand with Agitator Support (64" top to floor)
2	each	250-US Gallon Polypropylene Open Top Conical Bottom Tank, (43"-dia. X 54"-H) complete with Removable Cover, 1/2"-PP Half Coupling / Outlet, one 1/2"-PP MPT Nipple / Outlet, 1/2"-PP Ball Valve w/Viton Gasket, Epoxy Coated Steel Stand with Agitator Support (74" top to floor)
2	each	20-350 rpm (1/4-hp), Variable Speed Lightnin Batch Mixers, 48"-L x 3/4"-dia. SS Shaft, 10"-dia. SS Impeller, 20° Tank Clamp
2	each	20-280 rpm (1/2-hp), Variable Speed Lightnin Batch Mixers, 48"-L x 3/4"-dia. SS Shaft, 11.2"-dia. Dual SS Impellers, 20° Tank Clamp
6	each	1/2" PVC Bulkheads with Neoprene Gasket
20	each	1/2"-PVC/Viton True Union Ball Valves, Socket/Threaded Ends
10	ft	1/2"-PVC SCH. 40 White B/E Pipe
10	ft	3/4" Clear PVC SCH. 40 Pipe
10	ft	1-1/2" Clear PVC SCH. 40 Pipe
10	ft	3" Clear PVC SCH. 40 Pipe
2	each	3/4" x 1/2" PVC SCH. 80 Socket Reducing Coupling
2	each	1-1/2" x 1/2" PVC SCH. 80 Socket Reducing Coupling
2	each	3" x 1-1/2" PVC SCH. 80 Socket Reducing Coupling
2	each	1-1/2" x 1/2" PVC SCH. 40 Reducing Bushing Slip x Slip
15	each	1/2"-PVC SCH. 40 Socket Tees (WHITE)
5	each	1/2"-PVC SCH. 40 90° Socket Elbows (WHITE)
5	each	1/2"-PVC SCH. 40 Socket Couplings (WHITE)
5	each	1/2"-PVC SCH. 40 F-Socket to FPT Adapter (WHITE)
5	each	1/2"-PVC SCH. 40 M-Socket to MPT Adapter (WHITE)
10	each	1/2"-PVC SCH. 80 MPT Nipple (Short)
10	each	1/2"-PVC SCH. 80 MPT Nipple (3"-L)
1	each	1-Pt., 705 Clear Weld-On Cement
1	each	1-Pt., P-70 Clear Weld-On Primer
1	each	11.5-gallon Pressure Filter, SS, Floor Mount, Neoprene Gasket
1	each	14"-dia. Filter Cloth
1	pkg.	14"-dia. Filter Paper
1	pkg	3/8" High-Flow Quick Disconnect Coupling Hose Barbed Body, Straight-Through, 3/8"-ID Tube, PP, (10pcs./pkg.)
1	pkg	3/8" High-Flow Quick Disconnect Coupling Hose Barbed

		Insert, Straight-Through, 3/8"-ID Tube, PP, (10 pcs./pkg.)
1	pkg	3/4"-MPT x 3/4"-ID Barbed Kynar Connector, (5 pcs./pkg.)
1	each	1/4"-Brass Female Pipe Thread Nipple 1/4"-NPT(M)
1	each	3/8"- MPT Brass Pipe Bushing 1/4"-FPT
1	each	1/2"-MPT Brass Pipe Bushing 1/4"-FPT
1	each	1/4"-NPT 90° Swivel Joint
2	each	275-Gallon Mauser Steel/HDPE Totes with Sloped Bottom and 2"-PVC/EPDM Ball Valve
2	each	1/2"-FPT Hayward PVC Valve with EPDM Diaphragm
1	each	1/8"-Thick Gray PVC Perforated Sheet (4' x 8')
1	each	28 mesh - Stainless Steel 12" Testing Sieve, Full Height
1	each	16 mesh - Stainless Steel 12" Testing Sieve, Full Height
1	each	12 mesh - Stainless Steel 12" Tesing Sieve, Full Height
1	each	48 mesh - Stainless Steel 12" Tesing Sieve, Full Height
2	each	Easy-Load II Pump Head (SS Rotor)
3	each	2-Channel Easy-Load II Pump Head (SS Rotor)
2	each	L/S Variable Speed Modular Drive (6-600 rpm)
2	each	Mounting Hardware for Easy Load Heads (SS, 2 Heads)
1	box	L/S 16-Neoprene, Precision Pump Tubing (50-ft)
1	box	L/S 25-Neoprene, Precision Pump Tubing (50-ft)
4	each	I/P Easy Load Pump Head (SS Rotor)
4	each	I/P Variable Speed Modular Drive (20-650 rpm)
1	box	I/P 73 - Neoprene, Precision Pump Tubing (50-ft)
1	box	I/P 82 - Neoprene, Precision Pump Tubing (50-ft)
1	each	Replacement Combination pH/ATC Electrode
2	each	General Purpose ORP Electrode (Double Juction, BNC)
4	each	2"-PVC SCH.80 Threaded Union Pressure Fitting
2	each	2"-MPT x 1/2"-FPT PVC SCH.80 Reducing Bushing
2	each	2"-MPT PVC SCH.80 Plugs
1	pkg.	1/2"-MPT x 3/8"-ID Barber Kynar Connector, (10 pcs./pkg.)
1	each	250-mL Vaseline - Petroleum Jelly
		Teflon Tape
1	each	3" Flat Head Paint Brush
1	roll	Craft Paper
1	each	Garden Hose Nozzle
1	each	1/2"-Chuck Cordless Drill
1	each	1/2"- Chuck Pistol Grip Drill
1	each	Wood Drill Bit Set - Large
1	each	Channel Pliers - Large
1	each	Channel Pliers - Medium
1	each	Pliers - Medium
1	each	Crescent Wrench - Large
1	each	Crescent Wrench - Medium
1	each	Vise Grips - Large
1	each	Vise Grips Large Jaw Locking Pliers
1	each	Plastic Hammer - Medium
1	each	Screw Driver Set - Multi
1	each	Exactor Knife and Blades

1	each	Tubing Cutter
1	each	Tool Box - Small
1	each	Round File - Medium
1	each	Round File - Small
1	each	Medium Flat file - Medium
1	each	Sand Paper Pack
1	each	Bench Vise
1	each	Ratchet Threaded Sets, 1/2" thru 1" NPT

APPENDIX C – ICP Analysis of Yava Leachate Before and After Filtering and Backwash Solids

iPL
International Plasma Labs Ltd.
ISO 9001:2000 Certified Company

2036 Columbia Street
Vancouver B C
Canada V5Y 3E1

Website [illegible]
Email [illegible]
Phone [illegible]
Fax [illegible]

Certificate# 05D0723
Client Vizon SciTec Inc.
Project 404-0751
Shipment#
PO# R67097
No of Samples 6
Analysis #1 Pb Zn Mn Ca Assay
Analysis #2: ICP(H2O)30 Water Package in mg/L
Analysis #3
Comment #1
Comment #2
Date In Apr 11 2005
Date Out Apr 11 2005

Sample Name	SampleType	Al mg/L	Sb mg/L	As mg/L	Ba mg/L	Bi mg/L	Cd mg/L	Ca mg/L	Cr mg/L	Co mg/L	Cu mg/L	Fe mg/L	La mg/L	Pb mg/L	Mg mg/L	Mn mg/L
Leachate Soln	Solution	108.2	<0.1	<0.2	<0.01	<0.1	<0.01	2306.8	12.37	0.93	0.77	1393.2	0.42	10773.35	874.4	350.98
Filtrate Soln	Solution	104.4	<0.1	<0.2	<0.01	<0.1	<0.01	2306.1	12.01	0.9	0.74	1390.99	0.43	10486.56	951.7	340.86
Backwash Soln	Solution	16.4	<0.1	<0.2	<0.01	<0.1	<0.01	391.2	1.93	0.26	0.34	294.88	0.27	1314.15	141.4	53.52
Minimum detection		0.2	0.1	0.2	0.01	0.1	0.01	0.1	0.01	0.01	0.01	0.03	0.05	0.05	0.1	0.01
Maximum detection		9999	9999	9999	999	9999	999	9999	9999	9999	9999	9999	999	9999	9999	999
Method		ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O

Sample Name	SampleType	Hg mg/L	Mo mg/L	Ni mg/L	P mg/L	K mg/L	Sc mg/L	Ag mg/L	Na mg/L	Sr mg/L	Tl mg/L	Ti mg/L	W mg/L	V mg/L	Zn mg/L	Zr mg/L
Leachate Soln	Solution	<0.05	0.34	3.6	<0.1	49	0.22	<0.02	30	1.69	<0.2	<0.1	<0.1	<0.01	20.71	<0.01
Filtrate Soln	Solution	<0.05	0.31	3.5	<0.1	49	0.21	<0.02	30	1.71	<0.2	<0.1	<0.1	0.06	20.06	0.03
Backwash Soln	Solution	<0.05	0.13	0.73	<0.1	23	0.03	<0.02	14	0.32	<0.2	<0.1	<0.1	<0.01	3.5	<0.01
Minimum detection		0.05	0.02	0.02	0.1	2	0.01	0.03	1	0.01	0.2	0.1	0.1	0.01	0.01	0.01
Maximum detection		9999	9999	9999	9999	9999	100	999	9999	999	999	999	9999	999	9999	999
Method		ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O	ICPH2O

PFD

APPENDIX D

REVISED ORE LEACHING PROCEDURE

Summary

The purpose of this leaching procedure is to:

- Recover the optimal amount of the Pb, Zn, Mn and Ca in the leachate.
- Determine the quantity and effect of the secondary products which may have been formed during the dissolution of the minerals e.g., S^0 or $PbSO_4\downarrow$ (?) in the leaching process, and
- Determine the stoichiometric amount of reagents (acetic acid plus H_2O_2) required for optimum leaching efficiency.
- Compare the efficiency of oxygen with H_2O_2 in the leaching process.
  NB The container and fittings must not be constructed of materials (eg stainless steel and some rubbers) that will catalyze H_2O_2 to water; ideally the container and fittings should be made of glass only.

Procedure

A. Lixiviant (CH_3COOH + H_2O_2)

1 **Determine the volume of water to saturate the dry ore:**

   i. Weigh 50 g of the crushed ore in a 150 mL beaker.
   ii. Add 100 mL of deionized water to the beaker containing the crushed ore.
   iii. Gently shake the mixture of the crushed ore and water on a shaker for 20 minutes.
   iv. Filter the mixture to collect the filtrate (i.e., excess water not used by the ore) in a clean and dry graduated plastic vial. The difference between the original volume of water (100 mL) and the volume of filtrate is equivalent to the volume of water required to saturate the ore.

2 Set-up nine pre-weighed and numerically labelled (**1-9**) clean glass beakers of 150 mL capacity (or plastic vials with stoppers) in a laboratory corner with no direct exposure to light.

3 Conveniently weigh a known amount (50 g) of the ore in each of the nine beakers mentioned in step 2.

4 Prepare the lixiviant (CH_3COOH + H_2O_2 solution) to contain the **0.25:1**, **0.50:1**, **0.75:1**, **1:1**, **2:1**, **3:1**, **4:1** and **5:1** stoichiometric amount of the reagents needed (lixiviant:minerals) to leach the ore in the beakers. The amount of H_2O_2 in the lixiviant is proportional to the formation of PbO and ZnO based on Pb and Zn assay; the acetate addition is based on the formation of soluble Pb, Zn, Mn, Ca, Mg etc acetates. **Analytical grade freshly opened reagents must be used to prepare the lixiviant**.

5 Measure the pH of the freshly prepared lixiviant.

6 Add to the beakers labelled **1-8** containing the crushed ore mixture the amount of lixiviant calculated stoichiometrically to recover the Pb, Zn, Mn and Ca described above. Add the lowest **lixiviant:minerals** stoichiometric amount of the reagent required (**0.25:1**) to **beaker 1** followed by **0.5:1 (beaker 2)**, **0.75:1 (beaker 3)**, **1:1 (beaker 4)**, **2:1 (beaker 5)**, **3:1** (**beaker 6**), **4:1 (beaker 7)** and the highest stoichiometric amount of the reagent **[5:1 (beaker 8)]**. Add a stoichiometric amount of acetic acid only (**without H_2O_2**) as a lixiviant to the **9**th beaker to determine the effect of eliminating H_2O_2 in the leaching process. Dilute each lixiviant solution with the measured volume of deionized water

determined above (**Procedure A, step 1 a**) in order to collect enough leachate for analysis.

7 Shake the beakers gently on a shaker for 20 minutes.

8 Allow the beakers containing the mixture of the lixiviant and the crushed ore to settle for 20 minutes.

9 Filter the supernatant liquid to collect a clear filtrate (leachate) in a clean beaker.

10 Measure the pH of the leachate collected.

11 Analyze the clear filtrate for the Pb, Zn, Mn and Ca content with only the required volume of the sample for analysis. Collect the remainder of the leachate in plastic vials with stoppers.

12 Transfer the ore residue from the beaker to the filter paper.

13 Wash the ore residue thoroughly with water for all 9 samples until the washings are free of Pb, Zn, Mn and Ca. Take scanning electron micrographs (SEM) and x-ray diffraction (XRD) patterns of the ore residue and also analyze the residue for the Pb, Zn, Mn and Ca content plus S° and SO_4^{2-} with only the required amount of the residue for the analysis. Store the remainder of the residue in deionized water in sealed plastic vials.

14 Perform mass balance calculations to determine the leaching efficiency for Pb, Zn, Mn and Ca in each sample.

15 Prepare a detailed report on work and results.

B. Lixiviant (CH_3COOH followed by O_2)

1. Set-up a pre-weighed clean glass beaker of 150 mL capacity in a laboratory corner with no direct exposure to light.

2. Conveniently weigh a known amount (50 g) of the ore in the beaker.
3. Prepare the lixiviant (CH_3COOH only) to contain the stoichiometric amount of the reagent needed (lixiviant : minerals) to leach the ore in the beaker. The amount of acetic acid addition is based on the formation of soluble Pb, Zn, Mn, Ca, Mg etc acetates. Dilute the lixiviant solution (CH_3COOH) with the volume of water (**Procedure A step 1 a)** required to saturate the ore. **Analytical grade freshly opened reagents must be used to prepare the lixiviant**.

4. Add the lixiviant to the beaker containing the crushed ore.

5. Shake the beaker on a shaker for 20 minutes.

6. Bubble oxygen through the mixture of the crushed ore and lixiviant gently for 20 minutes on a shaker.

7. Repeat steps 8 to 15 from **Procedure A** described above.

C. Lixiviant (O_2 followed by CH_3COOH)

1. Set-up a glass column containing 50 g of the crushed ore in a laboratory corner with no direct exposure to light.

2. Saturate the ore with water by gently pumping the water through a conduit at the bottom of the column containing the crushed ore. Please refer to **Procedure A Step 1 a** for the exact volume of water required to saturate the ore.

3. Gently feed oxygen gas through the bottom of the column containing the crushed ore saturated with water for 20 minutes to saturate the ore with oxygen. Minimize the loss of oxygen gas using a column with a tubular glass arm equipped with a glass knob to act as a control valve.

4. Prepare the lixiviant (CH_3COOH only) to contain the stoichiometric amount of the reagent needed (lixiviant : minerals) to leach the ore in the beaker. The amount of acetic acid is based on the formation of soluble Pb, Zn, Mn, Ca, Mg etc acetates. **Analytical grade freshly opened reagents must be used to prepare the lixiviant**.

5. Measure the pH of the freshly prepared lixiviant.

6. Leach the ore in the column by pumping the lixiviant gently through the bottom of the column.

7. Collect the overflow (leachate) from the column in a clean beaker.

8. Measure the pH of the leachate collected.

9. Analyze the leachate for the Pb, Zn, Mn and Ca content with only the required volume of the leachate for analysis. Collect the remainder of the leachate in plastic vials.

10. Wash the ore residue in the column thoroughly until the washings are free of Pb, Zn, Mn and Ca. Take scanning electron micrographs (SEM) and x-ray diffraction (XRD) patterns of the residue and also analyze the residue for the Pb, Zn, Mn and Ca content plus S^o

and SO_4^{2-} with only the required amount of the residue for the analysis. Store the remainder of the residue in de-ionized water.

11. Perform mass balance calculations to determine the leaching efficiency for Pb, Zn, Mn and Ca.

12. Prepare a detailed report on work and results

Note: *Please advise us about the test work schedule in advance so that we may attend the leaching to become familiar with the process by observing the experiment in real-time.*

Vizon Scitec Inc.
3650 Wesbrook Mall
Vancouver, BC
V6S 2L2

Project No. 4040751

Progress Report No 2

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC. DURING JUNE 2005

Prepared by

Norman Chow, Kenneth Seto (VizonScitec Inc)
and
Clive Brereton , Alan Keylock (Noram Engineering and Constructors Ltd

INTRODUCTION

The work carried out in the month of June 2005 focused on ore leaching, lead (Pb) separation using ion exchange, precipitation of Pb as lead nitrate (Pb(NO3)2) using ammonium carbonate ((NH4)2CO3) and bulk sulphide precipitation of dissolved metal ions in Yava leachate.

Ore leach tests examined the possible use of nitric acid (HNO3), and hydrochloric acid (HCl) as alternative leachants to hydrogen peroxide (H2O2) and acetic acid (HAc). Four different HNO3 and one HCl leach conditions were examined on Yava ore. A leach tests was also conducted to examine the efficiency of excess HAc on dissolving a mixture of reagent grade manganese carbonate (MnCO3) and calcium carbonate (CaCO3). This test was conducted to examine the effectiveness of using HAc in the absence of H2O2 to leach Mn and Ca only.

Ion exchange studies to separate Pb ions from Yava leachate was conducted with Purolite C-104H resin. Bench scale beaker and single column flow through tests were conducted to obtain Pb loading and Pb stripping isotherms and breakthrough curves.

Precipitation of Pb as Pb(NO3)2 to produce lead carbonate (PbCO3) using (NH4)2CO3 was examined with the goal of examining coversion rate and purity of the PbCO3 precipitate.

A bulk sulphide precipitation test was conducted on Yava leachate to examine the effectiveness in which various metal ions (contained in Yava leachate) can be precipitated with hydrogen sulphide (H2S).

SUMMARY

1. HNO3 leach tests conducted on Yava ore showed increasing leach efficiencies with increasing HNO3 concentration and increasing leach times.
2. Leaching with concentrated HNO3 resulted in the formation of NO2 gas.
3. The efficiency of using excess HAc to leach reagent grade CaCO3 and MnCO3 mixture. Ca was completed dissolved during the 20 minute leach test, however, Mn was only 51% dissolved.
4. HCl leach tests conducted on Yava ore showed low leach efficiency of Pb (28.2%) under the conditions tested. In addition, HCl leaching resulted in the production of H2S gas.
5. Precipitation of Pb(NO3)2 with stoichiometric amount of (NH4)2CO3 resulted in an 80% coversion of Pb to PbCO3, with the final precipitate being 100% cerussite type $PbCO_3$.

6. Beaker and column Pb ion exchange testing was conducted with a Purolite C104H resin. The selectively for Pb was not perfect with some Fe, Al and Ca picked up by the resin. Loading kinetics tended to be slow. Pb stripping was easily conducted with a 22 wt% HNO3 eluent.
7. Beaker and column Pb ion exchange testing was conducted with a Purolite C104H resin. The selectively for Pb was not perfect with some Fe, Al and Ca picked up by the resin. Loading kinetics tended to be slow. Pb stripping was easily conducted with a 22 wt% HNO3 eluent.
8. Beaker Pb ion exchange testing was conducted with Eichrom's Monophos and Diphonix resins. The selectively for Pb was poor for both of these resins.
9. Sulphide precipitation of Yava leachate showing the amount of each type of metal that can be precipitated with H2S.

RESULTS AND DISCUSSION

Ore Leach Study

Leaching with HNO3

Four leach tests with HNO3 were conducted on Yava ore (Sample LDRM2). Each leach test was identified at Test # D-1, D-2, D-3 and D-4 respectively. Tests D-1 and D-2 were conducted according to the procedure outlined in Appendix A. Test D-3 was conducted according to the procedure in Appendix A with the exception that leaching was conducted with 69 weight percent (wt%) concentrated ACS reagent grade HNO3 (equivalent to 14.3 M HNO3) and increased leach time (60 minutes instead of 20 minutes). Test D-4 was conducted similar to D-3 with the exception that leaching was carried out at a pulp density of 50 wt% solids instead of 65 wt%, leach time was 16 hours instead of 60 minutes and agitation was carried out in a magnetic stir plate instead of swirling in an Erlenmeyer flask. Test conditions are summarized in Table 1.

Table 1. HNO3 Leach Test Conditions

Test #	HNO3 Conc. (M)	Pulp Density (%)	Leach Time (Minutes)	Temperature (deg C)	Agitation
D-1	2.4	65%	20	25	Swirling
D-2	5.0	65%	20	25	Swirling
D-3	14.3	65%	60	25	Swirling
D-4	14.3	50%	960	25	Magnetic Stir Rod

ICP analysis and weight loss of the leach residues in comparison to the head sample (tabulated in Table 2) were used to calculate leach efficiencies for respective metals of interest (Table 3). Elemental sulphur (S^o), sulphate (SO_4) and total sulphur (S_t) analysis were conducted to estimate the effectiveness in which the HNO_3 oxidizes sulphides while minimizing SO_4 production (Table 2).

Table 2. ICP Metals, Sulphur Analysis and Weight Loss Data for Leach Residue.

Test #	Residue Weight (g)	Pb (wt%)	Zn (wt%)	Mn (wt%)	Ca (wt%)	S^o (wt%)	SO_4 (wt%)	S_t (wt%)
Head	50.00	8.04	0.080	0.040	0.32	0.21	0.28	1.53
D-1	47.93	5.75	0.050	0.014	0.04	0.28	0.21	1.46
D-2	47.04	3.51	0.047	0.016	0.05	0.61	0.23	1.64

D-3	47.01	3.19	0.011	0.014	0.04	0.67	0.59	1.43
D-4	45.34	1.19	0.008	0.010	0.03	0.65	0.38	1.20

Table 3. Leach Efficiencies

Test #	**Pb (%)**	**Zn (%)**	**Mn (%)**	**Ca (%)**
D-1	31.4	40.1	66.4	88.0
D-2	58.9	44.7	62.4	85.3
D-3	62.7	87.1	67.1	88.2
D-4	86.6	90.9	77.3	91.5

When leaching with concentrated (69 wt%) HNO3 (Tests D-3 and D-4), a reddish/brown gas was produced (likely being nitrogen dioxide (NO2)). Issues related to handling this gas will need to be addressed should concentrated HNO3 be considered as a leachant. A photo of the gas is shown in Figure 1.

A photo of the elemental sulphur (yellow material) produced via the oxidation of sulphides with HNO3 (Leach Test D-3) is shown in Figure 2.

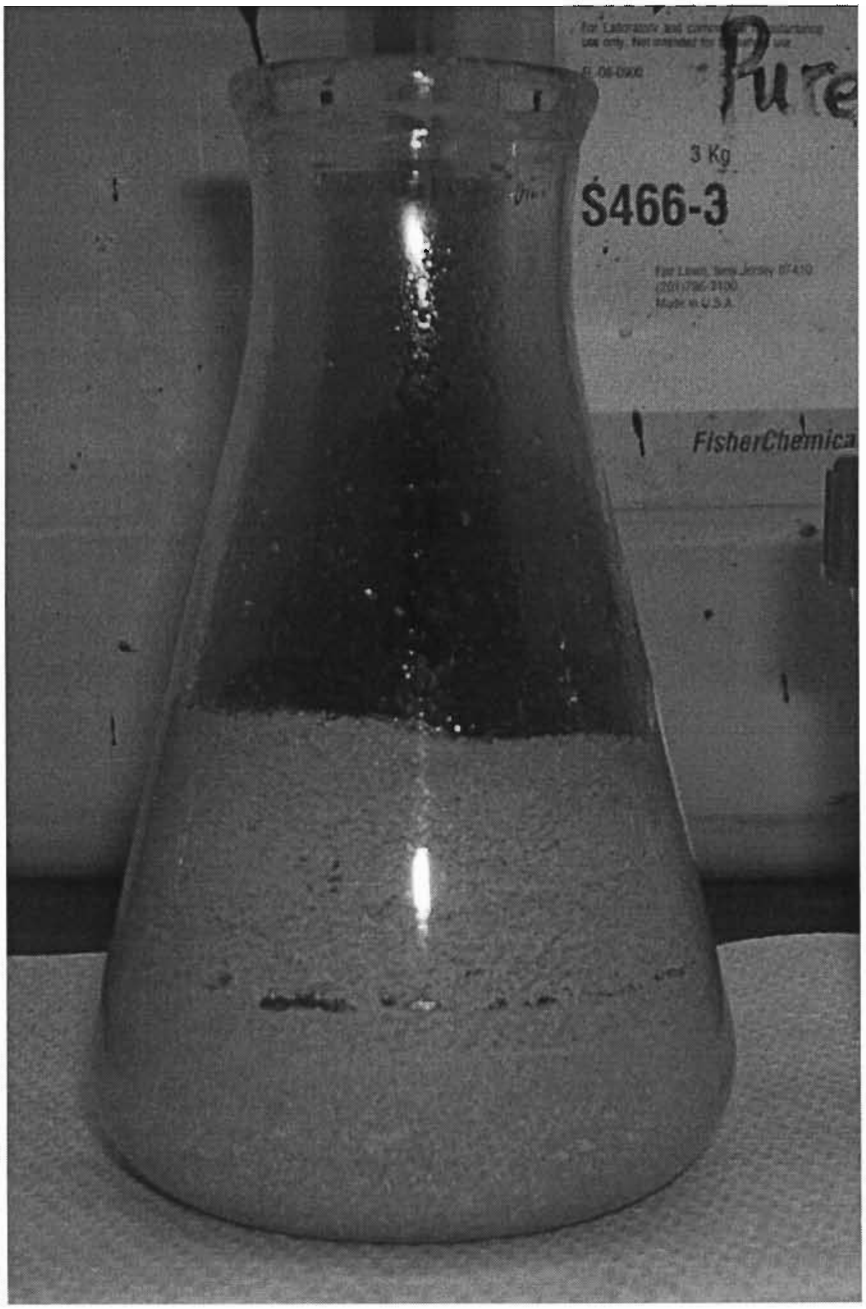


Figure 1. Reddish/Brown Gas Produced in Leach Test D-3

Figure 2. Elemental Sulphur (Yellow Material) Produced in Leach Test D-3

Leaching with HCl

One leach test with HCl was conducted on Yava ore (Sample LDRM2). The leach test was identified at Test # E-1. The leach test was conducted according to the procedure in Appendix A with the exception that concentrated 37 wt% HCl (equivalent to 10.75 M HCl) was used as the leachant and the test was conducted at a pulp density of 50 wt% solids for about 16 hours. A summary of the test conditions are shown in Table 4.

Table 4. HCl Leach Test Conditions

Test #	HCl Conc. (M)	Pulp Density (%)	Leach Time (Minutes)	Temperature (deg C)	Agitation
E-1	10.75	50%	960	25	Magnetic Stir Rod

ICP analysis and weight loss of the leach residues in comparison to the head sample (tabulated in Table 5) were used to calculate leach efficiencies for respective metals of interest (Table 6). Elemental sulphur (S^o), sulphate (SO_4) and total sulphur (S_t) analysis were conducted to see the effect of HCl (which in not oxidizing) on the sulphides and SO_4 production (Table 5). The decrease in S^o after leaching indicates that there is no oxidation of sulphides.

Table 5. ICP Metals, Sulphur Analysis and Weight Loss Data for Leach Residue for HCl Leach.

Test #	Residue Weight (g)	Pb (wt%)	Zn (wt%)	Mn (wt%)	Ca (wt%)	S^o (wt%)	SO_4 (wt%)	S_t (wt%)
Head	50.00	8.04	0.080	0.040	0.32	0.21	0.28	1.53
E-1	48.00	6.01	0.003	0.006	0.03	0.08	0.18	0.44

Table 6. Leach Efficiencies HCl Leach

Test #	Pb (%)	Zn (%)	Mn (%)	Ca (%)
E-1	28.2	96.4	86.9	91.0

During filtration and washing of the residue solids collected after leaching with concentrated (37 wt%) HCl, a strong odour of rotton eggs was observed (likely to be hydrogen sulphide (H2S)). In addition, a white precipitate was formed during the filtration process, suggesting that some of the metals were at the solubility limit. Although recoveries for Zn, Mn and Ca were good, the low recovery for Pb and H2S generation makes concentrated HCl and undesirable leachant.

Leaching CaCO3 and MnCO3 with HAc

A Leach test was conducted on CaCO3 and MnCO3 using an HAc leachant in excess. The purpose of this test is to examine the efficiency in which HAc (in the absence of H2O2) can leach Ca and Mn at the Yava deposit.

The leach test was conducted in a 500 mL beaker with a mixture of reagent grade chemicals containing 0.15 g of 99% purity CaCO3, 0.16 g of 97.08% purity MnCO3, 18.67 g of 99.7% purity HAc and 81.02 g of deionized water. The mixture was swirled for 20 minutes and then the solids were allowed to settle for 20 minutes. The solids were then filtered and the filtrate submitted for ICP analysis. The concentration of Ca in the filtrate was 663.96 mg/L and the Mn in the filtrate was 382.33 mg/L. This corresponds to a leach efficiency of 100% for Ca and 51% for Mn.

Pb(NO3)2 Precipitation Using (NH4)2CO3 to Produce PbCO3

Lead carbonate precipitation tests using reagent grade lead nitrate and ammonium carbonate in stoichiometric amounts were conducted to examine conversion rate and purity of PbCO3.

Reagent grade lead nitrate ($Pb(NO_3)_2$) and reagent grade ammonium carbonate ($(NH_4)_2CO_3$) was dissolved in 50-mL and 25-mL deionized water respectively. The amount of reagent grade lead nitrate and reagent grade ammonium carbonate was determined by the following chemical reaction:

$$Pb(NO_3)_2\ (aq) + (NH_4)_2CO_3\ (aq) \rightarrow PbCO_3\ (s) + 2NH_4NO_3\ (aq)$$

A target of 15-g $PbCO_3$ precipitate solids was used as the basis to calculate the stoichiometric amount of reagents required. The stoichiometric amounts of reagent grade lead nitrate and ammonium carbonate was calculated to be 19-g $Pb(NO_3)_2$ and 5-g $(NH_4)_2CO_3$.

Precipitation tests were carried out according to the following procedure:

1. Weigh 19-g of reagent grade lead nitrate into a 500-mL Pyrex beaker. Dissolve the lead nitrate solids with 50-mL of deionized water. Swirl beaker until the lead nitrate crystals are completely dissolved.
2. Weigh 5-g of reagent grade ammonium carbonate into a 500-mL Pyrex beaker. Dissolve the ammonium carbonate solids with 25-mL of deionized water. Swirl beaker until the ammonium carbonate crystals are completely dissolved.
3. Add the ammonium carbonate solution to the lead nitrate solution SLOWLY. Swirl beaker to ensure good mixing.
4. Filter white precipitate solids, thoroughly wash solids with deionized water and dry filter cake in the low temperature oven. Submit white precipitate solids for quantitative XRD analysis and retain filtrate solution for analysis, if requested.

The final weight of dry precipitate solids collected was approximately 12-g (80% conversion). The white precipitate was determined to be 100-wt% cerussite type $PbCO_3$ by quantitative XRD analysis (Rietveld Analysis).

Pb Ion Exchange Study

Beaker Testing of Purolite C-104H

Bench scale equilibrium rate loading tests for Pb were conducted using a Purolite C-104H ion exchange resin with a resin bulk volume/aqueous volume (R/A) ratio of 0.02. The Purolite C-104H resin was reported by SGS Lakefield to provide good selectivity for Pb when operated under overloaded conditions. Overloaded conditions indicate that the R/A ratio is kept low such that the resin will first load several different types of metal ions and then Pb will begin to displace non-Pb metal ions from active sites on the resin afterwards. Equilibrium rate loading tests were conducted at 25 deg C (Ambient), 40 deg C and 60 deg C to examine Pb loading kinetics as a function of temperature. The results plotted in Figure 3 shows that Pb loading kinetics for Purolite C-104H tends to improve with increasing temperature. Although loading kinetics appears slow for all temperatures. This is determined by the time in which the Pb concentration in the aqueous solution decreases until an constant (equilibrium) concentration is reached (indicating that the resin is fully loaded with Pb).

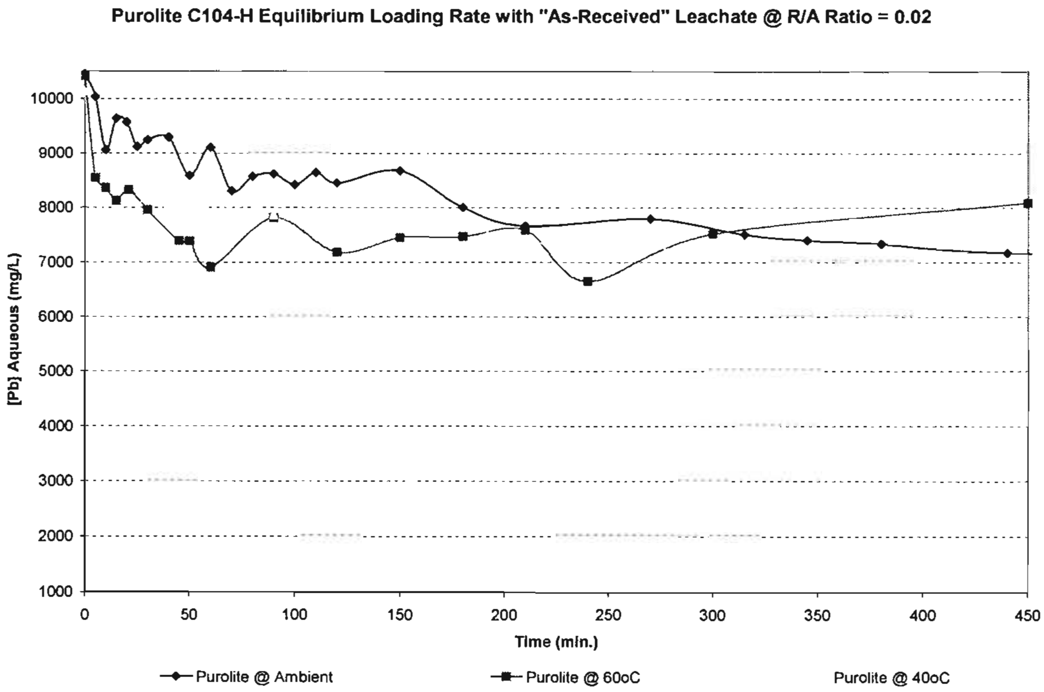


Figure 3. Bench Scale Purolite C-104H Pb Equilibrium Rate Loading

A series of bench scale stripping isotherm tests were performed to investigate the recovery of the Pb that had been pre-loaded onto Purolite C104H resin. The eluent used for stripping was 22-wt% HNO_3. This concentration was suggested by the resin manufacturer as the highest concentration that would not cause excessive wear of the resin while producing a Pb(NO3)2 strip liquor with this highest practical concentration. The conditions used in the stripping tests are summarized in Table 4.

Table 4. Experimental conditions during the stripping scope test

		Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**
Resin	mL	8	12	16	20
	Type	**Purolite**	**Purolite**	**Purolite**	**Purolite**
Aqueous	mL	200	200	200	200
R/A Ratio		0.04	0.06	0.08	0.10
Wt. Beaker	g	217	215	212	219
Wt. Leachate	g	208	208	208	208
Wt. Total Start	g	499	562	641	759
Wt. Total End	g	497	560	638	756
Temperature	°C	21	21	21	21
Time	hr.	4	4	4	4

The results of the stripping tests are summarized in Table 5. The loaded Pb was completely stripped under all conditions tested. Higher Pb concentrations were obtained with the higher R/A ratios (which is expected). An important observation is that a relatively high amount of Fe, Al and Ca reported to the strip liquor. This indicates that the Purolite C104H resin is not perfectly selective for Pb.

Table 5. Results of the Purolite C104H Stripping Tests

				Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**
	R/A Ratio		**Feed**	**0.04**	**0.06**	**0.08**	**0.10**
Pb	Resin	g/t	313600	6	5	6	5
	Aqueous	mg/L		38070	68510	100165	133651
	Stripping$_{Resin}$	%		**100.0%**	**100.0%**	**100.0%**	**100.0%**
Fe	Resin	g/t	11888	100	100	100	100
	Aqueous	mg/L		2353	4166	5801	3922
	Stripping$_{Resin}$	%		**99.2%**	**99.2%**	**99.2%**	**99.2%**
Al	Resin	g/t	1256	100	100	100	100
	Aqueous	mg/L		197.9	361.8	516.9	704.4
	Stripping$_{Resin}$	%		**92%**	**92%**	**92%**	**92%**
Cu	Resin	g/t	96	1	1	1	1
	Aqueous	mg/L		6.0	10.4	18.3	19.0
	Stripping$_{Resin}$	%		**99%**	**99%**	**99%**	**99%**
As	Resin	g/t	48	5	5	5	5
	Aqueous	mg/L		9.6	16.9	21.3	25.5
	Stripping$_{Resin}$	%		**90%**	**90%**	**90%**	**90%**
Sr	Resin	g/t	9	1	1	1	1
	Aqueous	mg/L		1.2	2.1	2.8	3.9
	Stripping$_{Resin}$	%		**89%**	**89%**	**89%**	**89%**
Mn	Resin	g/t	538	1	1	1	1
	Aqueous	mg/L		78	150	216	297
	Stripping$_{Resin}$	%		**100%**	**100%**	**100%**	**100%**
Zn	Resin	g/t	84	1	1	1	1
	Aqueous	mg/L		7.5	12.9	20.3	23.8
	Stripping$_{Resin}$	%		**99%**	**99%**	**99%**	**99%**
Ca	Resin	g/t	4696	100	100	100	100
	Aqueous	mg/L		722	1388	1977	2706
	Stripping$_{Resin}$	%		**98%**	**98%**	**98%**	**98%**
Cr	Resin	g/t	36	1	1	1	1
	Aqueous	mg/L		5.0	7.4	7.0	4.3
	Stripping$_{Resin}$	%		**97%**	**97%**	**97%**	**97%**
Na	Resin	g/t	684	100	100	100	100
	Aqueous	mg/L		9.0	7.0	7.0	9.0
	Stripping$_{Resin}$	%		**85%**	**85%**	**85%**	**85%**

Column Testing of Purolite C-104H

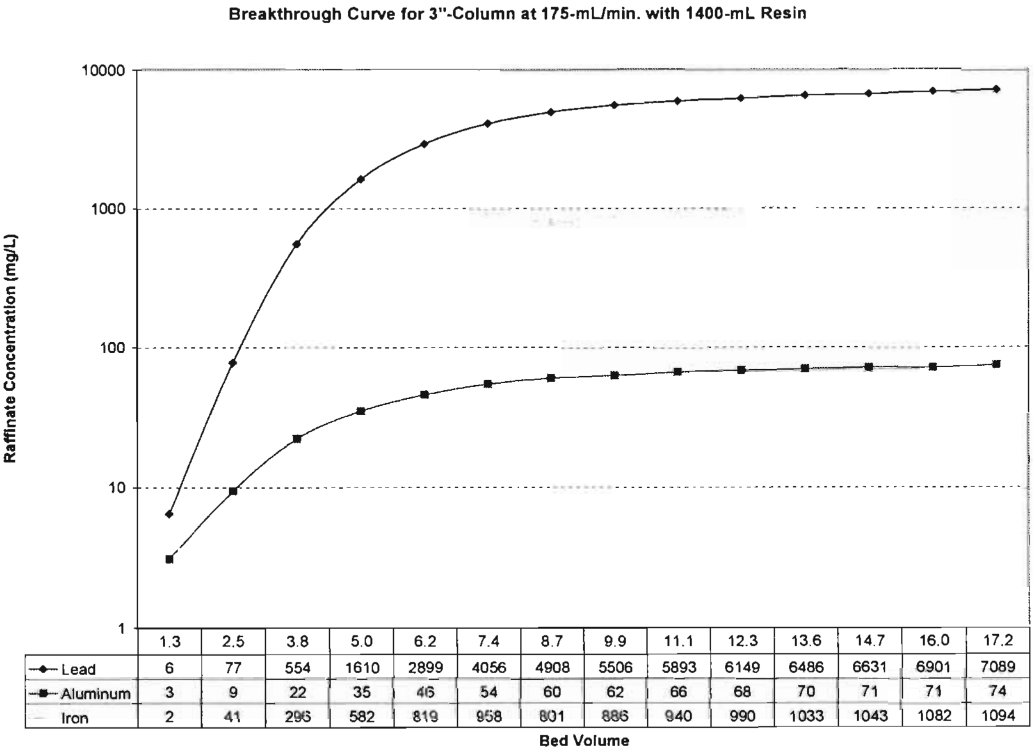
Pilot scale Pb ion exchange loading tests were conducted with a 3" column using Purolite C-104H resin. The test conditions are as follows:

Flow Rate Target = 175 mL/min and 765 mL/min
Resin Volume = 1400 mL
Leachate Volume @ Start= 25 L
Column Diameter = 7.727 cm
Column Height = 2.5 ft
Bed Depth = 29.85 cm
Cross Sectional Area = 46.89 cm^2
Bed Volume = 1400 mL
SG Wet Resin = 1.12
Feed Soln. SG = 1.02
Fill Head Space Height = 15.24 cm
BV Height @ Start = 31.274cm
BV Height @ End = 29.369 cm

The test procedure for Pb ion exchange column testing is as follows:

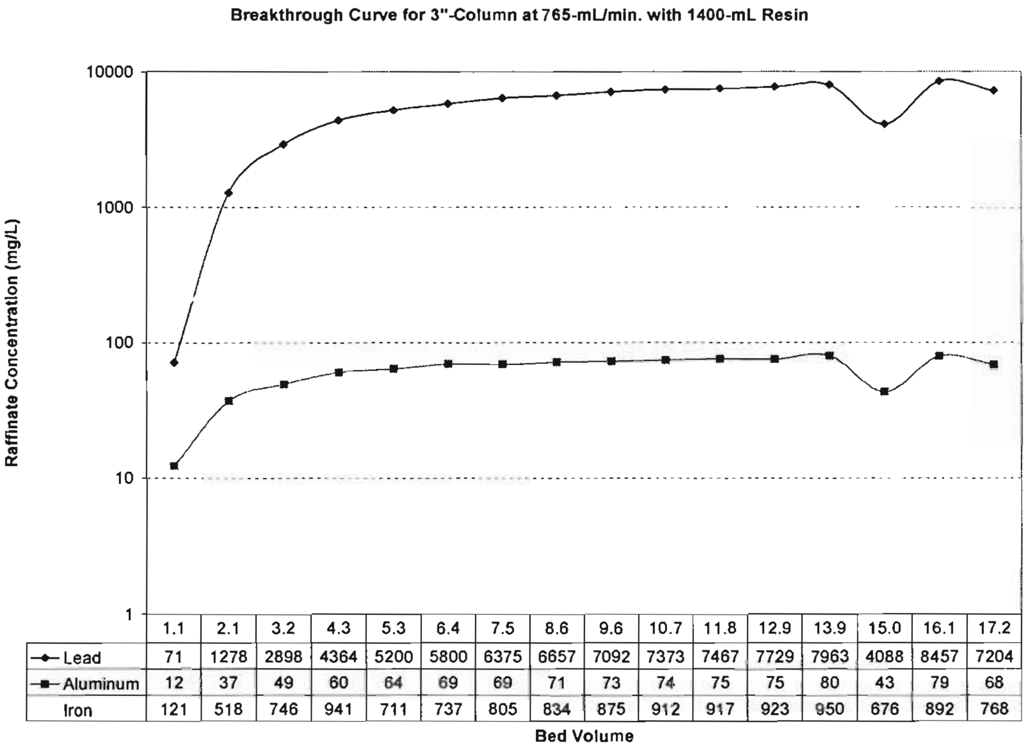
1) Pre-measure the desired volume of wet resin in a graduated cylinder.
2) Partially fill column with deionized water and transfer resin into the column with DI water.
3) Backflush/Stratification of resin column with deionized water.
4) Allow resin to naturally settle in column and mark bed volume (BV)
5) Drain column of excess deionized water to just above the BV mark.
6) Insert top splash/screen disc.
7) Calibrate pump to match desired flow rate and fill head space with feed solution (approx. 6" above BV mark).
8) Mark top of head space solution level and use this as the flow control level.
9) Start feed pump and timer. Control column discharge valve to match the flow control level.
10) Collect small aliquots at the intervals indicated below and record weight of the feed solution.
11) Record final BV level in the column and measure total raffinate weight. Sample raffinate and use it as feed for cycle 2.
12) Start cycle 2 at step 7 and repeat until 6 cycles have been completed.

The breakthrough curves (1st cycle) for the tests conducted at 175 ml/min and 765 ml/min are plotted in Figures 4 and 5 respectively. Figure 6 superimposes the breaktrhough curves for Pb at 175 ml/min and 765 ml/min and shows that a lower flow rate improves Pb loading.



	1.3	2.5	3.8	5.0	6.2	7.4	8.7	9.9	11.1	12.3	13.6	14.7	16.0	17.2
◆ Lead	6	77	554	1610	2899	4056	4908	5506	5893	6149	6486	6631	6901	7089
■ Aluminum	3	9	22	35	46	54	60	62	66	68	70	71	71	74
Iron	2	41	296	582	819	958	801	886	940	990	1033	1043	1082	1094

Figure 4. Breakthrough curve 175mL/min



	1.1	2.1	3.2	4.3	5.3	6.4	7.5	8.6	9.6	10.7	11.8	12.9	13.9	15.0	16.1	17.2
◆ Lead	71	1278	2898	4364	5200	5800	6375	6657	7092	7373	7467	7729	7963	4088	8457	7204
■ Aluminum	12	37	49	60	64	69	69	71	73	74	75	75	80	43	79	68
Iron	121	518	746	941	711	737	805	834	875	912	917	923	950	676	892	768

Figure 5. Breakthrough curve – 765 ml/min

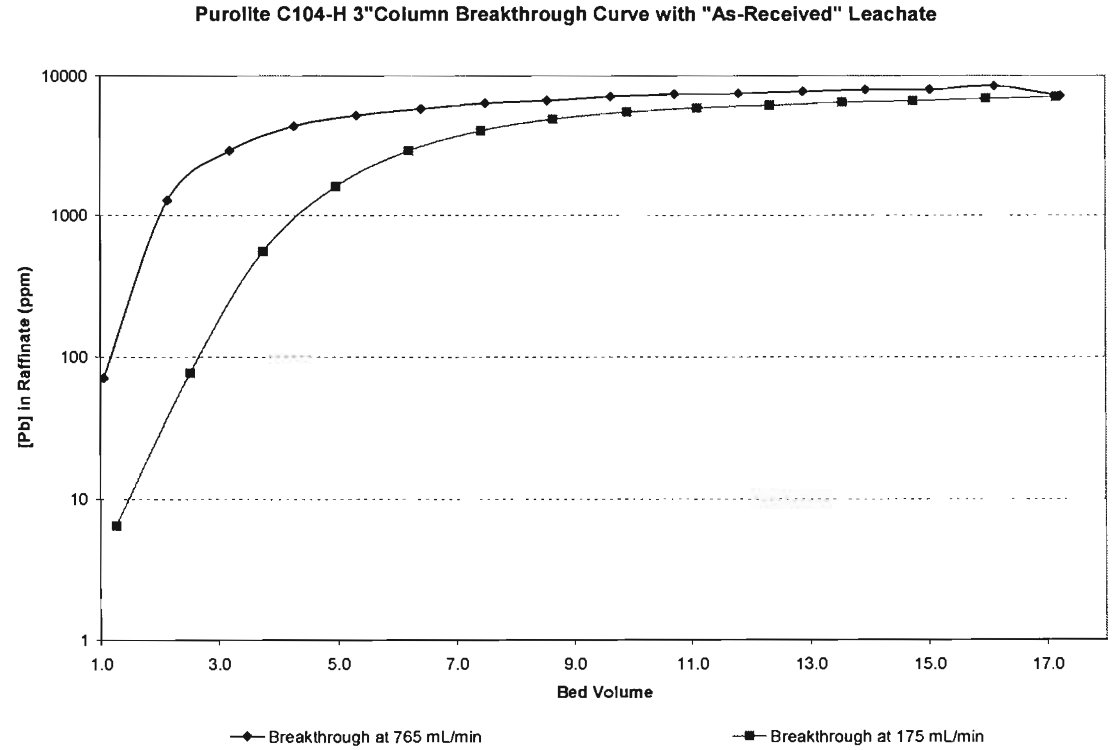


Figure 6. Comparison between the breakthrough curves of Pb at 765 mL/min and 175 mL/min

Beaker Testing of Eichrom Monophos and Diphonix Ion Exchange Resins

The Yava requirements for metals separation were discussed with Eichrom Inc. (a supplier of ion exchange resins). Two resins were provided for testing (Monophos and Diphonix). Bench scale testing was conducted with these ion exchange resins with the intent to examine loading capacity and selectivity under different R/A ratios. The test procedure is as follows:

1) Scoop an adequate amount of each resin into separate 500 mL beakers and wash thoroughly with deionized water. Decant water and fill beaker near the top. Allow resin to soak overnight.
2) Weight and label 6 clean and dry Pyrex beakers.
3) Measure appropriate amount of wet settled resin into their respective beakers, decant.
4) Measure 400 mL of Pb-Ion exchange raffinate solution into each beaker. Put aside a 10 mL sample of this solution into a sample container to be analyzed for metals by ICP.
5) Place the six beakers into a gang mixer and measure the initial pH (and temperature) of each beaker. Allow to mix for 24 hours.
6) After 24 hours, measure pH (and temp) of each beaker. Stop the gang mixer and remove.
7) Filter resin in Buchner funnel with filter paper and collect a sample of each filtrate for analysis. Store remaining filtrate in 500mL sample bottles and label.
8) Thoroughly wash the resin with deionized water and collect a sample of the resin for analysis. Store remaining resin in sample containers submerged in deionized water. Place the resin sample in a low temperature oven for at least 24 hours.
9) Submit raffinate, resin and filtrate samples for metals analysis by ICP.

A summary of the test conditions are show in Table 6.

Table 6. Monophos/Diphonix Pb Ion Exchange Test Conditions

		Beaker #11	**Beaker #12**	**Beaker #13**	**Beaker #14**	**Beaker #15**	**Beaker #16**
Resin Type	Type	**Monophos**	**Monophos**	**Monophos**	**Diphonix**	**Diphonix**	**Diphonix**
Resin	mL	8	16	48	8	16	48
	g	10	20	59	10	20	59
Aqueous	mL	400	400	400	400	400	400
	g	416	416	416	416	416	416
R/A Ratio		**0.02**	**0.04**	**0.12**	**0.02**	**0.04**	**0.12**
Initial pH		3.6	3.6	3.6	3.6	3.6	3.6
Final pH		3.6	3.6	3.6	3.6	3.6	3.7
Temp.	°C	22	22	22	22	22	22
Time	Hr	24	24	24	24	24	24

The results of the testing are summarized in Table 7.

Table 7. Monophos/Diphonix loading experiments – ICP results

				Beaker #11	Beaker #12	Beaker #13	Beaker #14	Beaker #15	Beaker #16
			Resin	**Monophos**	**Monophos**	**Monophos**	**Diphonix**	**Diphonix**	**Diphonix**
	R/A Ratio		**Feed**	**0.02**	**0.04**	**0.12**	**0.02**	**0.04**	**0.12**
Pb	**Resin**	g/t		77500	77000	76800	73200	73700	72500
	Aqueous	mg/L	8700	8000	7300	4500	8100	7300	4600
	Loaded $_{soln}$	%		**8.0%**	**16.1%**	**48.3%**	**6.9%**	**16.1%**	**47.1%**
	[Pb] Resin $_{Soln.}$	mg/L		35000	35000	35000	30000	35000	34167
Mn	**Resin**	g/t		2242	2029	1784	2286	2073	1750
	Aqueous	mg/L	249.8	226.4	205.7	138.4	231.1	208.7	139.8
	Loaded $_{soln}$	%		**9.3%**	**17.6%**	**44.6%**	**7.5%**	**16.5%**	**44.0%**
Fe	**Resin**	g/t		17463	16407	7080	16571	15377	7240
	Aqueous	mg/L	1215	917	580	26.1	920	598	26.6
	Loaded $_{soln}$	%		**24.5%**	**52.3%**	**97.9%**	**24.3%**	**100.0%**	**100.0%**

The results in Figure 7 show that both the Monophos and Diphonix are not specific for Pb. Under all conditions tested the Monophos and Diphonix resins have a higher affinity for Mn and Fe compared to Pb. As such, further testing with these two resins were abandoned.

Sulphide Precipitation Study

A bulk sulphide precipitation test was conducted on Yava leachate to examine the effectiveness in which various metal ions (contained in Yava leachate) can be precipitated with H2S. The sulphide precipitation step in the Yava process is conducted after the recovery of Pb, Zn and Mn to precipitate heavy, rare earth and other metals leaving behind a high purity Ca solution conductive to the production of precipitated calcium carbonate (PCC).

A 500-mL of "as-received" cartridge filtered Yava leachate was measured into a 1000-mL Pyrex Erlenmeyer flask and H_2S gas was gently sparged while swirling the flask for approximately 5 minutes in a fume hood. The Erlenmeyer flask was left to stand in the fume hood with the top open for about 16 hours. The solution was then sparged with nitrogen gas from approximately 60 minutes to remove any lingering H2S. The solution was then filtered in a buchner funnel. The precipitate was predominantly black and the filtrate was colourless and clear.

The assay results and percent precipitation are summarized in Table 8,

Table 8. Assay Results and Percent Precipitation of Various Metals using H2S on Yava Leachate

Element	Leachate (mg/L)	Filtrate (mg/L)	% ppt. (%)	Element	Leachate (mg/L)	Filtrate (mg/L)	% ppt. (%)
Al	92	88	4%	Mn	284	293	0%
Sb	0.05	0.05	0%	Hg	0.02	0.02	0%
As	3.1	0.03	99%	Mo	0.01	0.01	0%
Ba	0.01	0.44	0%	Ni	3.4	0.52	84%
Be	0.04	0.06	0%	P	0.10	0.10	0%
Bi	0.01	0.01	0%	K	37	39	0%
B	0.01	1.9	0%	Se	0.26	0.05	81%
Cd	0.01	0.01	0%	Si	69	66	4%
Ca	1869	1999	0%	Ag	0.02	0.02	0%
Cr	10.8	10.4	3%	Na	23.7	24.9	0%
Co	0.73	0.35	52%	Sr	1.76	1.72	0%
Cu	3.3	0.17	95%	Tl	0.20	0.20	0%
Fe	927	1135	0%	Sn	0.10	0.10	0%
Pb	3979	0.05	100%	Ti	0.01	0.01	0%
Li	0.20	0.20	0%	W	0.10	0.10	0%
Mg	783	814	0%	V	0.01	0.01	0%
				Zn	18.4	0.62	97%

3

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Project No. 4040751

Progress Report No 3

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC. DURING JULY 2005

Prepared by

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and
Clive Brereton ,Alan Keylock (Noram Engineering and Constructors Ltd)

INTRODUCTION

The work carried out in the month of July 2005 focused on removal of iron (Fe) from Yava leachate using solvent extraction, manganese (Mn) separation studies using ion exchange and lead hydroxide (Pb(OH)2)/ lead carbonate (PbCO3) precipitation studies to examine conversion efficiency.

Reported in Yava Pilot Plant Report No. 2 dated June 2005, Pb separation using Purolite C104H ion exchange resin resulted in Fe contamination in the Pb(NO3)2 strip liquor. As such, the removal of Fe prior to Pb ion exchange is required to produce a high purity Pb(NO3)2 solution. Solvent extraction test work was conducted on using a Cyanex 272 extractant supplied by Cytec Industries.

Mn separation studies were conducted with Eichrom's Monophos and Diphonix ion exchange resins (as suggested by the manufacturer). Solutions used for the Mn ion exchange studies were produced using a three stage Pb ion exchange removal process using a Purolite C104H resin

Precipitation experiments were conducted by precipitating reagent grade Pb(NO3)2 with sodium hydroxide (NaOH) to produce Pb(OH)2, and sodium carbonate (Na2CO3) to produce PbCO3. This testwork was conducted to examine the conversion efficiency as a function of stoichiometric ratio of NaOH or Na2CO3.

SUMMARY

1. Cyanex 272 was capable of removing Fe from Yava leachate with a single stage solvent extraction treatment with a low R/A ratio of 0.5 and pH of 2.25.

2. Selective Mn Ion Exchange conducted on Yava Pb depleted leachate was unsuccessful using Eichrom's Monophos and Diphonix ion exchange Resins.

3. Pb(OH)2 precipitation efficiency improves with increasing stoichiometric ratio of NaOH with respect to Pb(NO3)2 up to 100%. Above an NaOH stoichiometric ratio 100%, Pb begins to re-dissolve into solution.

4. PbCO3 precipitation efficiency improves with increasing stoichiometric ratio of Na2CO3 up to 120%.

RESULTS AND DISCUSSION

Solvent Extraction Study

Removal of Fe using Cyanex 272

Solvent extraction experiments were conducted with the intention of removing Fe from Yava leachate prior to Pb ion exchange.

The first set of solvent extraction experiments consisted of roughly determining the selectivity of the Cyanex 272 extractant as a function of pH (in order to estimate the desired operating pH). Test work was conducted at a fixed O/A ratio, however, it should be noted that O/A ratio may also affect selectivity particularly when the O/A ratio is low resulting in certain metal ions displacing other ones when the organic extractant is operated closer to its capacity. The test conditions for the initial solvent extraction experiments are summarized in Table 1.

Table 1. Summary of Initial Solvent Extraction Experiments for Fe Removal

	Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**
Extractant	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272
Feed pH	Natural	3.0	2.5	2.0	1.5	1.0
Organic	80-mL	80-mL	80-mL	80-mL	80-mL	80-mL
Aqueous	60-mL	60-mL	60-mL	60-mL	60-mL	60-mL
O/A Ratio	4/3	4/3	4/3	4/3	4/3	4/3
pH Adjust	None	50%-HNO_3	50%-HNO_3	50%-HNO_3	50%-HNO_3	50%-HNO_3

The procedure for testing selectivity with solvent extraction is follows:

1. Prepare an organic mixture containing a 10 volume % Cyanex 272 extractant in Isopar M diluent.
2. Prepare a sufficient volume of 50 wt% HNO3 solution to be used for pH adjustment
3. Prepare separate beakers of Yava leachate adjusted to the desired pH for the desired number of experiments.
4. Fill separate separatory funnels with the desired volume of organic and aqueous solutions for each pH tested.
5. Close the stopcock valves and plug the ends of each separatory funnel and shake each funnel aggressively for 20 minutes.
6. Place each mixed separatory funnel on a stand and take note of the time in which phase break appears in the aqueous or the organic and the time in which complete phase separation occurs. Note whether there are any precipitates or stable emulsions after separation.
7. Open the stop cock valve and drain the aqueous phase into a container and submit sample for metals analysis by ICP
8. Pour the organic out of the top of the separatory funnel into a container and submit sample for metals analysis by ICP.

The results of the Cyanex 272 selectivity testing is summarized on the graph in Figure 1.

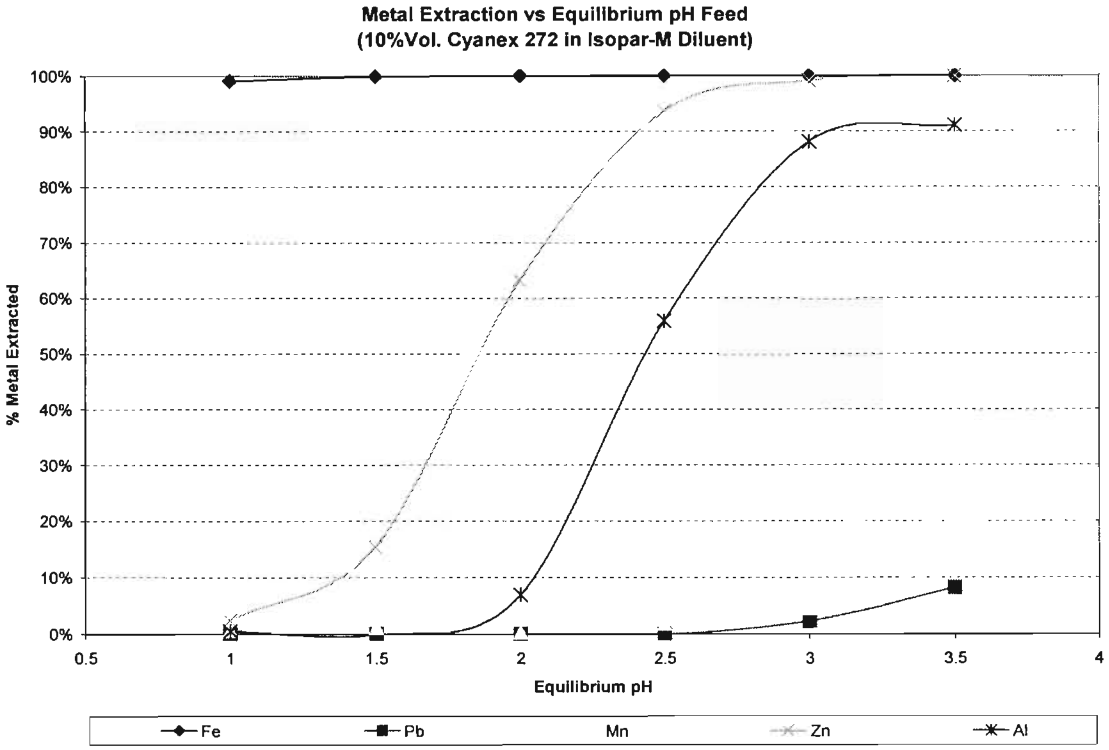

Figure 1.

The removal of Fe with Cyanex 272 was efficient at all pH's tested (in the range of pH 1.0 to 3.5). At pH above 1.0, Zn begins to load onto the extractant. At pH above 1.75, Al begins to load. At pH above 2.25, Mn begins to load and at pH above 2.5 Pb begins to load. Based on the selectivity chart in Figure 1, a pH of 2.25 would be a practical pH to conduct Fe removal by solvent extraction. At this pH, the loading of Mn and Pb would be negligible while acid consumption to lower the pH from the natural pH of 3.5 to 2.25 would be minimized (at the expense of loading Zn).

Experimental work was conducted to generate an equilibrium isotherm for Fe loading by solvent extraction. The goal is to determine practical operating conditions that minimize organic usage and number of extraction stages while efficiently removing Fe. Consideration is also given to selectivity for Fe as a function of O/A ratio and other observations that may affect practical operation (such as high viscosities, precipitation, etc.). Fe equilibrium isotherm experiments were conducted at O/A ratios of 0.5, 1.0, 1.3, 1.7, 2.0 and 3.0. The pH was 2.25 for all experiments.

The results of the Fe loading equilibrium isotherm test work are plotted in Figure 2 with data showing selectivity of Fe and Zn summarized in Table 2.

APPENDIX B

Equipment Procured to Conduct Yava Pilot Study

Qnty.	Unit	Item Description
2	each	Easy-Load II Pump Head (SS Rotor)
2	each	L/S Variable Speed Modular Drive (6-600 rpm)
1	box	L/S 15-Neoprene, High Performance Precision Pump Tubing
1	box	L/S 24-Neoprene, High Performance Precision Pump Tubing
1	box	L/S 35-Neoprene, High Performance Precision Pump Tubing
3	each	7"x7" - Vela Magnetic Stirring Hot Plates (38-371°C)
4	each	1-1/2" x 3/8" Mono Mold Stir Bars
4	each	1-1/2" x 3/8" Mono Mold Stir Bars (with Spin Ring)
1	each	Stir-Pak Speed Controller
1	each	Stir-Pak Mixer Head (9-900 rpm)
1	each	Stir-Pak Mixing Shaft, SS, (3/8"-dia. x 30"-L)
1	each	Stir-Pak Propellers, SS, (3"-dia.)
1	each	Stir-Pak Double Clamp
1	each	Stir-Pak Support Stand (29"-H x 5/8"-dia.)
1	each	PIG Filter Cart
		Single Cartridge Filter with Air Operated Pump on Cart
1	each	Big Blue Dual Cartridge Filter Housing (20")
2	box	1-micron, Housing Replacement Filters, PP, 20"-L, 20 each/box
1	box	10-micron, Housing Replacement Filters, PP, 20"-L, 20 each/box
12	each	25-1 micron Dual-Gradient Density Cartridge, 20"-L
1	each	0.025-0.25 GPM Polysulfone In-Line Flowmeter, PVC Float
1	each	0.1-1.0 GPM Polysulfone In-Line Flowmeter, SS Float, PM
1	each	0.2-2.0 GPM Polysulfone In-Line Flowmeter, SS Float, PM
1	each	Microprocessor Controlled Submersible Sump Pump
1	box	10-mL B-D Disposable Syringes (with Luer-Lock Tips)
1	box	60-mL B-D Disposable Syringes (with Luer-Lock Tips)
1	pkg.	Graduated 4-oz Sample Containers, PP (500 each/pkg.), Nonsterile
2	pkg.	1-Quart Square Tall Bottles with Screw Caps, HDPE (36 each/pkg.)
40	each	20-L Ropak Round Plastic Containers (Std.White)
40	each	20-L Ropak T5FT Cover for Dry and Liquid Seals (EZ Cover)
15	each	FOH-200 55-gallon Blue Polyethylene Open Head Drums
1	each	OHAUS Champ Deluxe Bench Scale (250-kg Capacity)
1	each	Sartorius Basic Portable Balance (6000-g Capacity)
1	each	1/4" Thermoplastic Air Hose, 50-ft with 1/4"-NPT (M) Ends
2	each	1/4"-Brass Female Pipe Thread Nipple 1/4"-NPT(F)
2	each	1/4"-Brass Female Pipe Thread Coupler 1/4"-NPT(F)
100	ft	3/8"-ID x 5/8"-OD Clear Vinyl Tubing
100	ft	1/2"-ID x 3/4"-OD Clear Vinyl Tubing

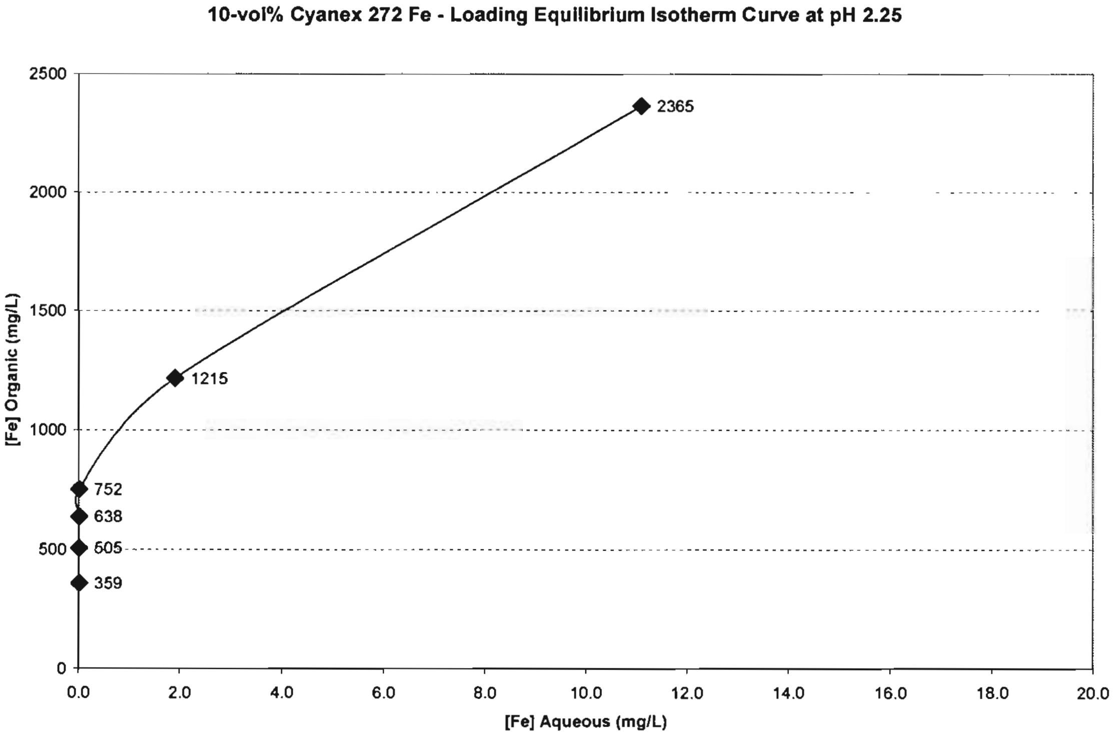

Figure 2. Fe Loading Equilibrium Isotherm with Cyanex 272.

Table 2. Fe Loading Equilibrium Isotherm Data with Cyanex 272.

			Feed	**Funnel #1**	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**
	O/A Ratio			**0.5**	**1.0**	**1.3**	**1.7**	**2.0**	**3.0**
Fe	**Organic**	mg/L		2365	1215	752	638	505	359
	Aqueous	mg/L	1202	11.1	1.90	0.01	0.01	0.01	0.01
	Loaded $_{Aq.}$	%		**99.1%**	**99.8%**	**100.0%**	**100.0%**	**100.0%**	**100.0%**
Zn	**Organic**	mg/L		15.0	12.3	7.8	7.0	5.6	4.2
	Aqueous	mg/L	15.2	15.0	12.86	5.58	0.95	0.10	<0.01
	Loaded $_{Aq.}$	%		**1.6%**	**15.6%**	**63.4%**	**93.8%**	**99.3%**	**99.9%**

The information in Figure 2 and Table 2 shows that a low O/A ratio (0.5) is capable of removing most of the Fe from the aqueous solution is a single stage extraction. Another interesting observation is that the selectivity for Fe over Zn improves with the low O/A ratio. This is likely due to the displacement of Zn with Fe by operating closer to the loading capacity of the organic extractant.

Mn Separation Studies Using Eichrom Monophos and Diphonix Ion Exchange Resins

Eichrom Inc, suggested that their Monophos and Diphonix ion exchange resins may be useful in selectively removing Mn from Yava's leachate that has be pre-treated to remove Pb. As such, bench scale ion exchange test work was conducted with these two resins.

Prior to testing, a representative Pb depleted leachate was produced by conducting a three stage batch Pb removal process using a Purolite C104H ion exchange resin. This Pb depleted leachate was used as the feed solution for the Mn ion exchange test work. The test conductions are summarized in Table 3, while the test procedure was as follows:

1. Scoop an adequate amount of each resin into separate 500 mL beakers and wash thoroughly with deionized water. Decant the wash water and refill the beaker with fresh DI water. Allow the resin to soak overnight.
2. Weigh and label six clean and dry Pyrex beakers.
3. Measure appropriate amount of wet settled resin into their respective beakers.
4. Measure 400 mL of Pb depleted leachate into each beaker. Put aside a 10 mL sample of this solution into a sample container for metals analysis by ICP.
5. Place six beakers into a gangue mixer and measure the initial pH and temperature of each beaker. Allow to mix for 24 hours.
6. After 24 hours, measure pH and temperature of each beaker. Stop the gangue mixer and remove beakers. Make note of any colour change in each beaker.
7. Filter resin with a Buchner funnel and collect a sample of each filtrate for analysis. Store remaining filtrate in labeled sample bottles.
8. Thoroughly wash the resin with deionized water and collect a sample of the resin for analysis. Place the resin sample in a low temperature oven for at least 24 hours. Store remaining resin in labeled sample containers, submerged in deionized water.
9. Submit raffinate, resin and filtrate for metals analysis by ICP.

Table 3. Mn Ion Exchange Test Conditions

		Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**	**Beaker #5**	**Beaker #6**
Resin Type	Type	**Monophos**	**Monophos**	**Monophos**	**Diphonix**	**Diphonix**	**Diphonix**
Resin	mL	8	16	48	8	16	48
Aqueous	mL	400	400	400	400	400	400
R/A Ratio		0.02	0.04	0.12	0.02	0.04	0.12
Initial pH		3.6	3.6	3.6	3.6	3.6	3.6
Final pH		3.6	3.6	3.6	3.6	3.6	3.6
Temperature	°C	22	22	22	22	22	22
Time	Hr	24	24	24	24	24	24

The results of the Mn ion exchange experiments are summarized in Table 4. For low R/A ratios (0.02 in Beakers #1 and #4 for Monophos and Diphonix respectively) no Mn was loaded onto the resin. Mn loading increases with increasing R/A ratios for both resins. However, neither resin exhibited selectivity for Mn.

Table 4. Results of Mn Ion Exchange Experiments with Monophos and Diphonix.

				Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**	**Beaker #5**	**Beaker #6**
			Feed	**Monophos**	**Monophos**	**Monophos**	**Diphonix**	**Diphonix**	**Diphonix**
[redacted]	**Resin**	g/t		10492	6812	2560	10591	6818	2600
	[Pb] Aqueous	mg/L	160	58	36	15	59	32	15
	Loaded $_{soln}$	%		**63.8%**	**77.5%**	**90.6%**	**63.1%**	**80.0%**	**90.8%**
[redacted]	**Resin**	g/t		15511	11353	5119	15446	11364	4766
	Aqueous	mg/L	184.87	185	150	51	185	120	54
	Loaded $_{soln}$	%		**0.0%**	**18.9%**	**72.4%**	**0.0%**	**35.1%**	**70.8%**
	Loading Capacity $_{Soln.}$	mg/L		0	872	1116	0	1622	1091
[redacted]	**Resin**	g/t		397	227	192	397	249	194
	Aqueous	mg/L	6.63	0.03	0.78	0.03	0.78	0.78	0.79
	Loaded $_{soln}$	%		**99.5%**	**88.2%**	**99.5%**	**88.2%**	**88.2%**	**88.1%**
[redacted]	**Resin**	g/t		430	323	254	451	328	270
	Aqueous	mg/L	9	5.4	6.1	6.4	5.8	5.8	6.8
	Loaded $_{soln}$	%		40.0%	32.2%	28.9%	35.6%	35.6%	24.4%
[redacted]	**Resin**	g/t		10084	9980	8056	10936	9815	7680
	Aqueous	mg/L	590.2	529.3	492.3	424.6	531.6	486.9	430.8
	Loaded $_{soln}$	%		10.3%	16.6%	28.1%	9.9%	17.5%	27.0%
[redacted]	**Resin**	g/t		355	297	142	390	301	143
	Aqueous	mg/L	9.72	6.25	4.43	1.68	6.21	4.12	1.88
	Loaded $_{soln}$	%		35.7%	54.4%	82.7%	36.1%	57.6%	80.7%

Pb Precipitation Studies as Hydroxides and Carbonates

Precipitation experiments were conducted by precipitating reagent grade Pb(NO3)2 with sodium hydroxide (NaOH) to produce Pb(OH)2, and sodium carbonate (Na2CO3) to produce PbCO3. This test work was conducted to examine the conversion efficiency as a function of stoichiometric ratio of NaOH or Na2CO3.

Pb(OH)2 Precipitation Studies

The precipitation of Pb(OH)2 is conducted by mixing reagent grade Pb(NO3)2 with NaOH solution per Reaction (1).

$$\mathrm{Pb(NO3)2 + NaOH \rightarrow Pb(OH)2 + 2NaNO3} \qquad (1)$$

Experiments were conducted by mixing solutions containing Pb(NO3)2 and NaOH at stoichiometric ratios of 90%, 100%, 110%, 120% and 150% NaOH. Conversion efficiency was examined by analyzing the filtrate solution after precipitation and filtration of the solids. The results summarized in Table 5 show that more Pb re-dissolves at increasing stoichiometric NaOH ratios greater than 100%. X-Ray Diffraction was attempted on the solid material collected from the 100% stoichiometric NaOH experiment showed that the material was quite amorphous which is consistent with Pb(OH)2.

Table 5. Results of the Pb(OH)2 Precipitation Studies

Stoichiometric Amount of NaOH	**90%**	**100%**	**110%**	**120%**	**150%**
Filtrate [Pb]	21 ppm	N/A	2600 ppm	2700 ppm	6800 ppm
Final pH	10.2	12.5	12.8	12.9	13.2

PbCO3 Precipitation Studies

The precipitation of PbCO3 is conducted by mixing reagent grade Pb(NO3)2 with Na2CO3 solution per Reaction (2).

$$\text{Pb(NO3)2 + Na2CO3} \rightarrow \text{PbCO3 + 2NaNO3} \qquad (2)$$

Experiments were conducted by mixing solutions containing Pb(NO3)2 and Na2CO3 at stoichiometric ratios of 90%, 100%, 110%, 120% and 150% Na2CO3. Conversion efficiency was examined by analyzing the filtrate solution after precipitation and filtration of the solids. The results summarized in Table 6 show that more PbCO3 precipitates with increasing stoichiometric amount of Na2CO3 up to 120%. X-Ray Diffraction conducted on the solid material collected from the 100% stoichiometric Na2CO3 experiment showed that the material was 100% PbCO3.

Table 6. Results for the PbCO3 Precipitation Studies

Stoichiometric Amount of Na2CO3	**90%**	**100%**	**110%**	**120%**	**150%**
Filtrate [Pb]	13000 ppm	250 ppm	10 ppm	9 ppm	17 ppm
Final pH	4.7	5.9	8.6	8.6	8.7

4

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Project No. 4040751

Progress Report No 4

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC.

DURING AUGUST 2005

Prepared by
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and
Clive Brereton , Alan Keylock (Noram Engineering and Constructors Ltd)

INTRODUCTION

The work carried out in the month of August 2005 focused on Pb separation studies using solvent extraction and ion exchange. Pb(OH)2 and PbCO3 precipitation studies were also conducted to examine filterability of the solids.

As reported in Yava Pilot Plant Report No. 2 dated June 2005, Fe loading was observed on ion exchange resins tested for Pb separation. As such, future Pb separation studies are conducted with Yava leachate with Fe removed by solvent extraction with a Cyanex 272 extractant. Yava Pilot Plant Report No. 2 also reported that the Purolite C104H ion exchange resin had promising Pb selectivity when operated in overloaded conditions. One drawback of the Purolite C104H resin was the slow kinetics for Pb loading encountered when operating in overloaded conditions (due to the requirement of Pb displacing other metal ions that have loaded onto the resin). In an attempt to improve Pb separation kinetics, Pb solvent extraction studies were conducted with a Cyanex 301 extractant and ion exchange studies conducted with a Bayer TP 207 resin.

Precipitation of Pb as Pb(NO3)2 to produce Pb(OH)2 using NaOH, and produce PbCO3 using Na2CO3 were examined with the goal of examining filterability of the solids.

SUMMARY

1. Pb separation by solvent extraction using a Cyanex 301 extractant resulted in promising Pb selectivity with respect to loading. Pb stripping experiments on Cyanex 301 will be conducted at a later date.
2. Pb separation by ion exchange using a Bayer TP 207 ion exchange resin resulted in promising Pb selectivity with respect to loading. The manufacturer indicated that an improvement of results may be obtained by first acid treating the TP 207 resin to produce a hydrogen form. This will be conducted at a later date.
3. Precipitation experiments were conducted to produce Pb(OH)2 and PbCO3. In general, the PbCO3 is less gelatinous than the Pb(OH)2 with the exception when Pb(OH)2 is

precipitated at pH 13 (which results in a non-gelatinous, easy to filter precipitate) or when PbCO3 is precipitated with a high excess of Na2CO3 (greater than 110% stoichiometric) which results in a more gelatinous precipitate.

RESULTS AND DISCUSSION

Pb Separation Study

Solvent Extraction Using Cyanex 301 to Examine Pb Loading

The Yava process was discussed with Cytec Industries with the intent of seeking an extractant with high Pb selectivity and good loading kinetics. Cytec recommended testing of their Cyanex 301 product which is used in the separation of Ni but has been observed by Cytec to be selective for heavy metals at low pH (ph < 2.0).

Solvent extraction loading studies were carried out with Cyanex 301 extractant at pH 0.7, 1.0, 1.3, 1.6, 2.0 and 2.3 with the goal of examining selectivity for Pb. The feed solutions used for this study consisted of Yava leachate that had been treated to remove most of the Fe using a Cyanex 272 extractant. Fe removal was conducted at pH 2.25 such that removal of Pb and Mn are minimized. The test conditions for the Cyanex 301 solvent extraction experiments are summarized in Table 1.

Table 1. Summary of Pb Solvent Extraction Experiments with Cyanex 301

	Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**
Extractant	10.0 v.% Cyanex 301	10.0 v.% Cyanex 301	10.0 v.% Cyanex 301	10.0 v.% Cyanex 301	10.0 v.% Cyanex 301	10.0 v.% Cyanex 301
Feed pH	2.3	2.0	1.6	1.3	1.0	0.7
Organic	90-mL	90-mL	90-mL	90-mL	90-mL	90-mL
Aqueous	60-mL	60-mL	60-mL	60-mL	60-mL	60-mL
O/A Ratio	3/2	3/2	3/2	3/2	3/2	3/2
pH Adjust	None	50%-HNO_3	50%-HNO_3	50%-HNO_3	50%-HNO_3	50%-HNO_3

The procedure for Pb removal with Cyanex 301 including Fe removal by Cyanex 272 is as follows:

1. Prepare 4000-mL of a 10% by volume Cyanex 272 extractant in Isopar M diluent.
2. Prepare 1000-mL of a 10% by volume Cyanex 301 extractant in Isopar M diluent.
3. Prepare 150-g of a 50-wt.% HNO3 solution in a clearly labeled sample container for pH adjustment.
4. Pre-measure 4000-mL of cartridge filtered Yava leachate in a 5-L HDPE vessel equipped with an overhead mixer and impeller.
5. Start the overhead mixer and adjust the pH of the Yava leachate solution to 2.25 using the 50wt.% HNO3 solution. Take a sample of the solution for analysis.
6. Using a 2-L separatory funnel, conduct several batch shake out tests with the 10 vol.% Cyanex 272 at an O/A Ratio of 4/3 to produce approximately 4000-mL of aqueous raffinate solution identified as Fe removed Yava leachate.
7. Using a graduated cylinder, measure separately 80-mL of Fe removed Yava leachate into six clearly labeled sample containers. Store remaining Fe removed Yava leachate for future testing.
8. Adjust the pH of the pre-measured Fe removed Yava leachate to the specified pH listed in Table 1 using the 50-wt.% HNO3 solution.
9. Using a graduated cylinder, carefully measure 60-mL of the respective pH adjusted Fe removed Yava leachate into separate clearly labeled sample containers. Retain the 20-mL of each pH adjusted Fe removed Yava leachate for potential analysis.

10. Weigh, record and label six clean and dry 250-mL Pyrex separatory funnels with stop cocks and valves.
11. Measure 90-mL of the freshly prepared Cyanex 301 containing organic solution using a graduated cylinder. Close the valve of the respective separatory funnels and carefully transfer the pre-measured Cyanex 301 solution into each separatory funnel. Measure and record the total weight of the separatory funnel.
12. Carefully transfer 60-mL of pH adjusted Fe removed Yava leachate into each respective separatory funnel containing the freshly prepared Cyanex 301 solution.
13. Place the fitted stop cock in the separatory funnel and shake aggressively for 20 minutes. Place the separatory funnel into a stand and take note of the time for initial phase break to appear in the organic or aqueous, and time to complete separation of both phases. Take note whether there are any precipitates or stable emulsions after separation, as well as describe the colour and clarity both phases.
14. Remove the stop cock and drain the aqueous phase by opening the bottom valve of the separatory funnel, into separate clean and labeled 100-mL polypropylene sample containers. Measure and record the total weight of the separatory funnel after draining the aqueous phase. Submit aqueous samples for metals analysis by ICP, and Pb and Fe assay by atomic adsorption (AA)
15. Pour the organic phase out of the top of the separatory funnel into a clean and labeled 100-mL polypropylene sample container. Submit organic solution samples for metals analysis by ICP, and Pb and Fe assay by AA.

A photograph of the experimental setup is shown in Figure 1.

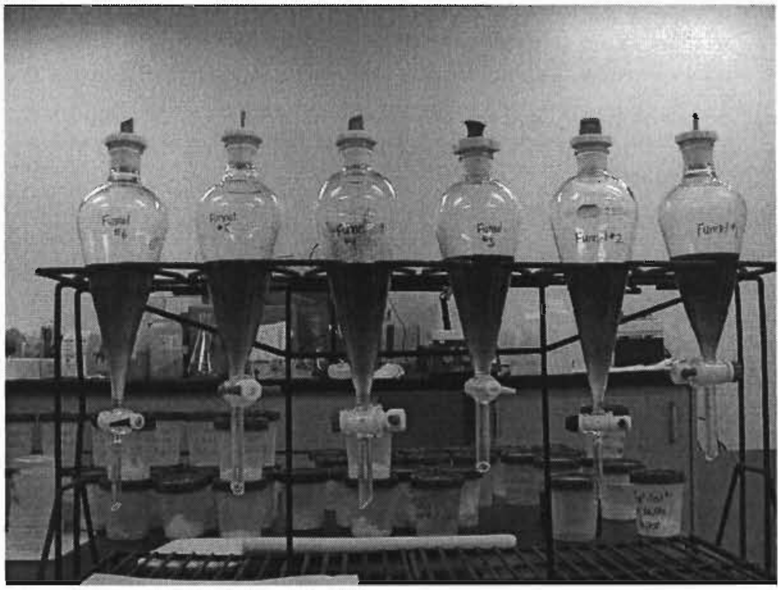

Figure 1. Loaded 10-vol.% Cyanex 301 in Isopar-M diluent, Funnel #1 to Funnel #6, from left to right (aqueous raffinate has been removed – organic phase is green).

The analytical results of the Pb loading test work with Cyanex 301 are summarized in Table 2. The test results show that Cyanex 301 is high specific for Pb against Mn, Mg, Cu and Ca. 100% of the Pb in the initial feed solutions was loaded onto the Cyanex 301 extractant whereas 0% of the Mn, Mg, Cu and Ca were loaded (with the exception of 7% and 3% of the Cu at pH 2.3 and 2.0 respectively). Fe, Zn and Ni were also loaded onto the Cyanex 301, however, the concentration of these metals in the feed solution are extremely low in comparison to the Pb. As such, Cyanex 301 shows good promise as a potential extractant to selectively separate Pb to produce a high purity Pb product.

Stripping tests will be conducted in the future to examine the ease in which Pb can be removed from the loaded Cyanex 301 extractant.

Table 2. Analytical Results of Pb Separation Test Work Using Cyanex 301

				Funnel#1	Funnel#2	Funnel#3	Funnel#4	Funnel#5	Funnel#6
	pH		Feed Av.	**2.3**	**2.0**	**1.6**	**1.3**	**1.0**	**0.7**
Pb	**Organic**	mg/L		5670	1508	4366	2693	4796	4752
	Aqueous	mg/L	8200	2.53	0.39	0.23	0.22	0.20	0.16
	Loaded $_{Aq.}$	%		**100%**	**100%**	**100%**	**100%**	**100%**	**100%**
Fe	**Organic**	mg/L		7.19	7.74	9.97	7.96	6.45	4.95
	Aqueous	mg/L	6.85	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
	Loaded $_{Aq.}$	%		**100%**	**100%**	**100%**	**100%**	**100%**	**100%**
Mn	**Organic**	mg/L		0.20	0.23	0.09	0.09	0.05	0.05
	Aqueous	mg/L	242	244	242	243	245	250	247
	Loaded $_{Aq.}$	%		**0%**	**0%**	**0%**	**0%**	**0%**	**0%**
Zn	**Organic**	mg/L		1.46	1.48	1.68	1.73	1.41	1.33
	Aqueous	mg/L	2.35	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
	Loaded $_{Aq.}$	%		**100%**	**100%**	**100%**	**100%**	**100%**	**100%**
Al	**Organic**	mg/L		28.4	30.5	23.3	20.5	15.9	8.3
	Aqueous	mg/L	69.5	14.7	20.2	30.9	37.8	47.8	55.5
	Loaded $_{Aq.}$	%		**79%**	**71%**	**56%**	**46%**	**31%**	**20%**
Mg	**Organic**	mg/L		<0.1	<0.1	1.0	1.0	<0.1	0.6
	Aqueous	mg/L	668	675.8	669.3	669.4	675.4	689	677.7
	Loaded $_{Aq.}$	%		**0%**	**0%**	**0%**	**0%**	**0%**	**0%**
Cu	**Organic**	mg/L		1.89	1.86	2.17	2.38	1.85	1.84
	Aqueous	mg/L	2.81	2.60	2.72	2.80	2.81	2.84	2.81
	Loaded $_{Aq.}$	%		**7%**	**3%**	**0%**	**0%**	**0%**	**0%**
Ca	**Organic**	mg/L		16.1	10.1	15.1	14.5	7.7	6.0
	Aqueous	mg/L	1540	1559	1547	1550	1561	1589	1569
	Loaded $_{Aq.}$	%		**0%**	**0%**	**0%**	**0%**	**0%**	**0%**
Ni	**Organic**	mg/L		1.63	1.41	1.75	1.75	1.61	1.44
	Aqueous	mg/L	2.75	0.39	<0.02	0.31	0.29	0.31	0.32
	Loaded $_{Aq.}$	%		**86%**	**99%**	**89%**	**89%**	**89%**	**88%**

Pb Separation by Ion Exchange Using Bayer TP207 Resin

The Yava process was discussed with Bayer Corporation with the intent of seeking an ion exchange resin with high Pb selectivity and good loading kinetics. Bayer recommended testing of their TP 207 ion exchange resin. For clarity, since Bayer Corporation had conducted numerous acquisitions the TP 207 resin was also represented by Lanxess Corp, Lewatit and Sybron Chemicals.

Pb separation studies were conducted with Bayer TP207 ion exchange resins using Fe removed Yava leachate as the feed solution. Bench scale testing examined the loading characteristics as a function of resin to aqueous bulk volume ratios (R/A ratios). The test conditions are summarized in Table 3.

Table 3. Test Conditions for Bayer TP 207 Ion Exchange Test Work for Loading of Pb

		Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**	**Beaker #5**	**Beaker #6**
Resin Type	Type	TP207	TP207	TP207	TP207	TP207	TP207
Resin	mL	8	12	16	20	32	48
Aqueous	mL	400	400	400	400	400	400
R/A Ratio		**0.02**	**0.03**	**0.04**	**0.05**	**0.08**	**0.12**
Initial pH		2.3	2.5	2.7	2.8	3	3.1
Final pH		2.5	2.6	2.8	3.0	3.2	3.3
Temperature	°C	22	22	22	22	22	22
Time	Hr	24	24	24	24	24	24

The procedure for conducting Pb load tests using Bayer TP 207 ion exchange resin is as follows:

1. Scoop an adequate amount of TP 207 resin into separate 500 mL beakers and wash thoroughly with deionized water. Decant water and fill beaker near the top. Allow resin to soak overnight.
2. Weigh and label six clean and dry 500-mL Pyrex beakers.
3. Measure appropriate amount of wet settled resin into each respective beaker and decant excess water. Using a 3-M acetic acid stock solution, rinse the resin and decant excess acetic acid solution.
4. Measure 400 mL of Fe removed Yava leachate into each beaker. Set aside 10-mL sample of the Fe removed Yava leachate for analysis.
5. Place the six beakers into a gangue mixer and measure the initial pH of each beaker. Allow to mix for 24 hours.
6. After 24 hours, measure pH of each beaker. Stop the gangue mixer and remove beakers. Make note of any colour changes in each beaker.
7. Filter resin in Buchner funnel with filter paper and collect a sample of each filtrate for analysis. Store remaining filtrate in separate 1000-mL sample bottles.
8. Thoroughly wash the resin with deionized water and collect a sample of the resin for analysis. Place the resin sample in a low temperature oven for at least 24 hours. Store remaining resin in separate sample containers submerged in deionized water.
9. Submit feed solution, raffinate filtrate and dry resin samples for metals analysis.

A photograph of the experimental set-up is shown in Figure 2.

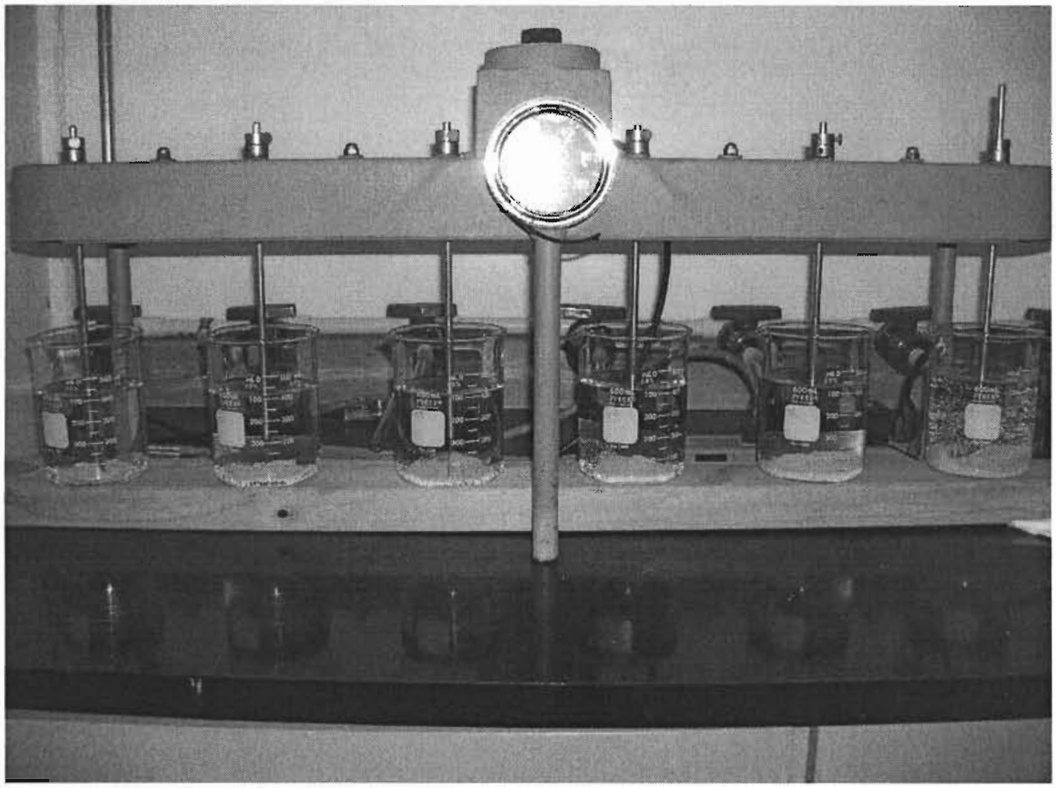

Figure 2. Experimental Set-up for Pb Separation Test Work Using Bayer TP 207 Ion Exchange Resin

The results of the Pb load test using the Bayer TP 207 ion exchange resin are summarized on Table 4. Selectivity for Pb can be determined by comparing the difference between percent metals loaded onto the ion exchange resin (identified at Loaded %) in comparison to other metal ions present in the Yava leachate. Selectivity for Pb is particularly important against other metal ions that are present in higher concentrations in the Yava leachate.

According to the results in Table 4, Bayer TP 207 resin shows promising selectivity for Pb over Mn, Mg and Ca. The selectivity improves when operating in conditions of higher loading (i.e. lower R/A ratio). At an R/A ratio of 0.02 a negligible amount of Mn, Mg and Ca loaded onto the ion exchange resin. At a higher R/A ratio of 0.12 Pb was 1.9 (=96.1%/49.7%), 6.3 and 3.1 times more selective than Mn, Mg and Ca respectively. The TP 207 was not Pb selective over Fe, Zn and Ni. However, these metal ions are present in the feed solution in low concentrations and therefore will have minimal effect on final Pb solution purity. Selectivity for Pb over Al and Cr improves with lower R/A ratio.

On interesting observation was the high Na that reported to the treated raffinate solution. This "negative loading" is likely caused by having a large amount of Na present on the "as received" TP 207 resin. The manufacturer acknowledged this to be the case and has recommended an acid pretreatment step to convert the TP 207 ion exchange resin from sodium (Na) form to hydrogen (H) form. The manufacturer also suggested that the TP 207 performance may improve with this treatment step. Test work with will be conducted at a later date with H form TP 207.

Table 4. Results of Pb Separation Test Work with TP 207 Ion Exchange

	R/A Ratio		Feed	0.02	0.03	0.04	0.05	0.08	0.12
[redacted]	Resin	g/t		252500	252600	247300	237500	199200	148300
	[Pb] Aqueous	mg/L	8401	5457	4154	2943	2144	820	325
	Loaded	%		**35.0%**	**50.6%**	**65.0%**	**74.5%**	**90.2%**	**96.1%**
	[Pb] Resin	g/L		147.2	43.4	30.3	16.0	16.5	4.1
[redacted]	Resin	g/t		5	6	19	62	772	1800
	Aqueous	mg/L	250	249	244	239	235	199	126
	Loaded	%		**0.2%**	**2.0%**	**4.1%**	**6.0%**	**20.3%**	**49.7%**
[redacted]	Resin	g/t		593	368	293	240	183	119
	Aqueous	mg/L	6.62	1.13	0.82	0.61	0.56	0.41	0.26
	Loaded	%		**82.9%**	**87.6%**	**90.8%**	**91.5%**	**93.8%**	**96.1%**
[redacted]	Resin	g/t		232	200	341	557	1131	1027
	Aqueous	mg/L	68.1	66.6	62.2	59.1	51.4	22.7	8.9
	Loaded	%		**2.2%**	**8.7%**	**13.2%**	**24.5%**	**66.7%**	**86.9%**
[redacted]	Resin	g/t		290	176	134	106	84	57
	Aqueous	mg/L	2.78	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
	Loaded	%		**99.6%**	**99.6%**	**99.6%**	**99.6%**	**99.6%**	**99.6%**
[redacted]	Resin	g/t		<100	<100	<100	<100	177	1800
	Aqueous	mg/L	683.6	688.6	675.3	671.6	667.9	645.7	579.5
	Loaded	%		**0.0%**	**1.2%**	**1.8%**	**2.3%**	**5.5%**	**15.2%**
[redacted]	Resin	g/t		187	128	102	86	66	44
	Aqueous	mg/L	2.27	0.57	0.2	0.1	0.05	<0.01	<0.01
	Loaded	%		**74.9%**	**91.2%**	**95.6%**	**97.8%**	**99.6%**	**99.6%**
[redacted]	Resin	g/t		276	<100	125	191	1934	6009
	Aqueous	mg/L	1592.6	1588	1564.5	1538.9	1524.2	1399.3	1095.3
	Loaded	%		**0.3%**	**1.8%**	**3.4%**	**4.3%**	**12.1%**	**31.2%**
[redacted]	Resin	g/t		59	73	83	93	102	81
	Aqueous	mg/L	6.00	5.27	4.57	3.92	3.29	1.64	0.93
	Loaded	%		**12.2%**	**23.8%**	**34.7%**	**45.2%**	**72.7%**	**84.5%**
[redacted]	Resin	g/t		196	151	113	93	64	45
	Aqueous	mg/L	2.67	0.51	0.32	0.27	<0.02	0.29	0.27
	Loaded	%		**80.9%**	**88.0%**	**89.9%**	**99.3%**	**89.1%**	**89.9%**
[redacted]	Resin	g/t		724	256	267	253	245	298
	Aqueous	mg/L	49	660	990	1455	1875	2952	4260
	Loaded	%		-1247%	-1920%	-2869%	-3727%	-5924%	-8594%

Pb Precipitation Studies as Hydroxides and Carbonates to Examine Filterability

Precipitation experiments were conducted by precipitating reagent grade $Pb(NO_3)_2$ with sodium hydroxide (NaOH) to produce $Pb(OH)_2$, and sodium carbonate (Na_2CO_3) to produce $PbCO_3$. This test work was conducted to examine filterability of the precipitates ($Pb(OH)_2$ versus $PbCO_3$). $Pb(OH)_2$ precipitation test work was conducted at 6 target pH's ranging from 8 to 13. $PbCO_3$ test work was conducted in according to the procedures reported in Yava Pilot Plant Report No. 3 dated July 2005. Filterability observations of $PbCO_3$ precipitates are reported below for comparison purposes.

Pb(OH)2 Precipitation Studies

The precipitation of $Pb(OH)_2$ is conducted by mixing reagent grade $Pb(NO_3)_2$ with NaOH solution per Reaction (1).

$$Pb(NO_3)_2 + NaOH \rightarrow Pb(OH)_2 + 2NaNO_3 \qquad (1)$$

Experiments were conducted by metering 5 M NaOH into solutions containing 219 g/L $Pb(NO_3)_2$ and observing the characteristics of the precipitates as a function of pH. The test procedure for this test work is as follows:

1. Weigh 13.8 g of reagent grade $Pb(NO_3)_2$ into a sample container. Add 60 mL deionized water and mix with a magnetic stir bar until $Pb(NO_3)_2$ crystals are completely dissolved.
2. Prepare 5 M NaOH stock solution for use with a pH controller
3. Calibrate the pH meter and take an initial reading of the $Pb(NO_3)_2$ solutions. Slowly add the 5 M NaOH solution in small increments to the $Pb(NO_3)_2$ solution while gently mixing with a stir bar until approximately the target pH is reached. Separate tests are to be conducted to approximately pH 8, 9, 10, 11, 12 and 13.
4. Observe precipitate formed and make note of precipitate characteristics related to filterability.
5. Filter precipitate solids in a Buchner funnel, thoroughly wash with deionized water, dry filter cake in a low temperature oven. Retain precipitate solids and filtrate samples for potential analysis.

The $Pb(OH)_2$ precipitation test results are summarized in Table 5. The target pH in which $Pb(OH)_2$ is precipitated plays an important role in the filterability of the precipitates. Precipitating at a low pH (pH 8) results in a gelatinous precipitate that is more difficult to filter. This gelatinous precipitate is observed up to pH 12. Interestingly at pH 13, a green/brown precipitate is formed that is very easy to filter. Photographs of the $Pb(OH)_2$ precipitation experiment at pH 13 is shown in Figure 3.

Table 5. Pb(OH)2 Precipitation Test Results

Target pH	**8**	**9**	**10**	**11**	**12**	**13**
Wt. $Pb(NO_3)_2$ (g)	13.8	13.8	13.8	13.8	13.8	13.8
Wt. 5 M NaOH (g)	11.8	14.0	16.8	16.8	17.2	25.6
Initial pH	3.2	3.3	3.3	3.3	3.5	3.5
Final pH	7.81	9.11	10.64	11.25	12.1	13.14
Precipitation Time	30 - min.	45 - min.	60 - min.	15 - min.	15 - min.	15 - min.
ppt observed	Immediate	Immediate	Immediate	Immediate	Immediate	Immediate
Wt $Pb(OH)_2$ ppt.	10.5	10.8	9.8	9.8	10.0	8.7
Filtrate Pb (ppm)	76	26.8	1	1	46	3473
Notes	milky white ppt	milky white ppt	milky white ppt	initial ppt white, final ppt lime /olive green	initial ppt white, final ppt lime /olive green	initial ppt white, final ppt green / brown

Figure 3. Precipitation of Pb(OH)2 at pH 13 with increasing time (going right).

PbCO3 Precipitation Studies

Precipitation experiments conducted by mixing reagent grade Pb(NO3)2 with Na2CO3 solution to produce PbCO3 were conducted and reported in Yava Pilot Plant Report No. 3 dated July 2005 with the exception that filterability results are reported below. Test results and observations of the precipitates are summarized on Table 6. Precipitation of PbCO3 initially results in a gelatinous precipitate, however, for the low Na2CO3 experiments (less than 110% stoichiometric Na2CO3) the gelatinous nature of the precipitates decrease over time as gas (suspected of being CO2 is released). With high excess Na2CO3 (above 110% stoichiometric Na2CO3), the PbCO3 precipitate remains gelatinous.

Table 6. PbCO3 Precipitation Test Results and Observations.

Stoichiometric Amount of Na2CO3	**90%**	**100%**	**110%**	**120%**	**150%**
Wt. $Pb(NO_3)_2$ (g)	18.6	18.6	18.6	18.6	18.6
Wt. Na_2CO_3 (g)	5.4	6.0	6.6	7.2	9.0
Initial pH	3.5	3.6	3.5	3.6	3.6
Final pH	4.7	5.9	8.6	8.6	8.7
Retention Time	10 - min.	10 - min.	10 - min.	10 - min.	10 - min.
ppt observed	Immediate	Immediate	Immediate	Immediate	Immediate
Wt $PbCO_3$ ppt.	13.8	15.4	15.2	15.4	15.9
Notes	gelatinous white ppt @ first then milky white, gas released	gelatinous white ppt @ first then milky white, gas released	gelatinous white ppt @ first then milky white, gas released	more gelatinous white ppt to milky white, no gas released	more gelatinous white ppt to milky white, no gas released

In comparison, the PbCO3 precipitates produced at stoichiometric amounts less than 110% are less gelatinous Pb(OH)2 precipitates produced at less than pH 12. However, the Pb(OH)2 precipitate produced at pH 13 was not gelatinous and easy to filter.

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Project No. 4040751

Progress Report No 5

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC. DURING SEPTEMBER 2005

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INTRODUCTION

The work carried out in the month of September 2005 focused on improving the removal of Fe and Zn from Yava leachate by solvent extraction, Pb separation studies using solvent extraction and producing a sample of Pb(NO3)2 crystal from Yava leachate.

As reported in Yava Pilot Plant Report No. 4 dated August 2005, both Cyanex 301 organic extractant and Bayer TP 207 ion exchange resin showed good selectivity for Pb against Mn, Mg and Ca. Cyanex 301 and Bayer TP 207 were not selective against Fe or Zn. As such, solvent extraction studies were conducted with Cyanex 272 with the intent of optimising the removal of Fe and Zn prior to Pb separation.

Pb stripping test work was conducted on loaded Cyanex 301. Pb stripping was examined using two different HNO3 concentrations to examine the efficiency in which Pb can be removed from loaded Cyanex 301 to recover Pb(NO3)2 solution of high purity.

A sample Pb(NO3)2 crystals was produced from Yava leachate for evaluation purposes.

SUMMARY

1. The conditions for removal of Fe and Zn from Yava leachate by solvent extraction with a Cyanex 272 extractant were optimised. The operating conditions are pH 2.8 of the aqueous feed and an O/A ratio of 0.5

2. Pb stripping tests using 0.5 and 1.5 M HNO3 on loaded Cyanex 301 were unsuccessful. Pb adsorbed strongly to the Cyanex 301 extractant and could not be removed with HNO3 under the conditions tested. The results will be discussed with the extractant manufacturer for comments.

3. A Pb(NO3)2 crystal sample was successfully produced from Yava leachate.

RESULTS AND DISCUSSION

Fe and Zn Removal Study

Solvent Extraction Using Cyanex 272 to Optimise Removal of Fe and Zn

Test work was conducted on using solvent extraction with a Cyanex 272 extractant to remove Fe and Zn prior to Pb separation. Prior studies have shown that the Cyanex 301 extractant and Bayer TP 207 ion exchange resin exhibit promising Pb selectivity against Mn, Mg and Ca but poor selectivity against Fe and Zn. As such, effective removal of Fe and Zn prior to Pb separation with either Cyanex 301 or Bayer TP 207 will improve the purity of the final Pb product.

As reported in Yava Pilot Plant Report No. 3 dated July 2005, Cyanex 272 is capable of removing Fe and Zn with minimal removal of Mn or Pb. A pH of approximately 2.8 for the aqueous feed solution was shown to achieve removal of Fe and Zn with low removal of Pb and Mn. Solvent extraction experiments were conduced with Cyanex 272 using pH 2.8 aqueous feed solutions at seven different O/A ratios (0.050, 0.075, 0.125, 0.250, 0.500, 1.000 and 1.500) in order to optimise Fe and Zn removal with minimal removal of Mn and Pb. Prior studies only focused on the removal of Fe prior to Pb separation. Although, Yava Pilot Plant Report No. 3 does show that Fe and Zn removal can be effectively achieved with a pH 2.25 aqueous feed solution at a high O/A ratio of 3. The drawback of this operation condition is the high use of organic extractants.

The test conditions for optimising Fe and Zn removal from Yava leachate are summarized on Table 1.

Table 1. Test Conditions for Optimising Fe and Zn Removal from Yava Leachate using Solvent Extraction

	Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**	**Funnel #7**
Extractant	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272	10.0v.% Cyanex 272
Feed pH	2.8	2.8	2.8	2.8	2.8	2.8	2.8
Organic	10-mL	15-mL	20-mL	30-mL	60-mL	80-mL	90-mL
Aqueous	200-mL	200-mL	160-mL	120-mL	120-mL	80-mL	60-mL
O/A Ratio	0.050	0.075	0.125	0.250	0.500	1.000	1.500

The test procedure for optimising Fe and Zn removal from Yava leachate using solvent extraction is as follows:

1. Prepare a 10% by volume Cyanex 272 extractant in Isopar M diluent.
2. Prepare 100-g of a 50-wt.% HNO3 solution in a clearly labeled sample container for pH adjustment.
3. Pre-measure 1000-mL of "as-received" cartridge filtered Yava leachate solution into a Pyrex beaker containing a magnetic stir rod.
4. Place the beaker on a magnetic stir plate and adjust the pH of the pre-measured leachate solution to 2.8 using the 50-wt.% HNO3 solution. Measure/record the final pH, ORP and amount of acid used to achieve desired pH for the 1000-mL leachate.
5. Using a graduated cylinder, measure pH adjusted leachate solution into the clearly labeled sample containers. Store remaining pH adjusted solution in a 1-L sample bottle for future reference, if required.
6. Weigh, record and label six clean and dry 250-mL Pyrex separatory funnels with stop cocks and valves.
7. Measure the organic solution using a graduated cylinder as specified in the table below. Close the valve of the respective separatory funnels and carefully transfer the pre-measured organic solution into the separatory funnel. Measure and record the total weight of the separatory funnel.
8. Carefully transfer the pH adjusted aqueous solution as specified below into each separatory funnel containing the respective organic solution.
9. Place the fitted stop cock in the separatory funnel and shake aggressively for 20 minutes. Place the separatory funnel into a stand and take note of the time for initial phase break to appear in the organic or aqueous, and time to complete separation of both phases. Note whether there are any precipitates or stable emulsions after separation, as well as describe the colour and clarity both phases.
10. Remove the stop cock and drain the aqueous phase using the bottom valve on the separatory funnel into separate clean and labeled 100-mL polypropylene sample containers. Measure and record the total

weight of the separatory funnel after draining the aqueous phase. Measure and record the pH/ORP of the aqueous solution and submit aqueous samples for ICP metals (H2O method), Pb assay (AA/ICP) and Fe assay (AA/ICP) analyses.

11. Pour the organic phase from the top of the separatory funnel into a clean and labeled 100-mL polypropylene sample container. Submit organic solution for ICP metals (H2O method), Pb assay (AA/ICP) and Fe assay (AA/ICP) analyses.

The results of this test work are summarized in Table 2 and selectivity for Fe and Zn against Mn and Pb are plotted on Figure 1. Based on these results an O/A ratio of approximately 0.5 (while operating with an aqueous feed of pH 2.8) resulted in the optimal selectivity of Fe and Zn against Pb and Mn. At lower O/A ratios, selectivity for Zn decreases and at higher O/A ratios small amounts of Mn and Pb begin to load onto the Cyanex 272 extractant. On another note, at O/A ratios of less than 0.25, the high concentration of Fe loaded into the organic phase results in high viscosity which could cause operational problems.

Table 2. Analytical Results for Fe and Zn Removal Tests Using Cyanex 272

				Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**	**Funnel #7**
	O/A Ratio		**Feed**	**0.050**	**0.075**	**0.125**	**0.250**	**0.500**	**1.000**	**1.500**
Fe	**Organic**	mg/L	1147	7632	7586	8079	4439	1991	985	644
	Aqueous	mg/L		714	562.6	114	17.5	2.75	0.84	0.64
	Loaded $_{Aq.}$	%		**37.8%**	**50.9%**	**90.1%**	**98.5%**	**99.8%**	**99.9%**	**99.9%**
Pb	**Organic**	mg/L	8142	121	38	104	117	316	415	457
	Aqueous	mg/L		7880	7893	8115	8045	8024	7747	7502
	Loaded $_{Aq.}$	%		**3.2%**	**3.1%**	**0.3%**	**1.2%**	**1.4%**	**4.8%**	**7.9%**
Mn	**Organic**	mg/L	234	2.98	0.34	1.31	1.55	4.05	6.04	7.36
	Aqueous	mg/L		222	227	233	228	227	219	219
	Loaded $_{Aq.}$	%		**5.1%**	**3.3%**	**0.6%**	**2.7%**	**3.3%**	**6.6%**	**6.6%**
Zn	**Organic**	mg/L	14.5	10.1	16.0	54.5	35.4	22.6	11.1	7.35
	Aqueous	mg/L		13.3	12.4	5.03	2.17	0.54	0.15	0.08
	Loaded $_{Aq.}$	%		**8.4%**	**14.6%**	**65.3%**	**85.0%**	**96.3%**	**99.0%**	**99.4%**
Al	**Organic**	mg/L	77.3	164.2	152.6	205.6	145.6	100.5	55.8	41.4
	Aqueous	mg/L		64.1	58.9	41.1	25.9	13.9	9.60	8.20
	Loaded $_{Aq.}$	%		**17.1%**	**23.8%**	**46.8%**	**66.5%**	**82.0%**	**87.6%**	**89.4%**
Mg	**Organic**	mg/L	633	7.10	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
	Aqueous	mg/L		606	617	631	620	623	615	636
	Loaded $_{Aq.}$	%		**4.3%**	**2.5%**	**0.3%**	**2.1%**	**1.6%**	**2.9%**	**-0.4%**
Cu	**Organic**	mg/L	2.82	1.52	1.11	1.37	0.88	0.91	0.88	0.78
	Aqueous	mg/L		2.64	2.64	2.63	2.59	2.56	2.56	2.55
	Loaded $_{Aq.}$	%		**6.4%**	**6.4%**	**6.7%**	**8.2%**	**9.2%**	**9.2%**	**9.6%**
Ca	**Organic**	mg/L	1493	27.5	13.2	11.2	9.7	11.0	13.7	11.8
	Aqueous	mg/L		1416	1442	1485	1459	1458	1436	1475
	Loaded $_{Aq.}$	%		**5.1%**	**3.4%**	**0.5%**	**2.2%**	**2.3%**	**3.8%**	**1.2%**

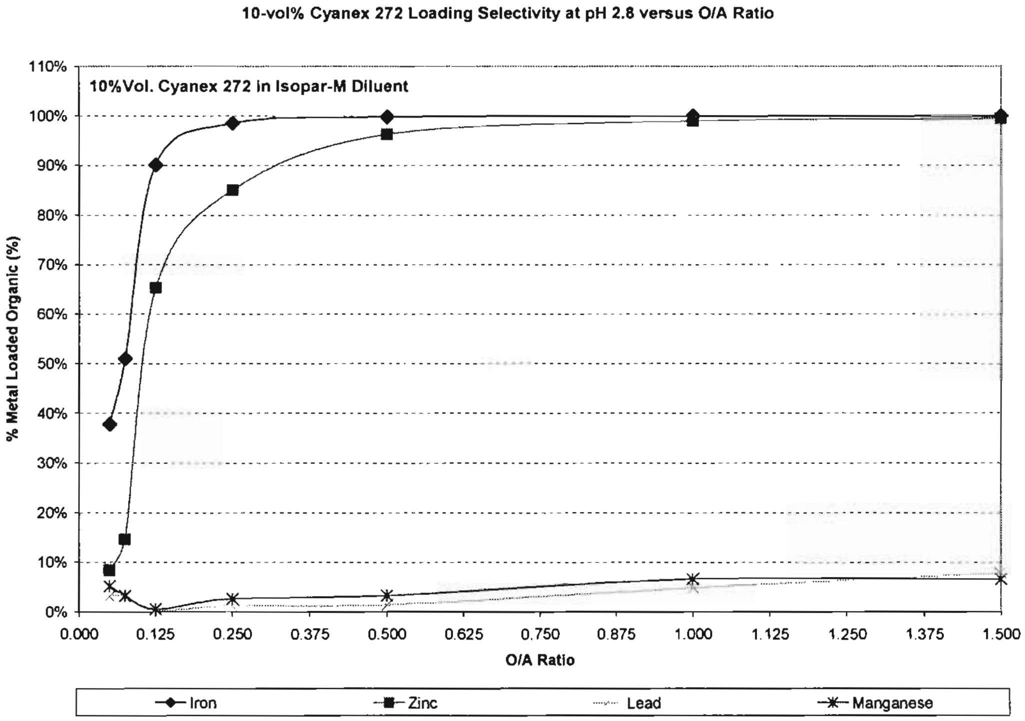

Figure 1. The effect of O/A Ratio on Fe and Zn Selectivity Against Mn and Pb Using Cyanex 272

Pb Separation Study

Stripping of Pb from Loaded Cyanex 301

Reported in Yava Pilot Plant Report No. 4 dated August 2005, Pb separation by solvent extraction using a Cyanex 301 extractant showed promising Pb loading selectivity against Mn, Mg and Ca. Test work was conducted to examine the efficiency in which Pb can be stripped from the loaded Cyanex 301 extractant using 0.5 and 1.5 M HNO3 as the eluent. The test conditions for the 0.5 M HNO3 and 1.5 M HNO3 are summarized in Tables 3 and 4 respectively.

Table 3. 0.5 M HNO3 Strip Test Conditions for Pb Loaded Cyanex 301

	Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**
Organic Solution	Pb Loaded 10.0v.% Cyanex 301	Pb Loaded 10.0v.% Cyanex 301	Pb Loaded 10.0v.% Cyanex 301	Pb Loaded 10.0v.% Cyanex 301	Pb Loaded 10.0v.% Cyanex 301	Pb Loaded 10.0v.% Cyanex 301
Aqueous Solution	0.5 M HNO3	0.5 M HNO3	0.5 M HNO3	0.5 M HNO3	0.5 M HNO3	0.5 M HNO3
Organic	40-mL	60-mL	80-mL	100-mL	120-mL	160-mL
Aqueous	160-mL	120-mL	80-mL	50-mL	40-mL	40-mL
O/A Ratio	0.250	0.500	1.000	2.000	3.000	4.000

Table 4. 1.5 M HNO3 Strip Test Conditions for Pb Loaded Cyanex 301

	Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**
Organic Solution	Pb Loaded 10.0v.%	Pb Loaded 10.0v.%	Pb Loaded 10.0v.%	Pb Loaded 10.0v.%	Pb Loaded 10.0v.%	Pb Loaded 10.0v.%

	Cyanex 301	Cyanex 301	Cyanex 301	Cyanex 301	Cyanex 301	Cyanex 301
Aqueous Solution	1.5 M HNO3	1.5 M HNO3	1.5 M HNO3	1.5 M HNO3	1.5 M HNO3	1.5 M HNO3
Organic	40-mL	60-mL	80-mL	100-mL	160-mL	160-mL
Aqueous	160-mL	120-mL	80-mL	50-mL	40-mL	20-mL
O/A Ratio	0.250	0.500	1.000	2.000	4.000	8.000

The Pb stripping test work on Cyanex 301 was conducted according to the following procedure:

1. Prepare 5000-mL of Fe and Zn removed Yava leachate using 10% by volume Cyanex 272 extractant in Isopar M diluent. The optimal conditions to remove Fe and Zn have been identified earlier in this report. These conditions consist of pH 2.8 for the aqueous feed solution and O/A ratio of 0.5.
2. Prepare 2000-mL of a 10% by volume Cyanex 301 extractant in Isopar M diluent
3. Prepare a stock solution of 50-wt.% HNO3 for pH adjustment.
4. Prepare a stock solution of 0.5 M HNO3 and 1.5 M HNO3 for stripping.
5. Prepare a Pb loaded Cyanex 301 organic solution from the Fe and Zn removed leachate. The procedure for loading the Cyanex 301 organic solution is described in Yava Pilot Plant Report No. 4 dated August 2005. The conditions for loading are O/A ratio of 0.4 and a pH for the aqueous feed of 2.25.
6. Prepare 6 separate 250 mL separatory funnels for testing different O/A ratios for Pb stripping with 0.5 M HNO3. O/A ratios tested are 0.25, 0.50, 1.00, 2.00, 3.00 and 4.00.
7. Fill each separatory funnel with 0.5 M HNO3 strip acid with the following volumes: 160, 120, 80, 50, 40 and 40 mL for Funnels #1 to #6 respectively.
8. Add the Pb loaded Cyanex 301 organic solution to each funnel with the following volumes: 40, 60, 80, 100, 120 and 160 mL for Funnels #1 to #6 respectively.
9. Place the fitted stop cock in the separatory funnel and shake aggressively for 20 minutes. Place the separatory funnel into a stand and take note of the time for initial phase break to appear in the organic or aqueous, and time to complete separation of both phases. Note whether there are any precipitates or stable emulsions after separation, as well as describe the colour and clarity both phases.
10. Remove the stop cock and drain the aqueous phase using the bottom valve on the separatory funnel into separate clean and labeled 100-mL polypropylene sample containers. Measure and record the total weight of the separatory funnel after draining the aqueous phase. Measure and record the pH/ORP of the aqueous solution and submit aqueous samples for ICP metals (H2O method), Pb assay (AA/ICP) and Fe assay (AA/ICP) analyses.
11. Pour the organic phase from the top of the separatory funnel into a clean and labeled 100-mL polypropylene sample container. Submit organic solution for ICP metals (H2O method), Pb assay (AA/ICP) and Fe assay (AA/ICP) analyses.
12. Repeat the above procedure except with the following changes: Step 7. Fill each separatory funnel with 1.5 M HNO3 strip acid with the following volumes: 160, 120, 80, 50, 40 and 20 mL for Funnels #1 to #6 respectively; Step 8. Add the Pb loaded Cyanex 301 organic solution to each funnel with the following volumes: 40, 60, 80, 100, 160 and 160 mL for Funnels #1 to #6 respectively.

A photograph of the test set-up for the 0.5 M HNO3 strip test is shown in Figure 2.

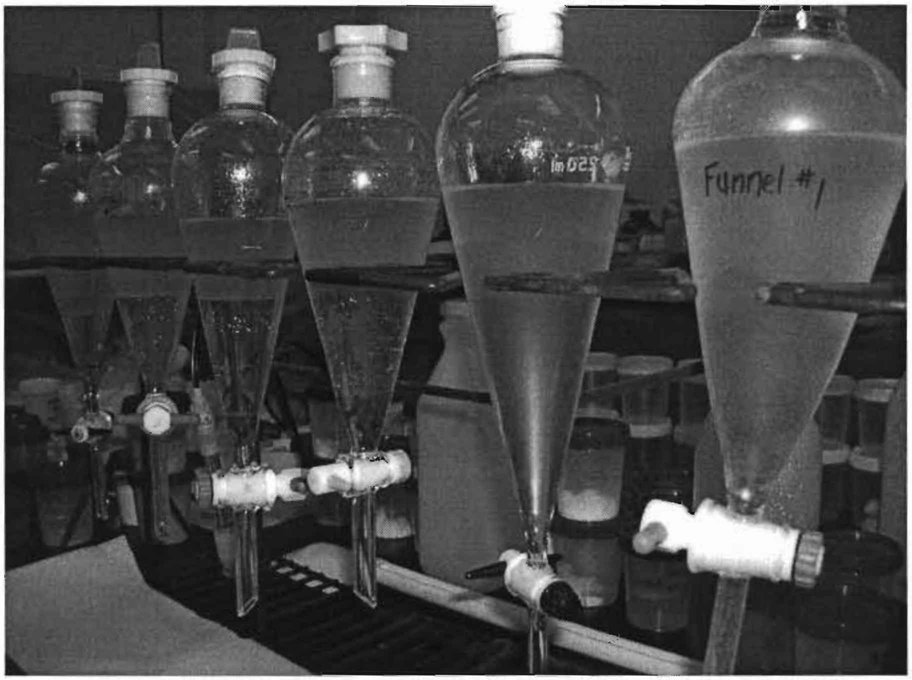

Figure 2. Photograph of the Pb Strip Tests using 0.5 M HNO3 on Pb Loaded Cyanex 301

The results of the 0.5 M HNO3 and 1.5 M HNO3 Pb stripping tests from loaded Cyanex 301 are summarized on Tables 5 and 6 respectively. In both cases, Pb removal from loaded Cyanex 301 was poor. These results will be discussed with the manufacturer for their opinion.

Table 5. Pb Stripping Test on Loaded Cyanex 301 using 0.5 M HNO3

				Funnel #1	Funnel #2	Funnel #3	Funnel #4	Funnel #5	Funnel #6
	O/A Ratio		Feed	0.25	0.50	1.00	2.00	3.00	4.00
Pb	**Calculated Organic**	mg/L	20500	20490	20496	20499	20500	20500	20500
	Aqueous	mg/L		2.58	1.76	0.65	0.81	1.09	1.30
	Stripped $_{Aq.}$	%		**0.05%**	**0.02%**	**0.00%**	**0.00%**	**0.00%**	**0.00%**

Table 6. Pb Stripping Test on Loaded Cyanex 301 using 1.5 M HNO3

				Funnel #1	Funnel #2	Funnel #3	Funnel #4	Funnel #5	Funnel #6
	O/A Ratio		Feed	0.25	0.50	1.00	2.00	4.00	8.00
Pb	**Calculated Organic**	mg/L	15000	14771	14906	14992	14994	14997	14999
	Aqueous	mg/L		57.26	46.84	7.54	12.10	11.63	7.41
	Stripped $_{Aq.}$	%		**1.53%**	**0.62%**	**0.05%**	**0.04%**	**0.02%**	**0.01%**

Pb(NO3)2 Crystal Production From Yava Leachate

A sample Pb(NO3)2 crystals was produced from Yava leachate for evaluation purposes. The Pb(NO3)2 crystals were produced by first removing Fe by solvent extraction using a Cyanex 272 extractant. The Fe removed Yava leachate was then loaded onto Purolite C104C ion exchange resin. Pb(NO3)2 solution was then produced by stripping Pb from the loaded resin with 22 wt% HNO3. The Pb(NO3)2 solution was then heated on a hot plate to remove water to form Pb(NO3)2 crystal. Photographs of the crystallisation process are shown in Figures 3 and 4. X-Ray diffraction (results in Appendix A) confirms that the crystal produced is Pb(NO3)2.

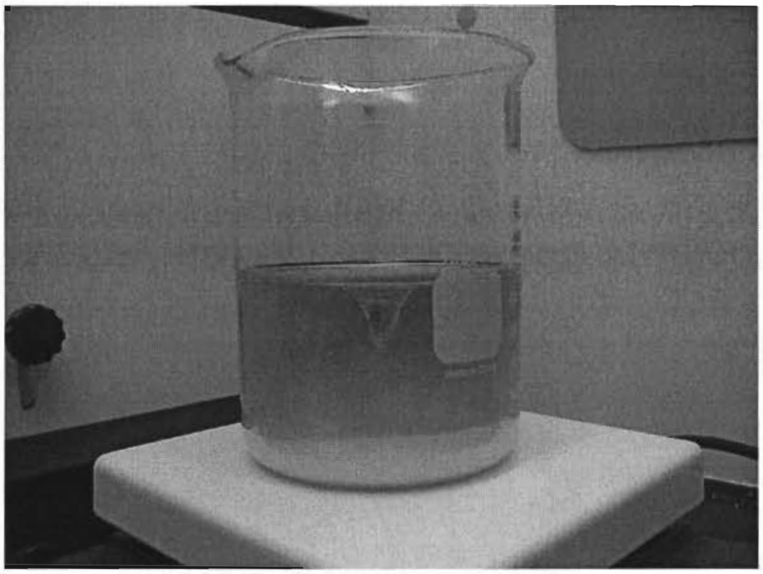

Figure 3. Pb(NO3)2 crystallization by evaporation of excess water – white Pb(NO3)2 crystals settling in beaker with impurities concentrated in the solution (brown).

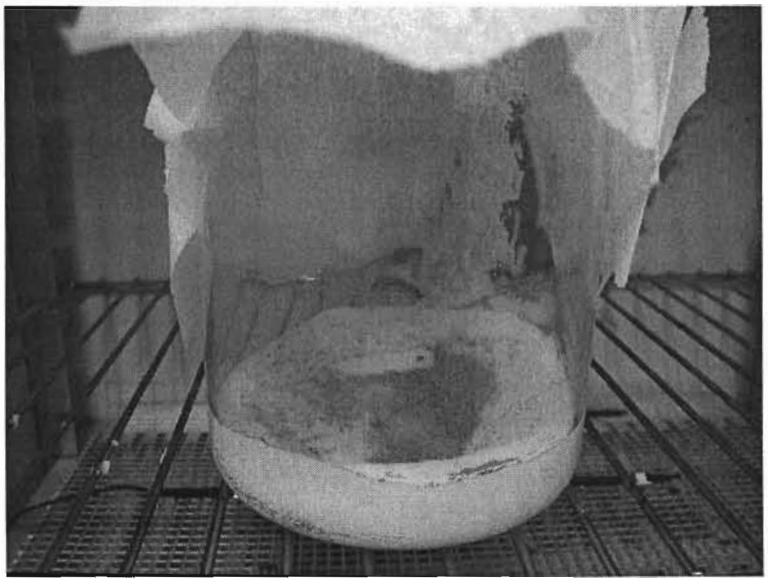

Figure 4. Pb(NO3)2 crystals dried in a low temperature oven (56°C).

Appendix A
X-RAY POWDER DIFFRACTION ANALYSIS OF ONE SAMPLE

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September 26, 2005

EXPERIMENTAL METHOD

The sample **“Batch #1, $Pb(NO_3)_2$”** was ground into fine powder and smeared on to a glass slide with ethanol. Step-scan X-ray powder-diffraction data were collected over a range 3-70°2θ with CuKα radiation on a standard Siemens (Bruker) D5000 Bragg-Brentano diffractometer equipped with a diffracted-beam graphite monochromator crystal, 2 mm (1°) divergence and antiscatter slits, 0.6 mm receiving slit and incident-beam Soller slit. The long fine-focus Cu X-ray tube was operated at 35 kV and 50 mA, using a take-off angle of 6°.

RESULTS

The X-ray diffractograms was analyzed using the International Centre for Diffraction Database PDF-4 using Search-Match software by Siemens (Bruker). The diffractogram is shown in Figure A1. The sample consists of Lead Nitrate, $Pb(NO_3)_2$.

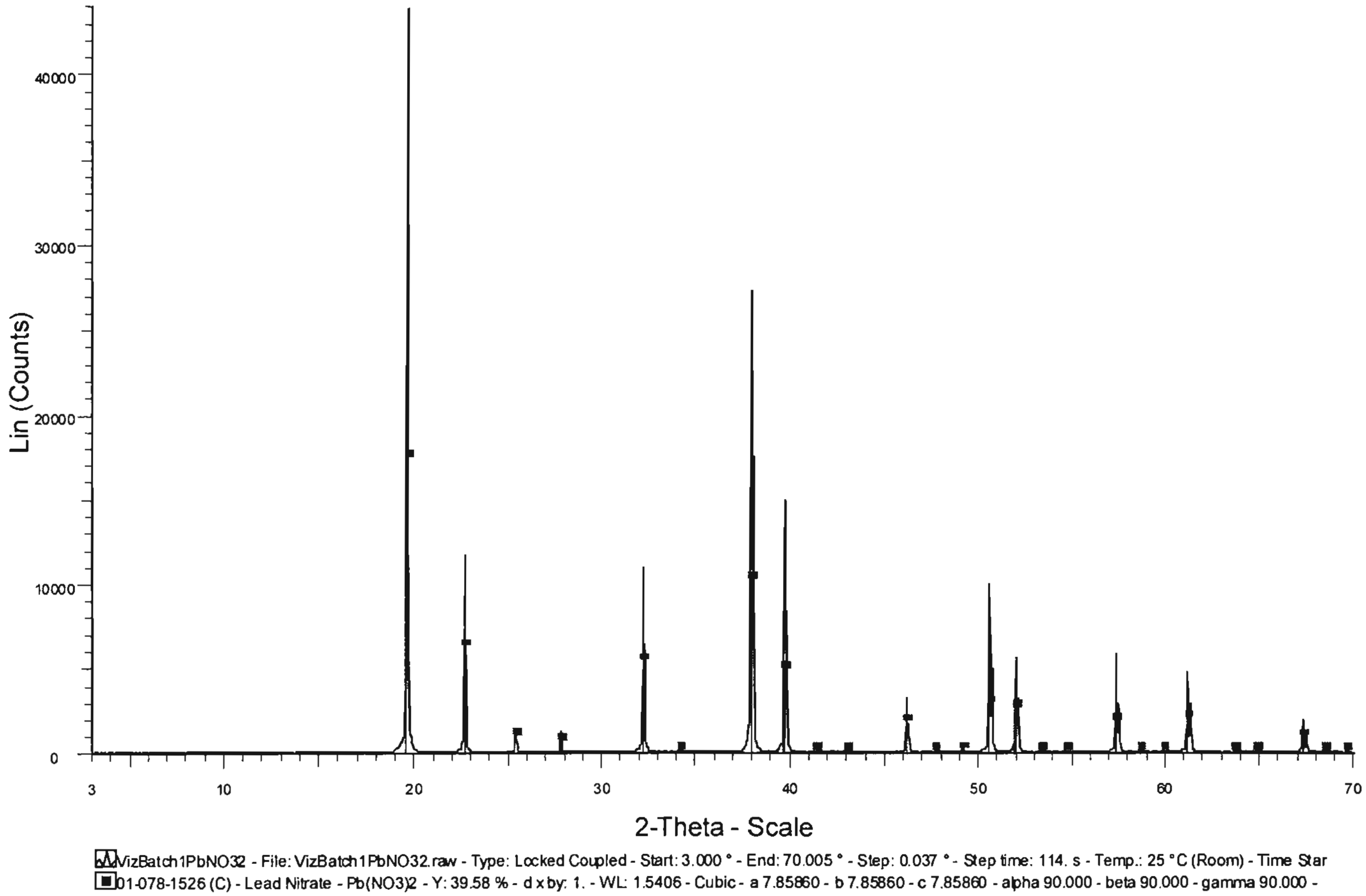

Figure A1. X-ray diffractogram of sample **Vizon Batch #1 – Pb(NO3)2**

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Project No. 4040751

Progress Report No 6

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC.
DURING OCTOBER 2005

Prepared by

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and
Clive Brereton , Alan Keylock (Noram Engineering and Constructors Ltd)

INTRODUCTION

The work carried out in the month of October 2005 focused on regeneration of Cyanex 272 used for Fe and Zn removal.

Prior studies showed that Cyanex 272 was an effective extractant for removal of Fe and Zn (with respect to loading) from Yava leachate. Test work will be conducted on developing a procedure to regenerate Cyanex 272 that has been loaded with Fe and Zn. Multiple cycle tests of loading Fe and Zn and removal of Fe and Zn will be conducted to test the robustness of Cyanex 272 for the Yava process.

SUMMARY

1. Regeneration of Cyanex 272 with only H2SO4 results in poor removal of residual Pb loaded onto the extractant. In addition, stripping Cyanex 272 results in PbSO4 precipitation in the organic phase.
2. Regeneration of Cyanex 272 with only HNO3 results in poor removal of Fe loaded onto the extractant.
3. Regeneration of Cyanex 272 was successful by using two elution steps. The first step called stripping removes Fe, Zn and Al with H2SO4. The second step called scrubbing removes Pb from the organic with HNO3. The preferred operating conditions consists of 200 g/L H2SO4, 2 M HNO3 and an O/A ratio of 1/1
4. Loading/Regeneration cycle tests showed that Cyanex 272 was robust for removal of Fe from Yava leachate.

RESULTS AND DISCUSSION

Regeneration of Cyanex 272 Used from Fe and Zn Removal

Reported in Yava Pilot Plant Report No. 5 dated September 2005, solvent extraction using Cyanex 272 was effective in removing Fe and Zn from Yava leachate prior to Pb separation. Removal of Fe and Zn prior to separation is important in improving purity of the final Pb product since practical methods tested for Pb separation were not specific against Fe and Zn. Test work was conducted on regeneration of Cyanex 272 after Fe and Zn removal from Yava leachate. The intent is to develop a practical method in which Cyanex 272 can be recycled in the Yava process.

Stripping Fe and Zn from Loaded Cyanex 272

Stripping experiments were conducted on Cyanex 272 with the intention of developing a practical method of regenerating the extractant to be recycled in the Yava process for Fe and Zn removal. Six different stripping conditions were tested, two using a H2SO4 eluent and four using a HNO3 eluent. The stripping test work was conducted on 10% Cyanex 272 that had been preloaded with Fe and Zn from Yava leachate that was adjusted to pH 2.8. The O/A ratio for loading was 1/1. The test conditions for stripping are summarized on Table 1 and the results of this test work is summarize on Table 2.

Table 1. Stripping Test Conditions for Regenerating Preloaded Cyanex 272

	Funnel #1	Funnel #2	Funnel #3	Funnel #4	Funnel #5	Funnel #6
Organic Solution	Fe-Zn Loaded 10.0v.% Cyanex 272	Fe-Zn Loaded 10.0v.% Cyanex 272	Fe-Zn Loaded 10.0v.% Cyanex 272	Fe-Zn Loaded 10.0v.% Cyanex 272	Fe-Zn Loaded 10.0v.% Cyanex 272	Fe-Zn Loaded 10.0v.% Cyanex 272
Aqueous Solution	75 g/L H2SO4	75 g/L H2SO4	0.5 M HNO3	0.5 M HNO3	1.0 M HNO3	1.0 M HNO3
Organic	80-mL	60-mL	80-mL	60-mL	80-mL	60-mL
Aqueous	80-mL	120-mL	80-mL	120-mL	80-mL	120-mL
O/A Ratio	1.0	0.5	1.0	0.5	1.0	0.5

Table 2. Analytical Results for H2SO4 and HNO3 Stripping Tests

				Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**
	O/A Ratio		**Feed**	**1.00**	**0.50**	**1.00**	**0.50**	**1.00**	**0.50**
	Eluent			**H2SO4**	**H2SO4**	**HNO3**	**HNO3**	**HNO3**	**HNO3**
Pb	**Aqueous pH 2.8 Feed**	mg/L	8582.15						
	Aqueous Raffinate	mg/L	8422.43						
	Aqueous Strip Liquor	mg/L		5.03	4.32	471.09	235.56	466.58	237.98
	Stripped $_{\text{Aq.}}$	%		**3.15%**	**5.41%**	**100.00%**	**100.00%**	**100.00%**	**100.00%**
Fe	**Aqueous pH 2.8 Feed**	mg/L	1118.31						
	Aqueous Raffinate	mg/L	3						
	Aqueous Strip Liquor	mg/L		1090.26	554.43	58.60	34.83	193.52	90.85
	Stripped $_{\text{Aq.}}$	%		**97.75%**	**99.42%**	**5.25%**	**6.25%**	**17.35%**	**16.29%**
Zn	**Aqueous pH 2.8 Feed**	mg/L	14.12						
	Aqueous Raffinate	mg/L	0.15						
	Aqueous Strip Liquor	mg/L		13.92	7.16	14.80	7.35	14.72	7.33
	Stripped $_{\text{Aq.}}$	%		**99.64%**	**100.00%**	**100.00%**	**100.00%**	**100.00%**	**100.00%**
Al	**Aqueous pH 2.8 Feed**	mg/L	70.9						
	Aqueous Raffinate	mg/L	8.6						
	Aqueous Strip Liquor	mg/L		58.00	29.90	61.00	30.10	60.60	30.40
	Stripped $_{\text{Aq.}}$	%		**93.10%**	**95.99%**	**97.91%**	**96.63%**	**97.27%**	**97.59%**

The analytical results summarized in Table 2 show that H2SO4 is not effective in removing residual Pb that loaded onto the Cyanex 272. In fact, elution of loaded Cyanex 272 with H2SO4 resulted in precipitate formation in the organic. This precipitate is suspected of being PbSO4 (See Figure 1). H2SO4, however, was effective in removing Fe, Zn and Al. HNO3 was not effective in removing Fe but effective in removing Pb, Zn and Al.

Clearly a two stage elution process is required to effectively regenerate loaded Cyanex 272. Further test work will be conducted by first eluting loaded Cyanex 272 with H2SO4 to remove Fe, Zn and Al (identified as the stripping step) followed by eluting with HNO3 to remove Pb and re-dissolve PbSO4 precipitate (identified as the scrubbing step).

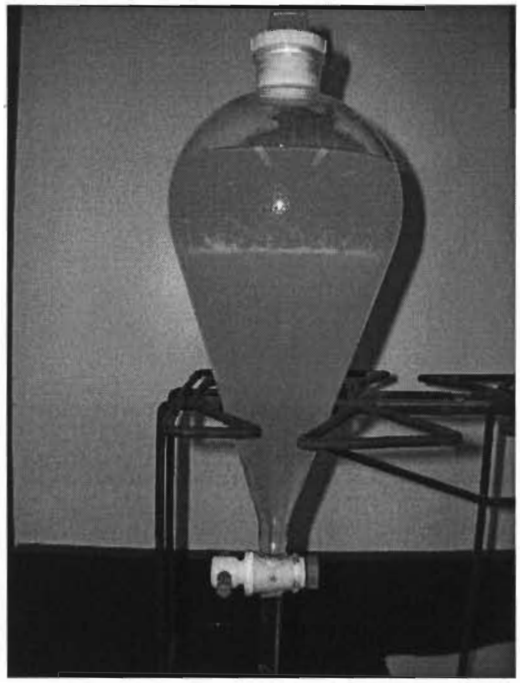

Figure 1. Cyanex 272 after elution with H_2SO_4 showing white precipitates at the organic/aqueous interface.

Stripping and Scrubbing Fe, Zn and Pb from Loaded Cyanex 272

Test work was conducted to regenerate loaded Cyanex 272 by first stripping the Cyanex 272 solution with H2SO4 followed by scrubbing the organic with HNO3. The effects of acid concentration and O/A ratios were examined.

The H_2SO_4 stripping tests were conducted on loaded Cyanex 272 under the conditions summarized on Table 3.

Table 3. H2SO4 Stripping Test Conditions on Loaded Cyanex 272 Organic Solution.

	Funnel #1	**Funnel #2**	**Funnel #3**	**Funnel #4**	**Funnel #5**	**Funnel #6**
H_2SO_4	100-g/L	100-g/L	150-g/L	150-g/L	200-g/L	200-g/L
Organic	80-mL	60-mL	80-mL	60-mL	80-mL	60-mL
Aqueous	80-mL	120-mL	80-mL	120-mL	80-mL	120-mL
O/A	1/1	1/2	1/1	1/2	1/1	1/2

In each stripping test, the separatory funnels were shaken for 10-minutes and then allowed to settle. In all tests, a stable emulsion and white precipitate solids were observed (suspected as being $PbSO_4$). The aqueous strip liquor was removed and submitted for metals analysis by ICP. The organic and most of the precipitate solids were carried into the next set of scrubbing tests with HNO3. The analytical results of the H2SO4 stripping tests are summarized on Table 4.

Table 4. Analytical Results for H2SO4 Stripping Tests

				Funnel #1	Funnel #2	Funnel #3	Funnel #4	Funnel #5	Funnel #6
	O/A Ratio		Feed	1.00	0.50	1.00	0.50	1.00	0.50
	Eluent			H2SO4	H2SO4	H2SO4	H2SO4	H2SO4	H2SO4
Pb	Aqueous pH 2.8 Feed	mg/L	8582.15						
	Aqueous Raffinate	mg/L	8422.43						
	Aqueous Strip Liquor	mg/L		5.31	6.00	11.85	5.34	11.53	4.41
	Stripped $_{Aq.}$	%		3.32%	7.51%	7.42%	6.69%	7.22%	5.52%
Fe	Aqueous pH 2.8 Feed	mg/L	1118.31						
	Aqueous Raffinate	mg/L	3						
	Aqueous Strip Liquor	mg/L		1175.90	570.25	1188.37	582.30	1210.33	571.72
	Stripped $_{Aq.}$	%		100.00%	100.00%	100.00%	100.00%	100.00%	100.00%
Zn	Aqueous pH 2.8 Feed	mg/L	14.12						
	Aqueous Raffinate	mg/L	0.15						
	Aqueous Strip Liquor	mg/L		14.71	7.32	14.75	7.37	14.52	7.17
	Stripped $_{Aq.}$	%		100.00%	100.00%	100.00%	100.00%	100.00%	100.00%
Al	Aqueous pH 2.8 Feed	mg/L	70.9						
	Aqueous Raffinate	mg/L	8.6						
	Aqueous Strip Liquor	mg/L		52.80	26.30	54.10	26.50	53.40	26.00
	Stripped $_{Aq.}$	%		84.75%	84.43%	86.84%	85.07%	85.71%	83.47%

As shown in Table 4, H2SO4 at all the concentrations and O/A ratios tested were effective in removing Fe, Zn and the majority of Al from the loaded Cyanex 272 organic solution.

Using the Cyanex 272 organic solutions that had been previously stripped with H2SO4, HNO3 scrubbing experiments were conducted according to the conditions summarized in Table 5. Note that the H2SO4 stripped Cyanex 272 organic solutions were re-introduced in the same respective funnel numbers for these tests.

Table 5. HNO3 Scrubbing Test Conditions on H2SO4 Stripped Cyanex 272 Organic Solutions.

	Funnel #1	Funnel #2	Funnel #3	Funnel #4	Funnel #5	Funnel #6
HNO_3	0.5-M	1.0-M	0.5-M	1.0-M	0.5-M	1.0-M
Organic	80-mL	60-mL	80-mL	60-mL	80-mL	60-mL
Aqueous	80-mL	60-mL	80-mL	60-mL	80-mL	60-mL
O/A	1/1	1/1	1/1	1/1	1/1	1/1

In each scrubbing test the separatory funnels were shaken for 10-minutes and then allowed to settle. Precipitates were noticed in each scrub test using 0.5-M HNO3 which implied 0.5-M HNO_3 was not strong enough to remove the $PbSO_4$ precipitate formed. A 1.0-M HNO3 scrub solution was sufficient to remove the precipitates formed by stripping with 100-g/L H2SO4 and 150-g/L H2SO4 but not enough to completely remove the precipitates formed by stripping with 200-g/L H2SO4 (Funnel #6).

Tests were conducted to determine whether the regenerated Cyanex 272 solution performs as well as a fresh Cyanex 272 solution. A 2.0-M HNO_3 stock solution was prepared and used to scrub the organic a second time in Funnels #2, #4 and #6 prior to recycling the stripped and scrubbed Cyanex 272 organic back to loading. A fresh 10-vol% Cyanex 272 solution was placed in Funnel #1. Load tests were then conducted with Yava leachate according to the test conditions summarized in Table 6.

Table 6. Loading Test Conditions for Fresh and Regenerated Cyanex 272 using Yava Leachate

	F1	**F2**	**F4**	**F6**
Organic	**Fresh**	**Recycle**	**Recycle**	**Recycle**
Organic	60-mL	60-mL	60-mL	60-mL
Aqueous	60-mL	60-mL	60-mL	60-mL
O/A	1/1	1/1	1/1	1/1

In each loading test each separatory funnel was shaken for 20-minutes and then allowed to settle. The aqueous raffinates produced from these loading tests were submitted for analyses (results summarized in Table 7).

Table 7. Raffinate Analysis From Loading Test Conducted on Fresh and Regenerated Cyanex 272

				Funnel #1	**Funnel #2**	**Funnel #4**	**Funnel #6**
	O/A Ratio		**Feed**	**1.00**	**1.00**	**1.00**	**1.00**
Pb	**Aqueous pH 2.8 Feed**	mg/L	8582.15				
	Aqueous Raffinate	mg/L		8593.45	8688.40	8668.14	8574.49
	Loaded %	%		**0.00%**	**0.00%**	**0.00%**	**0.09%**
Fe	**Aqueous pH 2.8 Feed**	mg/L	1118.31				
	Aqueous Raffinate	mg/L		1.75	1.27	0.80	0.56
	Loaded %	%		**99.84%**	**99.89%**	**99.93%**	**99.95%**
Zn	**Aqueous pH 2.8 Feed**	mg/L	14.12				
	Aqueous Raffinate	mg/L		0.4	1.28	1.6	1.27
	Loaded %	%		**97.17%**	**90.93%**	**88.81%**	**91.01%**
Al	**Aqueous pH 2.8 Feed**	mg/L	70.9				
	Aqueous Raffinate	mg/L		16.7	45.0	44.9	36.1
	Loaded %	%		**76.45%**	**36.53%**	**36.67%**	**49.08%**

The assay results in Table 7 show degradation with respect to Fe loading after a single cycle. There is a small amount of degradation with respect to Zn loading and more degradation with respect to Al loading. However, most important is the removal of Fe since it is present in higher concentrations and has a greater detrimental effect in Yava down stream process.

Given the information obtained to date, the preferred regeneration condition for Cyanex 272 is to strip with 200-g/L H_2SO_4 at an O/A ratio = 1/1 followed by scrubbing with 2-M HNO_3 at an O/A ratio = 1/1.

Cyanex 272 Load/Regeneration 10 Cycle Test

Robustness testing of Cyanex 272 was conducted by performing 10 consecutive load/regeneration tests and examining subsequent Fe loading performance after each cycle. A flow diagram describing this cycle test is shown in Figure 2.

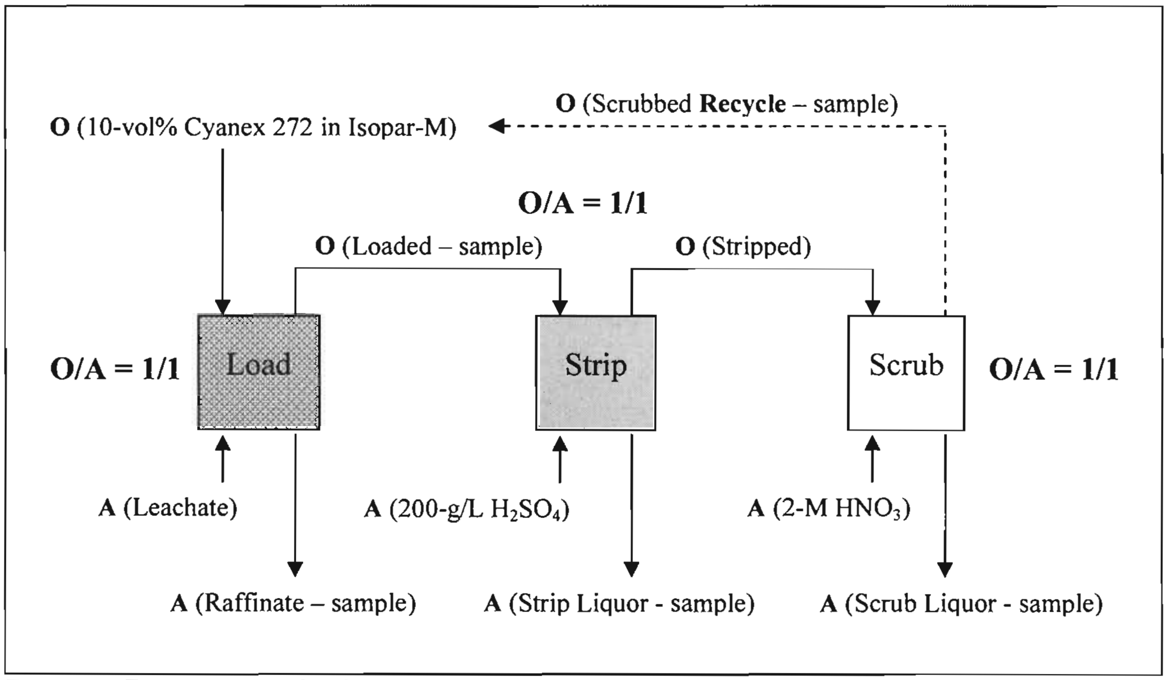

Figure 2. Flow Diagram for Cyanex 272 Robustness Testing.

Starting with freshly prepared 10-vol% Cyanex 272 in Isopar-M diluent, the first loading, stripping and scrubbing of the organic (cycle #1) will act as the baseline result for the remaining cycles to be compared. Samples of loaded organic and scrubbed organic will be taken at the locations shown in Figure 1 and used to evaluate degradation of the Cyanex 272.

A plot of Fe concentration in the loaded Cyanex 272 solution, and stripped and scrubbed Cyanex 272 solution for each cycle tested in shown in Figure 3. Since there is very little change in Fe concentration in the organic as a function of cycle, the Cyanex 272 exhibits good robustness for Fe removal.

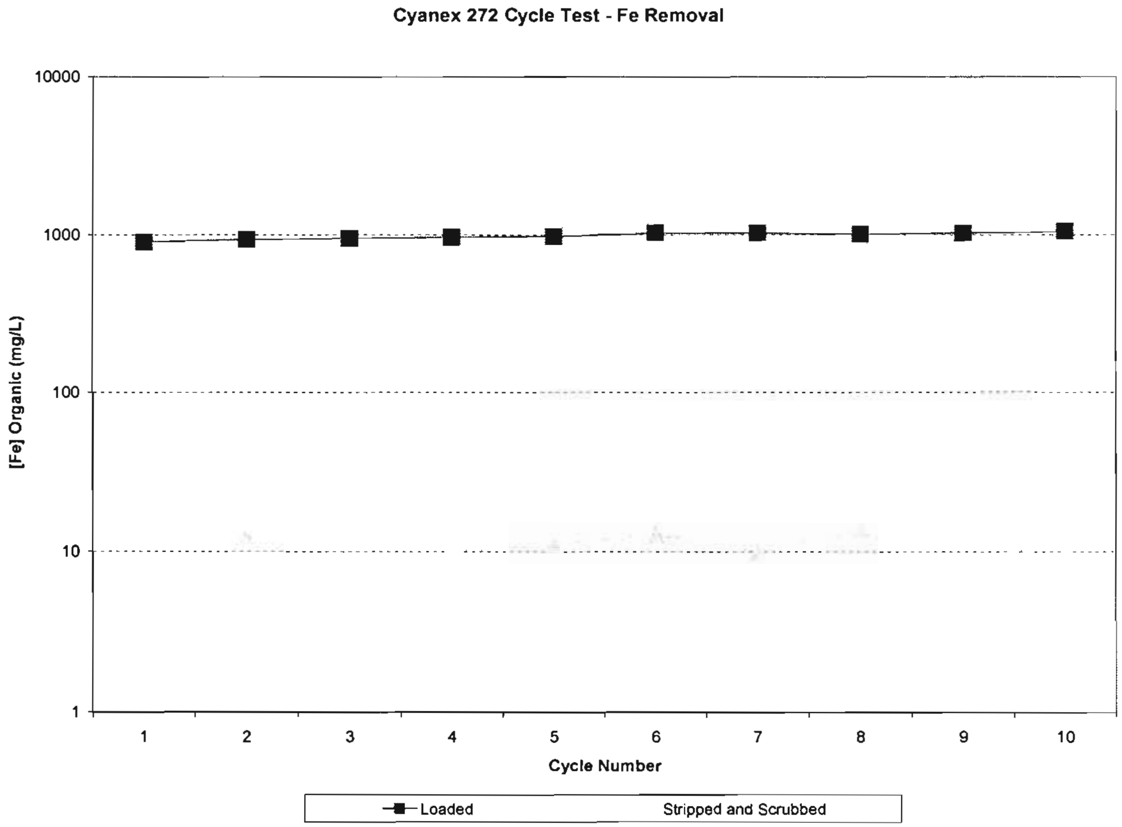

Figure 3. Iron Removal by Cyanex 272 Cycle Loading, Stripping and Scrubbing 10-Times.

The above graph shows that after 10 cycles of Fe-loading with pH 2.8 adjusted as-received leachate, Fe-stripping with 200-g/L H_2SO_4 and $PbSO_4$-scrubbing with 2-M HNO_3, the 10-vol% Cyanex 272 in Isopar-M diluent in Cycle #10 performed as good as in Cycle #1. Therefore no degradation occurred within the 10 cycles tested.

7

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Project No. 4040751

Progress Report No 7

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC. DURING NOVEMBER 2005

Prepared by

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and
Clive Brereton, Alan Keylock (Noram Engineering and Constructors Ltd)

INTRODUCTION

The work carried out in the month of November 2005 focused on reducing the acid consumption for the stripping and scrubbing steps required to regenerate Cyanex 272 used for Fe removal. In order to remove the Fe loaded onto Cyanex 272, a 200-g/L H_2SO_4 solution was used as the eluent. The traces of Pb that had loaded into organic phase react with the H2SO4 eluent to form a white $PbSO_4$ precipitate. To further recycle the organic phase the precipitate must be removed by scrubbing the organic solution with a 2-M HNO_3.

To minimize the acid consumption, it was decided to recycle the 200-g/L H_2SO_4 solution used for stripping the Fe-loaded Cyanex 272 (i.e. allowing the Fe to concentrate in the H2SO4 following each regeneration cycle). A test was performed to determine if the efficiency of the H2SO4 eluent would decrease after 6 cycles.

The production of 50-g $Pb(NO_3)$ from Yava leachate was investigated. Pb(NO3)2 was produced using Purolite C104-H ion exchange resin with leachate solutions with and without Fe removal by Cyanex 272. The purpose of this test was to produce a high purity $Pb(NO_3)$ crystals. Further washing of the crystals was used to achieve a high purity product.

Another option to improve the purity of the Pb(NO3)2 was to scrub the loaded Purolite C104H resin with acetic acid prior to stripping, a step that would ensure that traces of various metals co-loaded on the resin will be removed.

The loading efficiency and selectivity of Purolite S940 were investigated with Fe removed Yava leachate (treated with Cyanex 272).

Test work was conducted to determine the effects of pH of the aqueous feed solution on Pb selectivity of the resin. pH and equilibrium isotherm tests were carried out using Bayer TP207-Na (sodium form).

A test was conducted to determine if the greater than 12-wt% HNO_3 used to strip Pb from Bayer TP207-H resin would cause resin wear. Loaded Bayer TP207-H saved from a previous test was stripped with an R/A ratio of 1.25 using 10-mL of pre-loaded Bayer TP207-H and 8-mL of freshly prepared nitric acid strip solution at 10-wt%, 15-wt% and 20-wt% HNO_3.

SUMMARY

1. As suggested by the manufacturer, 200-g/L H_2SO_4 solution was used to successfully strip the Fe-loaded Cyanex 272. Pb traces in the organic phase reacted with the sulphuric acid, creating a white precipitate that required removing by scrubbing with 2-M HNO_3. The spent H_2SO_4 was recycled (allowing Fe to accumulate) and proved to be as effective as the fresh acid solution.
2. After 6 cycles, the recycled 200-g/L H_2SO_4 still performed as well as the fresh one, but it is still unknown how many cycles can the acid solution withstand without needing refreshing.
3. 50-g Pb(NO3)2 crystal samples were produced with Purolite C104-H ion exchange resin using as received Yava leachate and Fe removed Yava leachated (treated with Cyanex 272). Fe removal and water washing of the crystal was found to be necessary to produce a high purity Pb(NO3)2 product.
4. When 2-M, 4-M and 10-M acid acetic was used to scrub the Pb loaded Purolite C104-H resin to increase the purity of $Pb(NO_3)_2$, impurities such as Mn, Ca, Mg and Al were removed. 4-M acetic acid was found to be a good choice.
5. Purolite S940 was tested as an alternative for Purolite C104-H, but due to its lower Pb loading capacity was abandoned.
6. pH scope and equilibrium isotherm tests for Bayer TP207-Na (sodium form) were performed to determine optimum operation conditions.
7. Bayer TP207 Pb stripping scope test were conducted with HNO_3 at 10, 15 and 20wt%. It was determined that concentrations equal or greater than 15-wt% HNO_3 are too high, leading to the degradation of the Bayer TP207 ion exchange resin. The suggested maximum concentration that can be used for stripping was 12-wt% HNO_3.
8. A 12-wt% HNO_3 was used to successfully convert the Bayer TP207-Na (sodium form) to Bayer TP207-H (hydrogen form).

RESULTS AND DISCUSSION

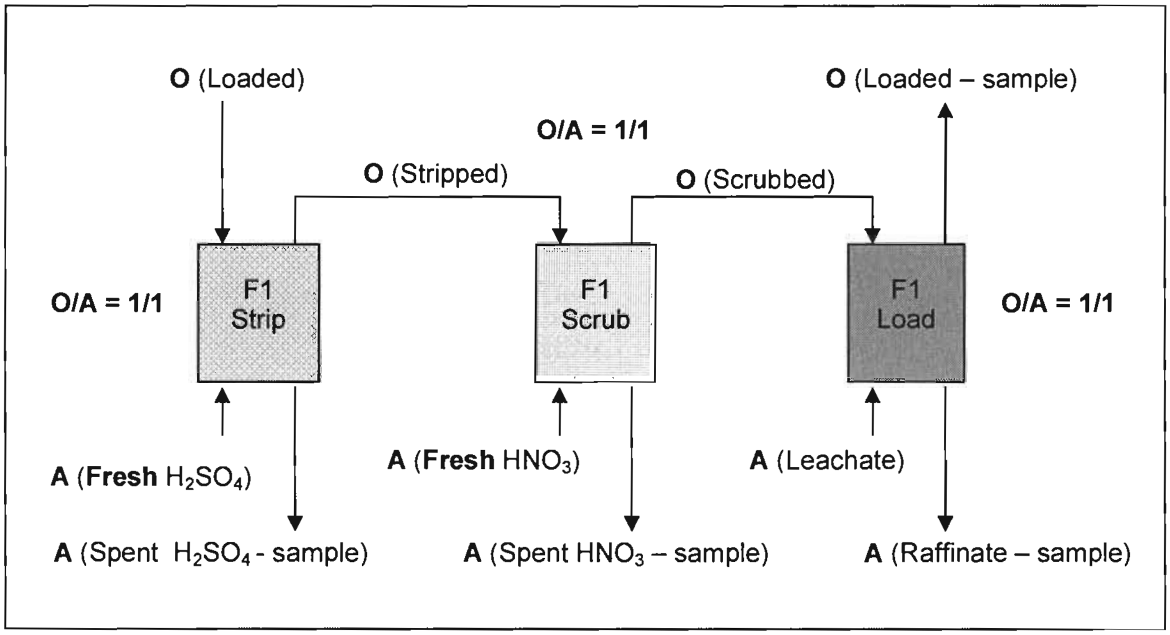
Acid Reduction Test (Stripping with H_2SO_4 and Scrubbing with HNO_3)

After establishing an effective Cyanex 272 regeneration process (involving H2SO4 stripping and HNO3 scrubbing), and testing this successfully over multiple cycles, the objective in this test program is to reduce the overall acid requirement for regeneration.

Several separatory funnel tests were designed to test the effectiveness of recycling the 200-g/L H_2SO_4 strip solution and 2-M HNO_3 scrub solution. A batch of freshly prepared 10-vol% Cyanex 272 in Isopar-M diluent was pre-loaded with a pH 2.8 adjusted (with 50-wt% HNO_3) as-received leachate solution at a O/A ratio = 1/1 (contacted for 20-minutes). This loaded organic was used in the acid reduction stripping/scrubbing scope test described below.

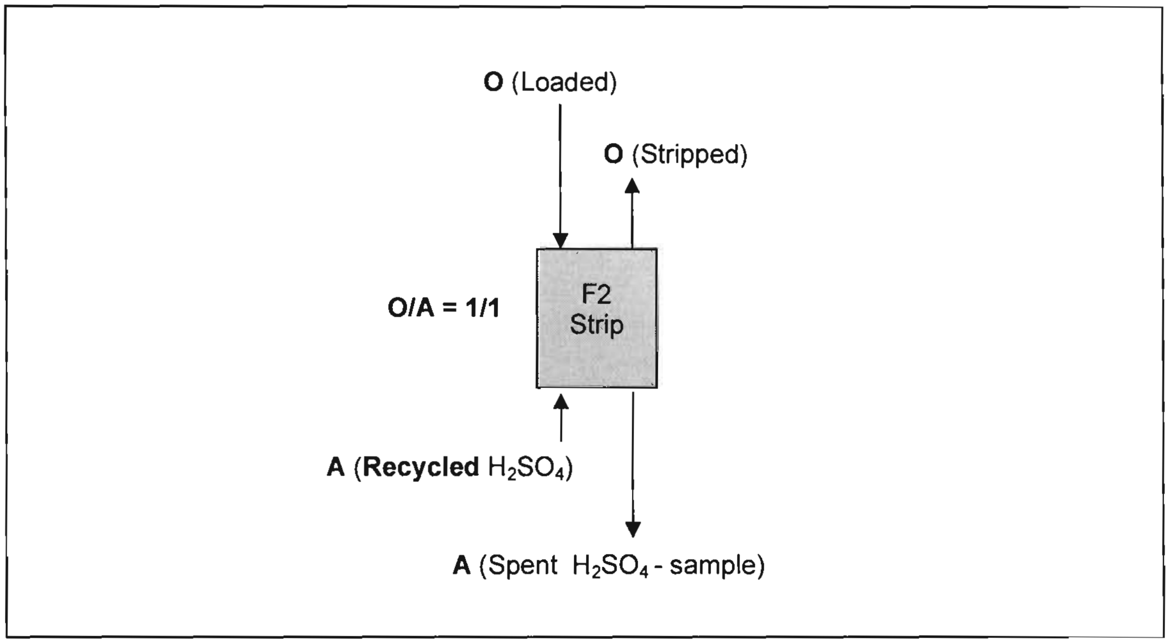
Funnel #1 was the baseline condition, stripping with fresh 200-g/L H_2SO_4 at O/A ratio = 1/1, scrubbing with fresh 2-M HNO_3 at O/A ratio = 1/1 and recycle loading with pH 2.8 adjusted leachate at O/A ratio = 1/1. A flow diagram of Funnel #1 is shown in **Figure 1.0** below.

Figure 1.0 Funnel #1 Test – Standard condition for performance comparison.



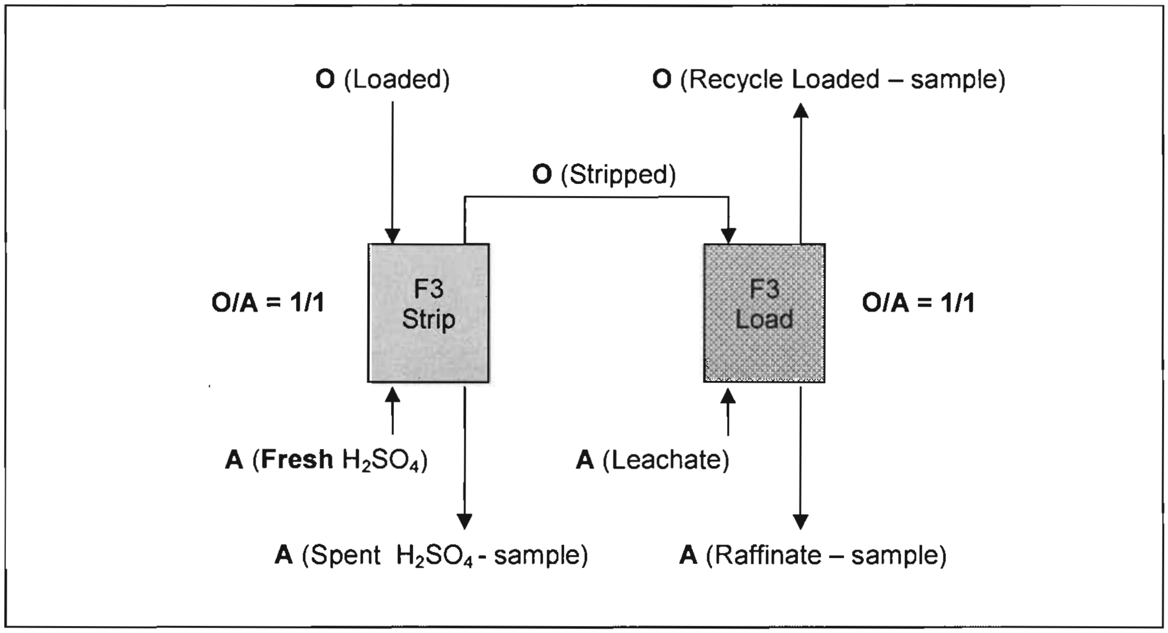
Funnel #2 was designed to test the stripping performance of a used 200-g/L H_2SO_4 strip liquor collected in a previous test program. The flow diagram of Funnel #2 is shown in **Figure 2.0** below.

Figure 2.0 Funnel #2 – To determine the effectiveness of recycled 200-g/L H2SO4 in stripping the loaded organic (10-vol% Cyanex 272).



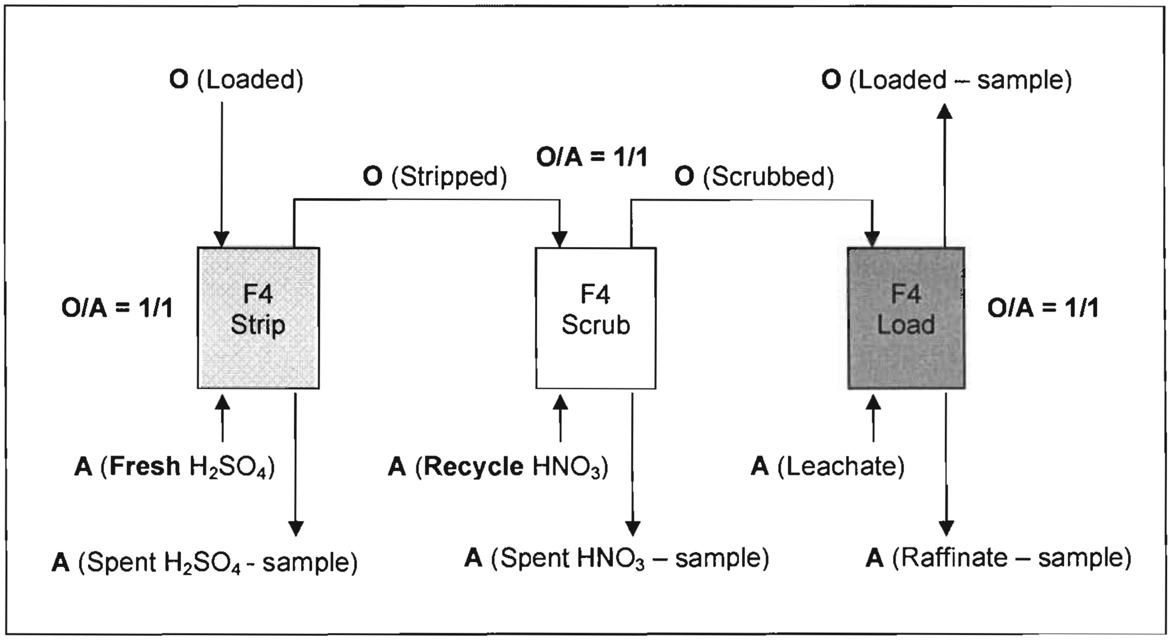
Funnel #3 was designed to test the requirement of the 2-M HNO_3 scrubbing after stripping with fresh 200-g/L H_2SO_4. The white precipitate solids ($PbSO_4$) may form a third phase after stripping with H_2SO_4 but may not interfere with the performance of the Cyanex 272 – this will be tested and verified. The flow diagram of funnel #3 is shown in **Figure 3.0**.

Figure 3.0 Funnel #3 – To determine the effects of not scrubbing before recycling the organic back to loading.



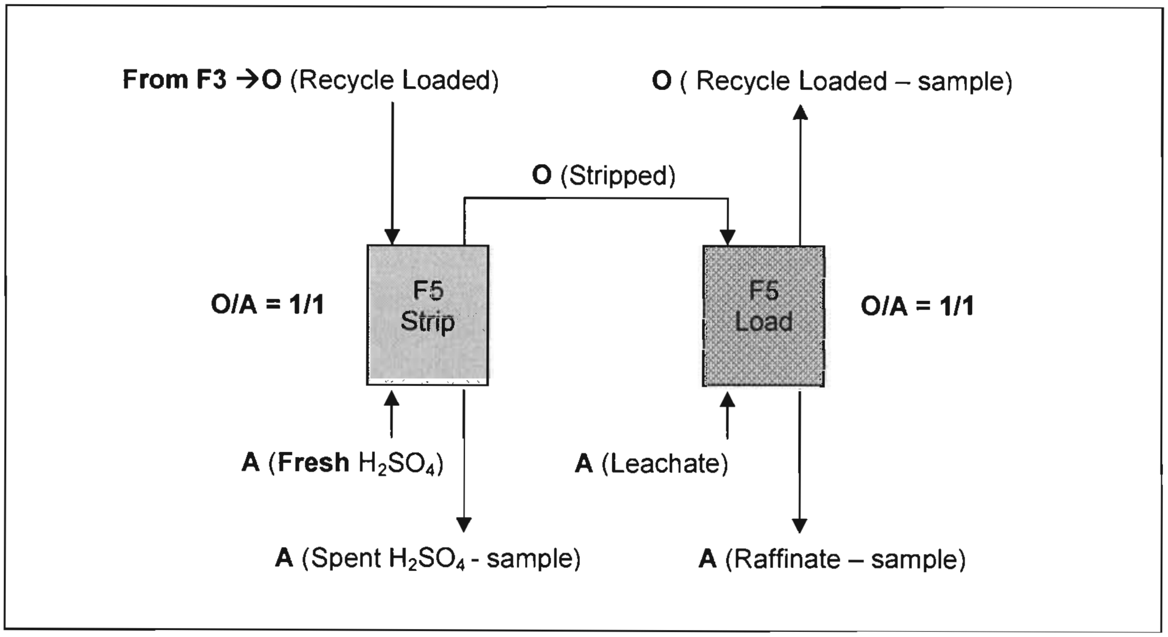
Funnel #4 was designed to test the scrubbing performance of a used 2-M HNO_3 scrub solution collected from a previous test program. The flow diagram of Funnel #4 is shown **Figure 4.0**.

Figure 4.0 Funnel #4 – To determine the effects of using a recycled 2-HNO_3 in during scrubbing.

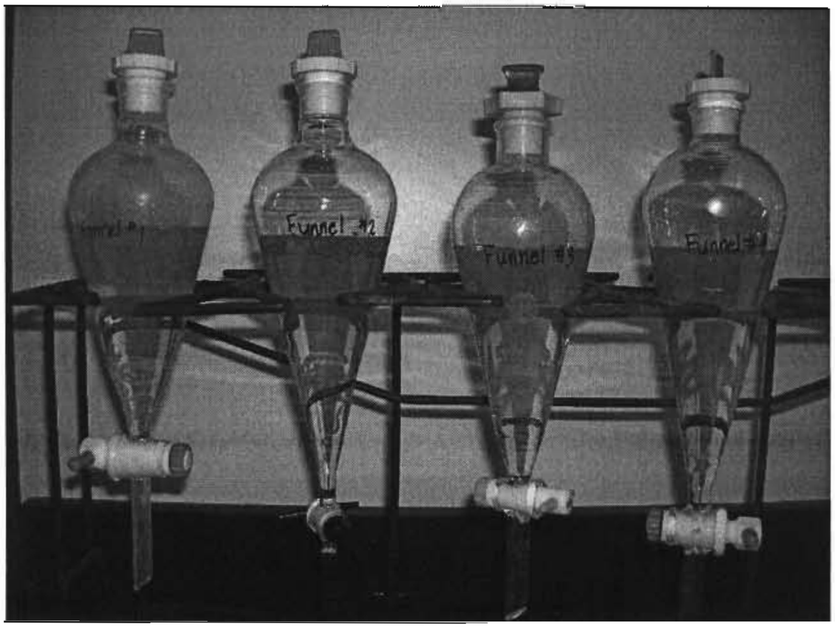


Funnel #5 was extending the design test of Funnel #3 to a second cycle where no 2-M HNO_3 scrubbing was performed after stripping with fresh 200-g/L H_2SO_4. The flow diagram of Funnel #5 is shown **Figure 5.0**.

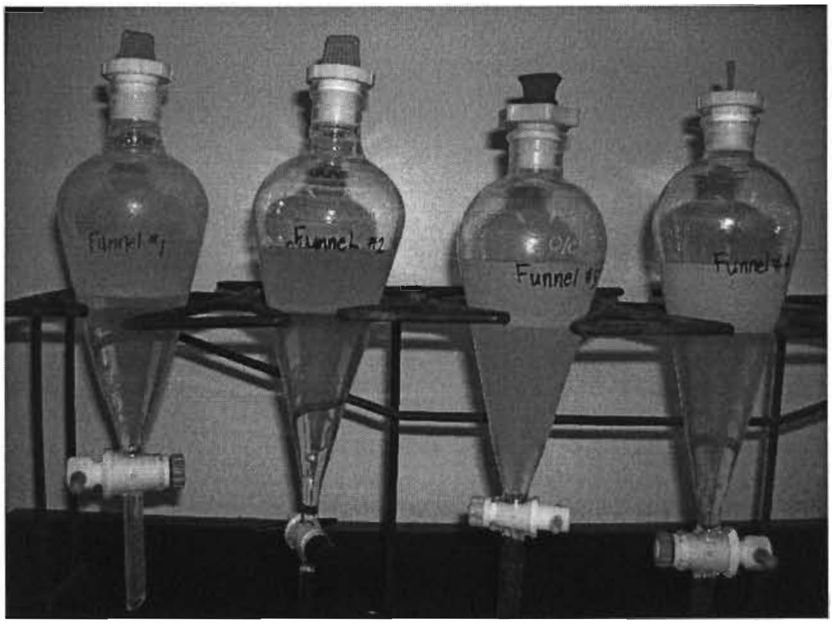
Figure 5.0 Funnel #5 – No scrub after stripping second cycle.



In **Picture 1.0** below, the Fe-loaded organic phase is ready to be stripped with 200-g/L H_2SO_4 where Funnel #2 (second from the left) contains a recycled 200-g/L H_2SO_4 and the remaining funnels contain freshly prepared 200-g/L H_2SO_4. After each separatory funnel is shaken for 10-minutes (contact time), the funnels are placed back onto the rack and the phases are allowed to separate. **Picture 2.0** shows complete phase separation.



Picture 1.0 Funnel #1 to Funnel #4 (from left to right) prior to 200-g/L H_2SO_4 stripping where the loaded organic phase is on top (dark orange/brown) and the aqueous H_2SO_4 strip solution is on the bottom (light brown).



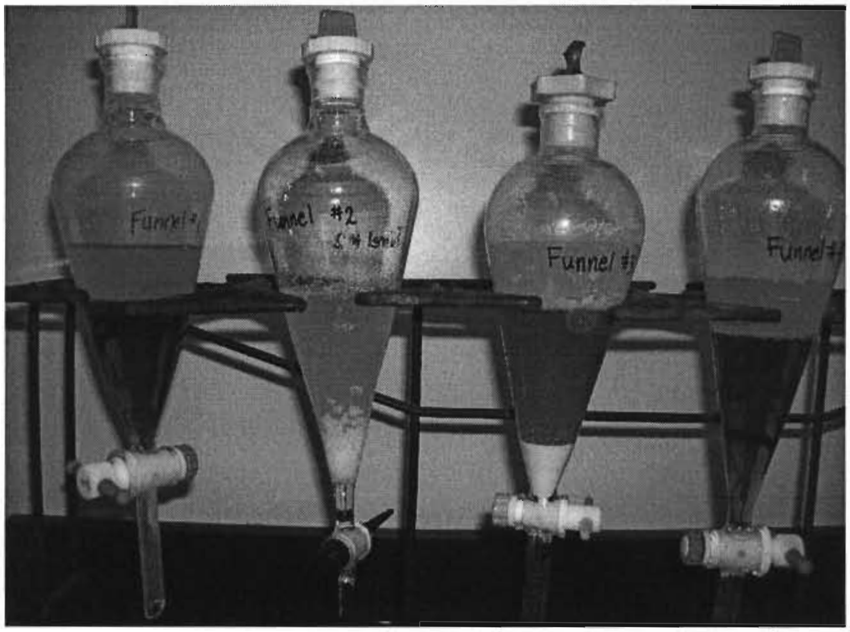
Picture 2.0 Funnel #1 to #4 (from left to right) organic and aqueous after stripping with 200-g/L H_2SO_4 where Funnel #2 was stripped with recycled H_2SO_4.

From **Picture 2.0** above, after stripping with 200-g/L H_2SO_4, a white precipitate can be observed in each of the funnels tests. This white precipitate is suspected to be $PbSO_4$ since Pb in contact with SO_4 will form a white precipitate product. The loaded organic prior to stripping was assayed to contain approximately 368-ppm Pb (co-extracted during Fe-loading) and the stripped organic was assayed to contain less than or equal than 1-ppm Pb. Thus, resulting in 99.8% Pb strip efficiency after stripping with 200-g/L H_2SO_4 at an O/A ratio = 1/1. If most of all the Pb was stripped from the organic phase then the Pb must be in the form of a precipitate solid ($PbSO_4$) and small amounts in the aqueous strip liquor (approximately 4-ppm).

The Fe stripping efficiency with 200-g/L H_2SO_4 using freshly prepared H_2SO_4 and recycled H_2SO_4 was determined to be 88.8% and 89.4%, respectively (based on Fe assay results of the organic). Therefore, the recycled 200-g/L H_2SO_4 proved to be as effective as freshly prepared 200-g/L H_2SO_4 strip solution. The next test program will test the effectiveness of this used 200-g/L H_2SO_4 strip solution after multiple recycles. The more recycle applications of this 200-g/L H_2SO_4 for stripping will result in lower costs for Fe removal by Cyanex 272 solvent extraction.

Interesting enough, the recycle loaded organic (where the organic has been stripped and scrubbed once) in Funnel #1 and Funnel #4 both loaded more Fe the second time around at the same O/A ratio = 1/1. This can not be attributed entirely to the recycled loading performance of the organic but most likely due to the margin of error of the assay results performed.

The recycle loading of the stripped organic in Funnel #1, Funnel #3 and Funnel #4 with pH 2.8 adjusted Yava leachate solution is shown in **Picture 3.0** just prior to shaking for 20-minutes. Without the application of scrubbing the organic with 2-M HNO_3 after stripping with 200-g/L H_2SO_4, the white $PbSO_4$ precipitate is much more apparent (settled at the bottom of Funnel #3).



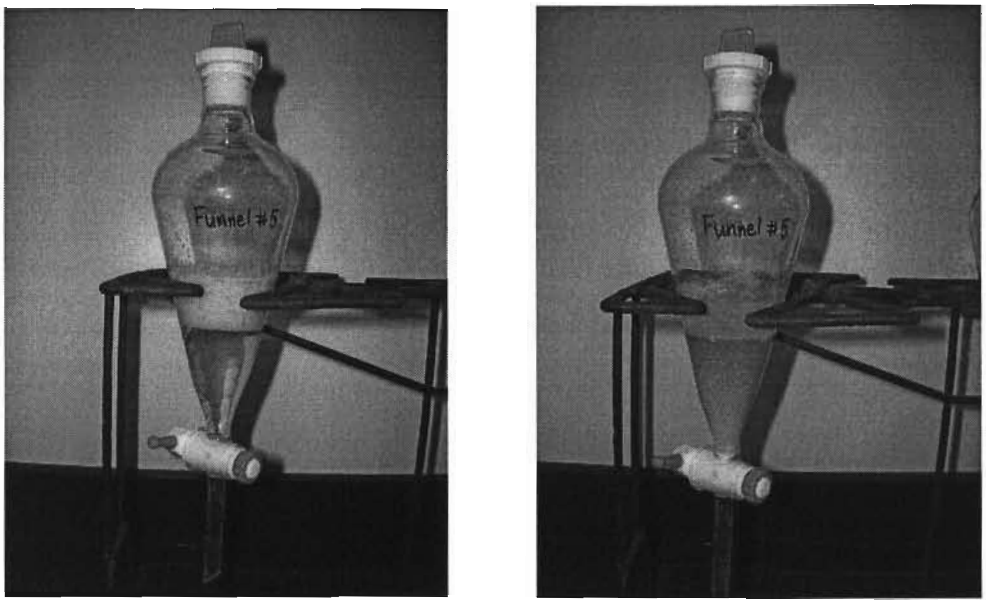
Picture 3.0 Funnel #1 to #4 (left to right) prior to recycle loading with pH 2.8 adjusted leachate solution, where Funnel #2 only contains stripped organic (not participating in this recycle loading shake-out test).

After shaking Funnel #1, Funnel #3 and Funnel #4 for 20-minutes (contact time), the funnels were placed back on the rack and the phases were allowed to separate, as shown in **Picture 4.0**. The white $PbSO_4$ precipitate solids in Funnel #3 has now formed a stable emulsion in the organic phase – peach color froth shown in **Picture 4.0** below.

Picture 4.0 Funnel #1, #3 and #4 organic and aqueous after recycled organic is re-loaded with pH 2.8 adjusted leachate solution.

This third solid phase in the solvent extraction process will tend to create a stable emulsion which will hinder separation kinetics as observed in Funnel #3 test, where the aqueous phase required approximately 15-minutes to achieve separation, oppose to 50-seconds observed in the other funnels.

Funnel #5 test was to determine if a 2-M HNO_3 scrubbing step was required to polish out the white $PbSO_4$ precipitate solids entrained in the organic extractant. It was apparent that this 2-M HNO_3 scrubbing to remove the $PbSO_4$ precipitate was required before recycling the organic.



Picture 5.0 Funnel #5 organic and aqueous after loading on the left and 200-g/L H_2SO_4 stripped organic (only) on the right.

H_2SO_4 acid recycle test (6 cycles)

The intent of this test was to see how many times the 200-g/L H_2SO_4 stripping solution for Cyanex 272 can be recycled before starting to lose its stripping efficiency. Previously it was noticed that if only recycled once, the acid solution is as effective as the fresh one, but more cycles were needed.

In the first cycle, 500-mL Cyanex 272 and 500-mL 200-g/L H_2SO_4 were shaken together for 10 minutes and 50-mL of each phase was taken for analysis. The remaining 450-mL was shaken again and a second 50-mL sample was taken from both organic and aqueous phase. The same procedure was continued, as shown in **Figure 6.0** and **Table 1.0**, for 6 cycles.

Figure 6.0. H_2SO_4 recycle test – 6 cycles

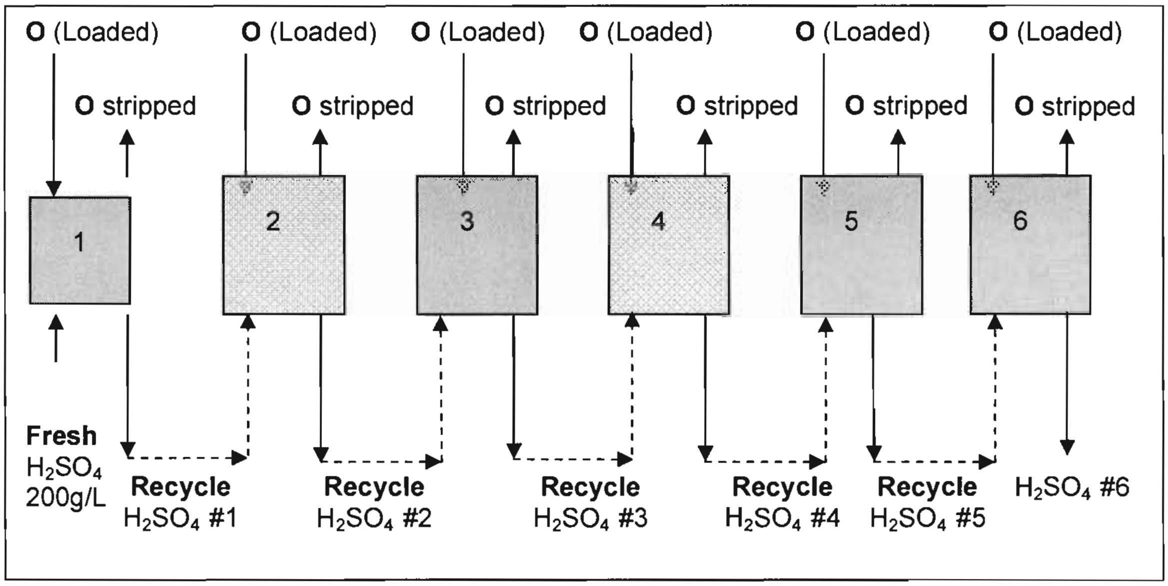


Table 1.0 H_2SO_4 stripping solution recycling – test condition

Cycle No.	**1**	**2**	**3**	**4**	**5**	**6**
Organic In	500-mL	450-mL	400-mL	350-mL	300-mL	250-mL
Aqueous In	500-mL	450-mL	400-mL	350-mL	300-mL	250-mL
O/A Ratio	1	1	1	1	1	1
Shaking Time	10-min	10-min	10-min	10-min	10-min	10-min
Phase Break	5-sec	8-sec	10-sec	8-sec	8-sec	8-sec
Complete Phase Separation	60-sec	50-sec	40-sec	30-sec	25-sec	80-sec
Colour/Clarity (org)	off white	off white	off white	off white	off white	off white
Colour (aq)	very light yellow	light yellow	yellow	darker yellow	dark yellow	dark yellow

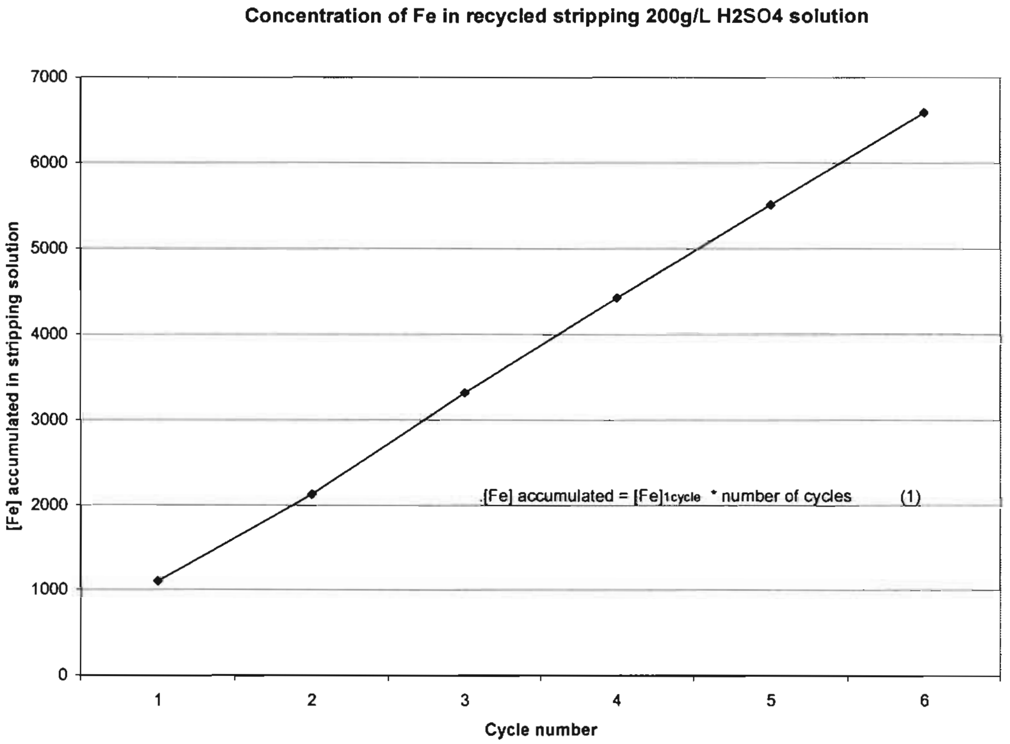
The stripping efficiency for the fresh H_2SO_4 solution was found previously to be greater than 98%. When the O/A ratio was kept constant at 1/1, greater than 98% of the Fe in the organic phase should transfer into aqueous phase. If during the first cycle, around 1101-mg/L Fe was found in the aqueous phase at the end of the 6th cycle almost six times as much Fe was found in the aqueous strip liquor solution (refer to **Table 2.0**).

Table 2.0 Analysis results for the aqueous and organic phase during sulphuric acid recycling test

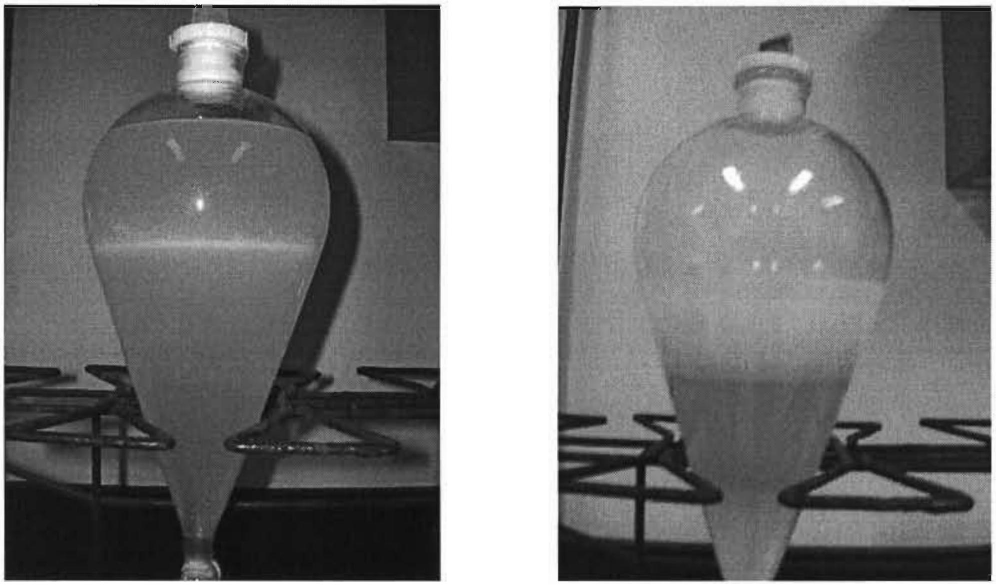
Cycle No			**1**	**2**	**3**	**4**	**5**	**6**
Fe	**Aqueous**	mg/L	*1101.34*	*2126.21*	*3319.22*	*4430.13*	*5509.5*	*6588.59*
	Organic	mg/L	-	-	30.72	34.65	24.67	48.93
Pb	**Aqueous**	mg/L	8.7	10.8	8.12	10.44	8.38	7.47
	Organic	mg/L	-	-	8.04	3.85	3.12	2.97
Al	**Aqueous**	mg/L	48.3	92.1	144.9	190.6	238.5	283.9
	Organic	mg/L	-	-	10.1	7.7	3.6	5.4
Mn	**Aqueous**	mg/L	4.42	8.39	13.09	17.28	21.55	25.66
	Organic	mg/L	-	-	0.12	0.07	0.05	0.14

A linear correlation was obtained when the concentration of Fe accumulated in the recycled strip solution was plotted versus the number of cycles performed - this can described using equation (1) in **Figure 7.0** below. Similarly, for Al and Mn, the amount of metal in aqueous solution after the 6th cycle was almost 6 times higher than in the first one, suggesting that the stripping with recycled H_2SO_4 is as effective as the stripping with fresh 200-g/L H_2SO_4.

Figure 7.0. Concentration of Fe accumulated in the stripping solution during H_2SO_4 recycling.



In **Picture 6.0** below, the color of the strip liquor becomes more yellow/brown as the H_2SO_4 is reused over six cycles. The Fe and Al accumulating in this strip liquor has not reached saturation and the free acid as H_2SO_4 was assayed to be approximately 186-g/L H_2SO_4 after reusing this strip solution six times. Thus, this strip solution has a potential to be used many more times before a reduction in stripping performance is observed. Further cycle tests will be performed at a later date.



Picture 6.0 Reusing 200-g/L H_2SO_4 strip solution over 6 cycles without make-up acid where the funnel to the left is cycle #1 stripping and the funnel to the right is cycle #6 stripping.

A possible future test program will be to determine if the strip liquor can be regenerated and put back into the stripping process. For example a low temperature ferric precipitation to remove iron.

50-g $Pb(NO_3)_2$ production with Purolite C104-H Ion Exchange Resin and As-Received Yava Leachate and Fe Removed Yava Leachate

To make approximately 50-g $Pb(NO_3)_2$ crystals for process verification we will need to target about 100-g $Pb(NO_3)_2$ due to the crystallization efficiency noticed in our first batch of $Pb(NO_3)_2$ made in the laboratory. In addition, the purest Pb(NO3)2 crystals are produced during the first crystallization heat cycle (where most of all impurities are with the saturated supernatant solution).

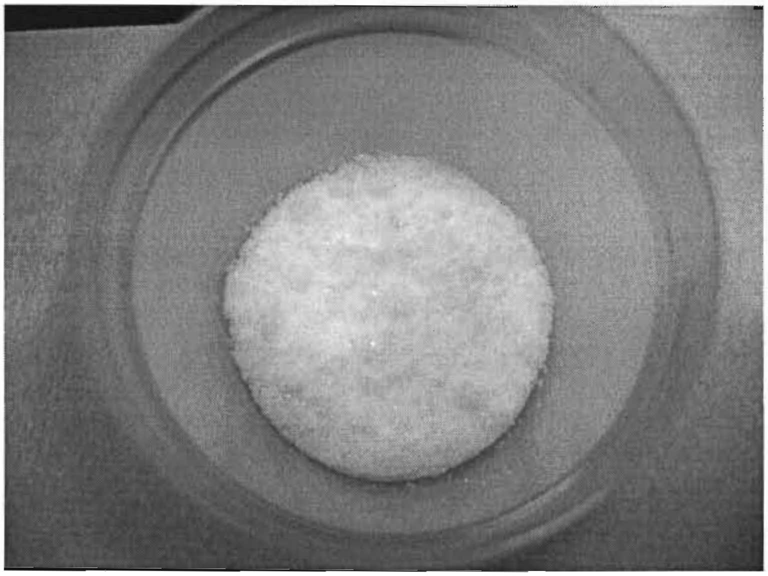
If we assume resin stripping efficiency to be 90%, Pb loading onto the resin to be 90% and Pb loses in the Fe removal step to be 10%, then we will required to treat approximately 11-L leachate solution at 8-g/L Pb. Another potential Pb loss will be during organic adsorption filtration of the aqueous raffinate (to remove organics that have carried over to the aqueous prior to ion exchange). It is estimated that 19-L of leachate is required with all the sampling and additional bench testing to establish rough resin loading conditions with Purolite C104-H (Resin Loading Scope Test).

From our leachate pH scope test with 10% by volume Cyanex 272 in Isopar-M diluent, we have observed that a leachate pH of approximately 2.8 would result in a good balance of iron removal and some impurity co-extraction (i.e. Al and Zn).

50 g of $Pb(NO_3)_2$ was obtained by loading Pb on Purolite C104 - H at an R/A ratio = 0.02 followed by stripping with 22-wt% HNO_3. Pb(NO3)2 crystals were produced by evaporating the strip liquor. The crystals were afterwards washed, filtered and dried in the oven over night. The final product contained impurities that included 384-ppm Fe, 155-ppm Al, 155-ppm Ca and 11-ppm Cu. Several solutions were provided to remove the impurities. Prior to loading on Purolite C104-H, Fe can be removed by Cyanex 272 solvent extraction. Once the Purolite C104-H resin is loaded, scrubbing this loaded resin with a 3-M CH_3COOH (acetic acid) can be employed to remove co-extracted Ca and Mn. The $Pb(NO_3)_2$ produced will be further purified by washing with deionized water and finally dried to produce a relatively pure $Pb(NO_3)_2$ product.

$Pb(NO_3)_2$ Purification test by washing

The raw $Pb(NO_3)_2$ crystals produced by evaporating the excess water in the Purolite C104-H ion exchange resin strip liquor resulted in some undesired impurities, such as Fe, Ca, Cu, Al, Mn and Zn. The raw $Pb(NO_3)_2$ crystals were washed with deionized water, filtered and dried in a low temperature oven before resubmitting a sample for analyses, shown in **Picture 7.0** below.



Picture 7.0 Deionized water washed $Pb(NO_3)_2$ crystals produced with Purolite C104-H ion exchange resin.

A impurity specification limits were received and compared with the $Pb(NO_3)_2$ product produced initially and after washing with deionized water in Table 3.0 below. The deionized water washed $Pb(NO_3)_2$ crystals achieved the impurity specification limits on all elements with the exception of Ca and Mn.

Table 3.0 $Pb(NO_3)_2$ crystal comparison of unwashed $Pb(NO_3)_2$ and washed $Pb(NO_3)_2$ with desired specifications provided.

Element	**Spec. Limit** (ppm)	**$Pb(NO_3)_2$** (ppm)	**Washed $Pb(NO_3)_2$** (ppm)	**Element**	**Spec. Limit** (ppm)	**$Pb(NO_3)_2$** (ppm)	**Washed $Pb(NO_3)_2$** (ppm)
Pb		62.26-%	62.49-%	**Hg**		<3	<3
Al	50	<100	<50	**Mo**		<1	<1
Sb	5	<5	<5	**Ni**	1	<1	<1
As	1	<5	<1	**P**		<100	<100
Ba		<2	<2	**K**		<100	<100
Bi		<2	<2	**Sc**		<1	<1
Cd	5	<0.2	<0.2	**Ag**	10	<0.1	<0.1
Ca	5	466	21	**Na**		122	<100
Cr		14	1	**Sr**		2	1
Co		<1	<1	**Tl**		<10	<10
Cu	10	10	3	**Ti**		<100	<100
Fe	5	<100	<5	**W**		<5	<5
La		<2	<2	**V**		<1	<1

Pb		622600	624900	**Zn**	5	1	1
Mg		<100	<100	**Zr**		<1	<1
Mn	1	44	3	**Sn**	1		

$Pb(NO_3)_2$ purification test by scrubbing (loaded IX-resin) with Acetic Acid prior to stripping.

After Fe removal by Cyanex 272 solvent extraction, an intermediate step was necessary to further remove Ca, Mn and remaining Fe in the leachate. Purolite C104H was loaded and then scrubbed with acetic acid in order to remove the undesirable metal ions. The concentrations of acetic acid used were 4, 6 and 10M, while the R/A ratio was kept constant at 1.25. The resulting aqueous solution was analyzed and the results are summarized in **Table 4.0**. Along with Ca, Mn, Mg and Al, the acid solution also removed Pb, which would not be a problem if the scrubbing solution is recycled.

The 4M acetic acid solution seems to be the best choice, removing less lead, but still separating 733mg/L Ca and 94mg/L Mn.

Table 4.0 Acetic acid scrubbing loaded Purolite C104-H resin scope test to remove impurities prior to stripping with 22-wt% HNO_3.

Element	Unit	4M CH_3COOH	6M CH_3COOH	10M CH_3COOH
Pb	ppm	3155.91	4811.02	7231.39
Al	ppm	22.3	33.6	42.6
Sb	ppm	<0.1	<0.1	<0.1
Ca	ppm	733.0	820.2	938.7
Cr	ppm	0.11	0.30	0.32
Cu	ppm	2.32	2.87	3.48
Fe	ppm	7.40	14.36	15.32
Mg	ppm	68.3	69.7	77.6
Mn	ppm	94.13	103.72	116.43

Purolite S940 Pb Separation Scope Test with Fe Removed Yava Leachate

The Purolite S940 resin equilibrium Pb-loading scope test has shown relatively low Pb-loading capabilities i.e. 50-g Pb per L-resin with little selectivity for Pb (Table 6.0). The test conditions are summarized in **Table 5.0**. No further testing will be conducted on this resin for the recovery of lead.

Table 5.0. Ion exchange removal of Pb by Purolite S940 test conditions.

	Feed	Beaker #1	Beaker #2	Beaker #3	Beaker #4	Beaker #5	Beaker #6
Resin		6-mL	9-mL	12-mL	15-mL	24-mL	36-mL
Aqueous		300-mL	300-mL	300-mL	300-mL	300-mL	300-mL
R/A Ratio		**0.02**	**0.03**	**0.04**	**0.05**	**0.08**	**0.12**
Initial pH	2.8	3.4	3.5	3.5	3.5	3.6	3.6
Final pH		3.6	3.7	3.7	3.8	3.8	3.9
Temperature		19°C	19°C	19°C	19°C	19°C	19°C
Time		24-hr	24-hr	24-hr	24-hr	24-hr	24-hr

Table 6.0. ICP data for Purolite S940 ion exchange test.

				Beaker #1	Beaker #2	Beaker #3	Beaker #4	Beaker #5	Beaker #6
R/A Ratio			**Feed**	**0.02**	**0.03**	**0.04**	**0.05**	**0.08**	**0.12**
Pb	**Resin**	%		26.07	24.45	24.73	23.57	20.58	16.85
	Aqueous	mg/L	7308	6012	5212	4467	3795	2336	1343
	Loaded $_{\text{Aq.}}$	%		**17.7%**	**28.7%**	**38.9%**	**48.1%**	**68.0%**	**81.6%**
	Resin Loading	g/L		64.8	69.9	71.0	70.3	62.2	49.7
Mn	**Resin**	g/t		2993	3275	3878	4131	4644	4279
	Aqueous	mg/L	214	200	180	163	144	89	47
	Loaded $_{\text{Aq.}}$	%		**6.5%**	**15.9%**	**23.6%**	**32.7%**	**58.6%**	**77.8%**
Fe	**Resin**	g/t		2855	2140	1857	1469	1102	687
	Aqueous	mg/L	29.71	6.5	4.39	2.74	1.8	0.86	0.63
	Loaded $_{\text{Aq.}}$	%		**78.1%**	**85.2%**	**90.8%**	**93.9%**	**97.1%**	**97.9%**
Al	**Resin**	g/t		902	676	521	483	295	194
	Aqueous	mg/L	12.2	4.5	4.3	4.1	4.2	3.8	3.3
	Loaded $_{\text{Aq.}}$	%		**63.1%**	**64.8%**	**66.4%**	**65.6%**	**68.9%**	**73.0%**

The performance of the Purolite S940 resin shows promise for selectivity of Pb at saturated conditions, as with the Purolite C104-H. However, the Pb loading of the Purolite S940 resin at 65-g Pb per L-resin is more than half the capacity of Purolite C104-H resin with similar R/A ratio loading at 150-g Pb per L-resin.

Bayer TP-207 (Na – form) pH and Equilibrium Isotherm Pb Separation Test Using Fe Removed Yava Leachate.

Bayer TP207 was used to load Pb from Fe removed Yava leachate and afterwards stripped with 22wt% HNO_3 to produce $Pb(NO_3)_2$. Several tests had to be performed to determine the optimum operation conditions. First, a pH scope test was carried out to decide if the pH of the Fe removed Yava leachate should be adjusted before loading. Second, an equilibrium isotherm was constructed to establish the R/A ratio and the number of stages necessary to achieve the best efficiency.

Bayer TP-207 (Na-form) pH Scope Test

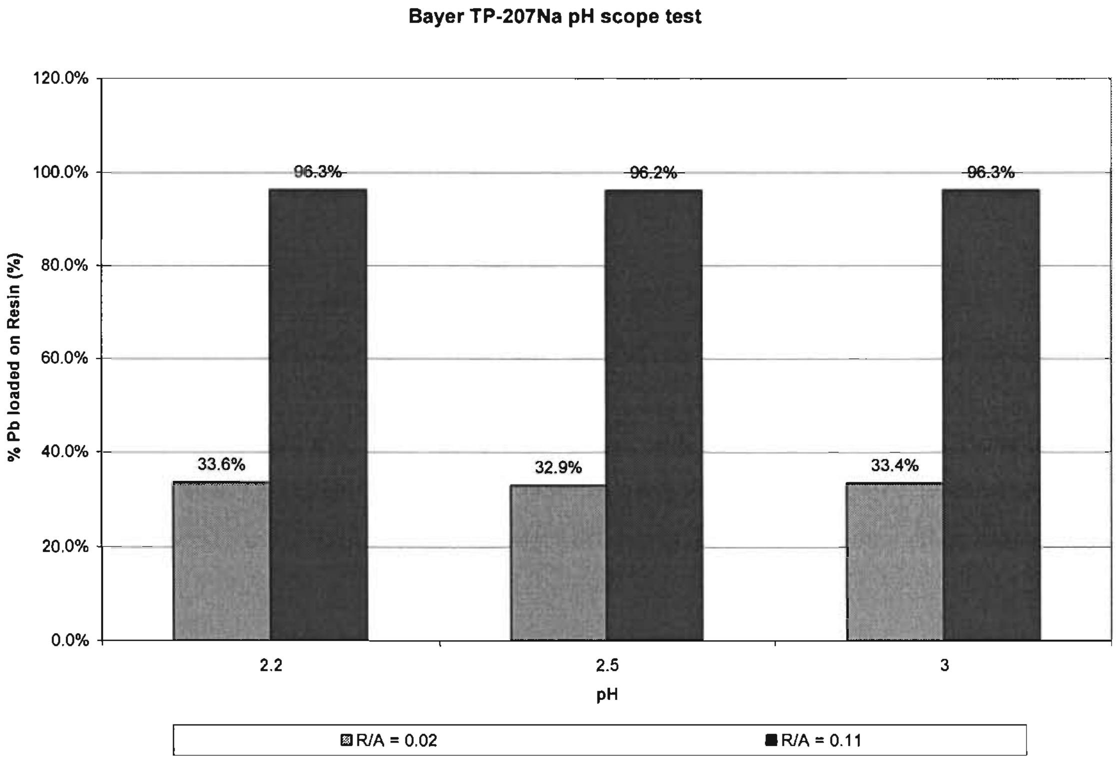
The test conditions are summarized in the Table 7.0. The natural pH was adjusted with 50%wt HNO_3 from 3 (the natural pH of the Fe removed Yava leachate) to 2.5 and 2.2 and the R/A was kept constant at 0.02 and 0.11, respectively.

Table 7.0 Bayer TP207-Na (sodium form) pH scope test conditions.

		Beaker #1	Beaker #2	Beaker #3	Beaker #4	Beaker #5	Beaker #6
Resin	mL	6	32	6	32	6	32
Aqueous	mL	300	300	300	300	300	300

R/A Ratio		**0.02**	**0.11**	**0.02**	**0.11**	**0.02**	**0.11**
Initial pH		3.0	3.0	2.5	2.5	2.2	2.2
Final pH		2.9	3.3	2.5	3.0	2.2	2.9
Temperature	°C	19	19	19	19	19	19
Time	Hr	24	24	24	24	24	24

Figure 8.0 Pb loading results at saturate (R/A = 0.02) and unsaturated (R/A = 0.11) conditions versus pH.



As seen in **Figure 8.0**, the pH value did not influence the Pb loading but slightly less Mn and Al were co-loaded at pH = 2.2.

Bayer TP207-Na (sodium form) equilibrium isotherm

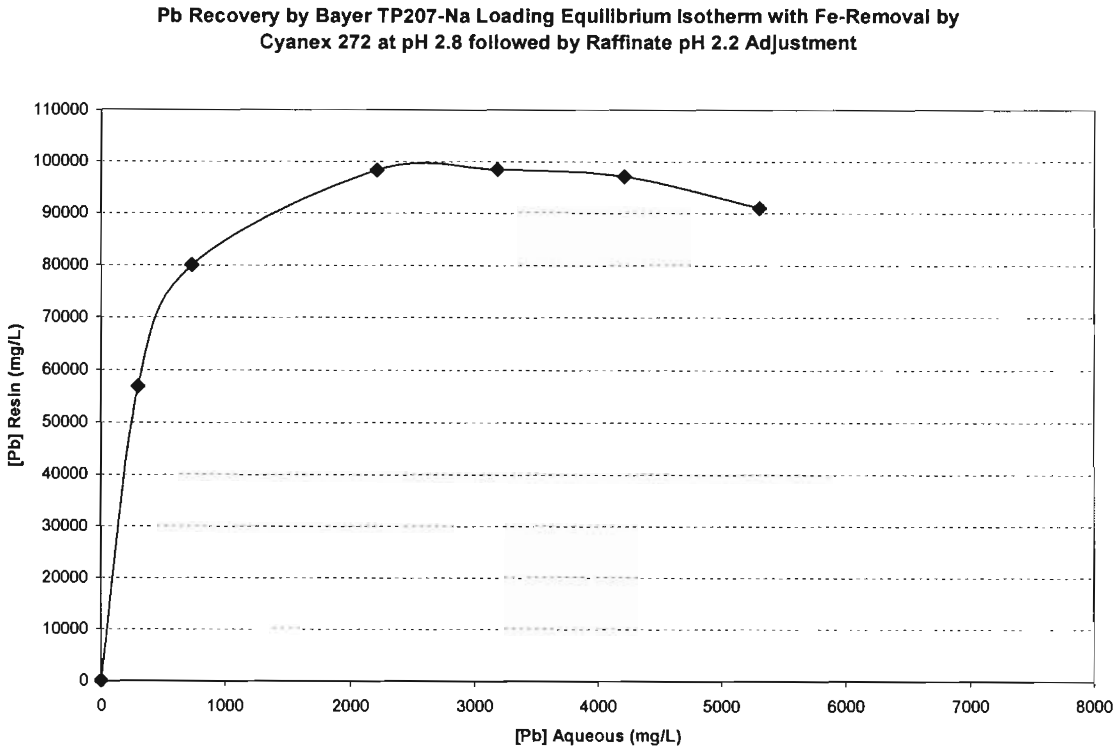
The experimental conditions for equilibrium Pb loading isotherm test are given in **Table 8.0** below. The results shown in Figure 9.0, will be used to determine the desired operating conditions in terms of R/A raito and number of stages.

Table 8.0 Bayer TP207-Na (sodium form) equilibrium Pb-loading conditions.

	Beaker #1	Beaker #2	Beaker #3	Beaker #4	Beaker #5	Beaker #6
Resin	8-mL	12-mL	16-mL	20-mL	32-mL	48-mL

Aqueous	400-mL	400-mL	400-mL	400-mL	400-mL	400-mL
R/A Ratio	**0.02**	**0.03**	**0.04**	**0.05**	**0.08**	**0.12**
Initial pH	2.2	2.2	2.3	2.4	2.6	2.9
Final pH	2.3	2.4	2.4	2.6	2.9	3.1
Temperature	19°C	19°C	19°C	19°C	19°C	19°C
Time	24-hr	24-hr	24-hr	24-hr	24-hr	24-hr

Figure 9.0 Bayer TP207-Na (sodium form) loading equilibrium isotherm with Cyanex 272 Fe-raffinate at pH 2.8 followed by pH 2.2 adjustment.



Bayer TP-207 stripping scope test with HNO3 at 10-wt%, 15-wt% and 20-wt%.

In order to reduce the amount of excess water to be evaporated during the lead nitrate crystallization step, the maximum tolerable nitric acid concentration would be desired. However, the Bayer TP207 resin is not compatible to oxidizing environments such as strong nitric acid (12-wt% HNO_3 maximum suggested by supplier). A scope test to evaluate the effects of over-oxidizing the resin was performed.

10-mL of wet pre-loaded Bayer TP207 resin was scooped into 3 separate 100-mL sample container. Into each of the 3 sample containers, containing the resin, 8-mL of nitric acid was added (strip R/A ratio = 1.25) with the concentration of 10-wt%, 15-wt% and 20-wt% HNO_3. Each sample was mixed for approximately 10-minutes and allowed to sit overnight.

Table 9.0 Selected metal assay results of stripped resin and strip liquor.

Sample	Type	Pb	Fe	Cu	Cr	Co	Ni	Mn	Zn

		(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)
10% - HNO_3	Resin	95894	3124	76	115	2	45	6	4
15% - HNO_3	Resin	20084	1289	52	93	4	<1	6	2
20% - HNO_3	Resin	19378	916	39	71	2	<1	8	1
10% - HNO_3	Soln.	13336	13.4	<0.01	0.31	3.13	4.39	331.7	7.96
15% - HNO_3	Soln.	33564	670.9	8.34	8.02	3.12	15.21	295.8	7.77
20% - HNO_3	Soln.	42828	946.2	13.37	14.15	3.96	19.30	326.0	9.71

Where a typical Bayer TP207-H loaded resin at an R/A ratio = 0.04 is approximately 270,000-ppm Pb or 27-wt% Pb (assayed). Using this average resin Pb loading assay of 27-wt% Pb, the calculated resin strip efficiencies based on the resin assays above (in **Table 9.0**) are 64%, 93% and 93% for 10-wt% HNO_3, 15-wt% HNO_3 and 20-wt% HNO_3, respectively.

It was determined that greater than or equal to 15-wt% HNO_3 was an excessive concentration for the Bayer TP207-H resin to tolerate when soaked overnight. The over-oxidized resin would clump and not move as freely as the test sample soaking in 10-wt% HNO_3 (suggested maximum, 12-wt% HNO_3). The resin treated with 20-wt% HNO_3 resulted in a pale yellow strip liquor which may have been the result of the decomposition of the Bayer TP207-H resin.

5-kg $Pb(NO_3)_2$ production layout / set-up & equipment and reagent procurement

Based on the experience accumulated during the 50-g $Pb(NO_3)_2$ bench test, the production of 5-kg $Pb(NO_3)_2$ was initiated in accordance to the procedure below.

The as received solution from SGS Lakefield Research must be first cartridge filtered into a stir tank reactor, where the pH will be adjusted to 2.8 with 50-wt% HNO_3 and then transferred for Fe removal by Cyanex 272 solvent extraction. After phase separation, the organic phase will be stripped with 200-g/L H_2SO_4 solution, scrubbed with 1-M HNO_3 and recycled back to loading. The solvent extraction Fe-raffinate aqueous solution is filtered through an organic filtration system to remove any entrained organic, the filtrate aqueous phase will undergo a Pb ion-exchange recovery by Purolite C104-H step, shown in **Figure 10.0**. During the ion-exchange step the resin will be Pb loaded, thoroughly washed with deionized water and stripped with 22-wt% HNO_3. The excess water in the strip liquor will be evaporated with heat and the $Pb(NO_3)_2$ crystals will be collected, washed with deionized water and dried to produce the final $Pb(NO_3)_2$ product.
The layout of the pilot plant is summarized in **Figure 11.0**.

Figure 10.0 Pb recovery by Purolite C104-H Ion exchange step – 3 stages.

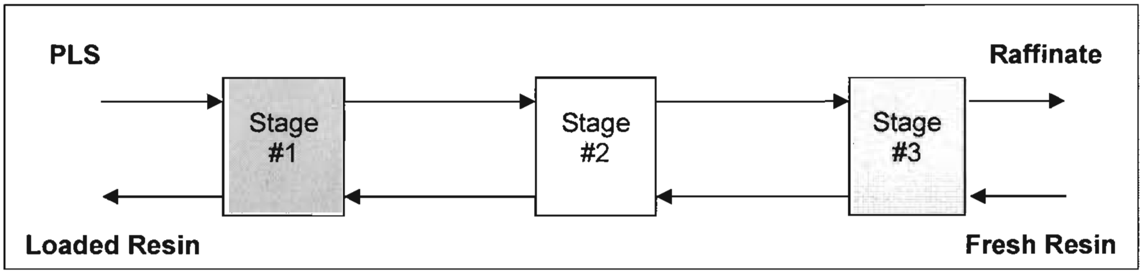
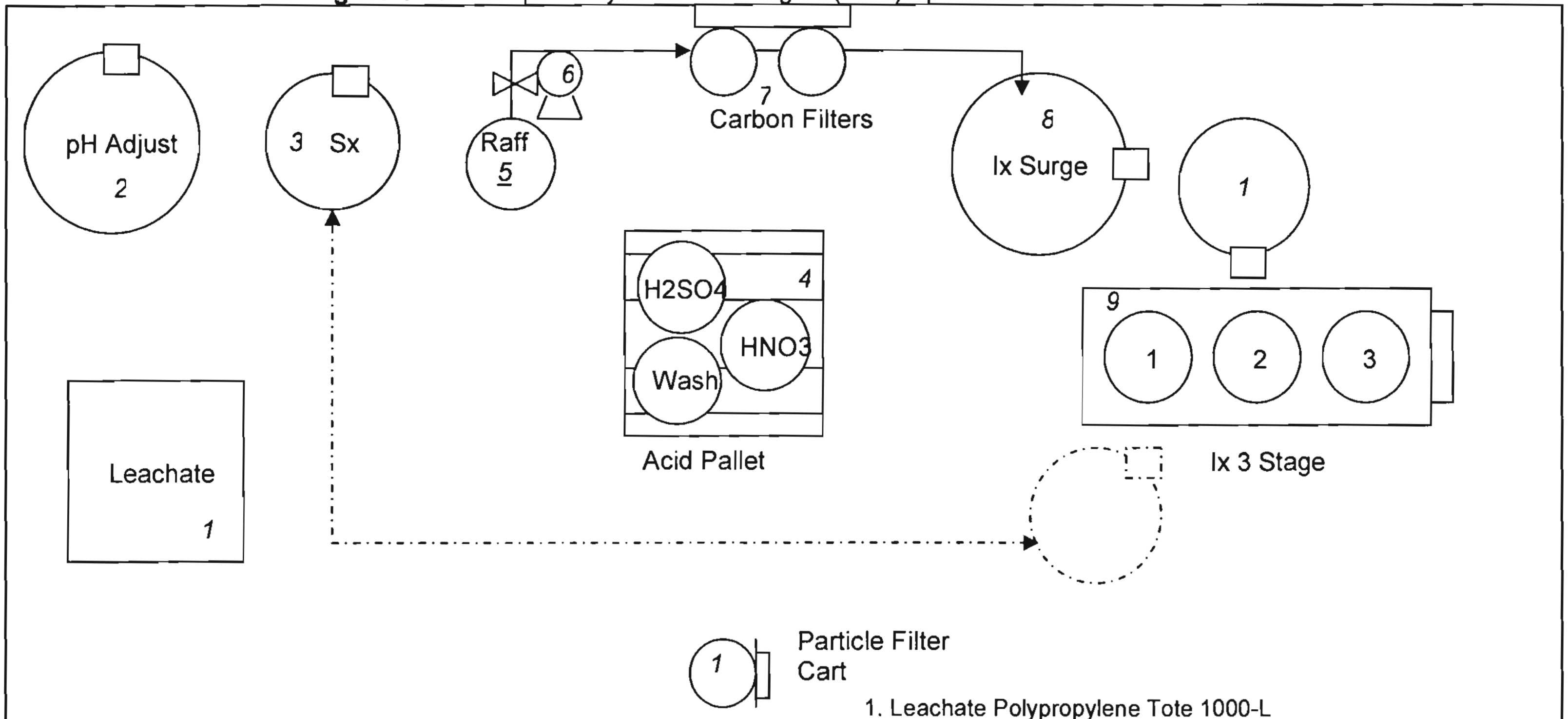

PLS
Raffinate
Stage #1
Stage #2
Stage #3
Loaded Resin
Fresh Resin

Figure 11.0 Pilot plant layout for the 5-kg Pb(NO3)2 production.



1. Leachate Polypropylene Tote 1000-L
2. pH Adjust Polypropylene Open Head Tank 1000-L
3. Sx Stir Polypropylene Open Head Tank Reactor 400-L
4. Acid Pallet with Two 200-L Drum for Nitric and Sulfuric Acid Storage
5. Raffinate 200-L Drum
6. Gear Pump
7. Dual Cartridge 20” Carbon Filter to Remove Organic
8. Ix Surge Polypropylene Open Head Tank Storage
9. 3 Stage Ix Drums 1, 2 & 3 200-L situated on a cart
10. 20 µm Particle 20” Filter Cart w/ Diaphragm Pump to Transfer Leachate and Acid

8

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Progress Report No 8

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC. DURING DECEMBER 2005

Prepared by

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INTRODUCTION

In preparation for the 5-kg Pb(NO3)2 production scheduled for January 2006, the Bayer TP207-Na had been conditioned with deionized water and converted to hydrogen form with 12-wt% HNO3. Loading isotherms and pH scope tests were conducted on the freshly obtained resin and on the ones recycled, to determine if the efficiency of the resin decreases after stripping. A stripping test was also performed on Bayer TP207Na to determine the stripping efficiency of 12wt% HNO_3 solution.

The Bayer TP260-H was suggested to be tested and compared to the Bayer TP207-H. A standard equilibrium loading isotherm pH scope test was performed with this resin on the Fe removed Yava leachate (removal by Cyanex 272 solvent extraction at pH 2.8).

A series of tests were devised to verify the pilot plant Cyanex 272 stripping/scrubbing procedure to minimize the use of acid required for regeneration. The loaded Cyanex 272 was initially stripped with H_2SO_4 and further scrubbed with 2M HNO_3 to remove the $PbSO_4$ precipitated. As an alternative, the loaded Cyanex 272 was first scrubbed with $0.5HNO_3$ to remove the Pb traces and then stripped with H_2SO_4. In both cases, supplementary washing with deionized water are necessary to remove the H_2SO_4 entrapped in the organic phase.

The pilot plant was set up for the 5-kg lead nitrate production and the necessary solutions and resin (Purolite C104-H) were prepared.

SUMMARY

1. The Bayer TP-Na form resin was converted to H form by using 12 wt% HNO_3 and further regenerated using the same method. 12 wt% seemed to perform well during the stripping, the only possible problem being the accumulation of Cr in the recycled resin that needs additional monitoring.
2. The fresh and recycled Bayer TP-207 loading isotherm tests showed that the two resins perform similarly with the exception that the recycled one seems to accumulate Cr and Mn in time. Further investigation is required.
3. A beaker test was performed to determine the stripping efficiency of 12wt% HNO_3 on loaded Bayer TP-207 (Na form).

4. pH scopes tests were carried out to compare the efficiency and the selectivity of Bayer TP-207 (H form) and Bayer TP-260 (H form). Though the two resins exhibited similar behaviours, TP-207 showed a higher loading capacity and it was chosen for further tests.
5. In order to minimize the acid consumption, a sequence test was conducted. The loaded Cyanex 272 was first stripped with 200g/L H_2SO_4 to recover Fe and scrubbed with 2M HNO3 to remove the Pb traces in the organic phase. As a better and acid saving alternative, an initial scrubbing of the loaded Cyanex 272 with 0.5M HNO_3, followed by stripping with 200g /L H_2SO_4 was proposed. As a third step, additional washing with DI water was required to remove any entrained H_2SO_4.
6. The pilot plant was set up and tested for leaks and flow in order to be able to start production of 5-kg lead nitrate in January. Reagents, resin and washing/stripping solutions were also prepared.

RESULTS AND DISCUSSION

Converting Bayer TP-207 (Na form) to Bayer TP-207 (H form).

It was discovered during the analyses of the resin and aqueous assays that the original batch of Bayer TP207 resin received from the supplier was in the sodium form. Thus, some tests were required to be repeated with the fresh Bayer TP207-Na (sodium form) converted to hydrogen form (Bayer TP207-H) by 12-wt% HNO_3 at a R/A ratio = 0.5 for overnight under gentle mixing. Then the resin must be thoroughly washed with deionized water before use. The resin can be stored immersed in deionized water. The Bayer TP207 resin assays of significance are provided in Table 1 at the specified conditions.

Table 1. Bayer TP207 resin assay as, Fresh – As-Received Resin, Freshly Converted to Hydrogen Form Resin, and Recycle Regenerated Hydrogen Form Resin.

Sample	Pb (ppm)	Na (ppm)	Cu (ppm)	Cr (ppm)
Fresh – As-Received Resin	83	89540	7	<1
Freshly Converted – H Resin	72	119	15	1
Recycle Regenerated - H Resin	259	<100	13	53

The obvious difference is the sodium content in the Fresh As-Received resin which has shown to cause some drifting in pH during the equilibrium loading. This pH movement is not desired since metal loading characteristics are dependant of pH. Thus, it would be difficult to attribute loading performance differences (i.e. selectivity and loading capacity) to the loading Resin/Aqueous ratio from pH changes.

It is also apparent the some lead and chromium are not stripped during the 12-wt% HNO_3 acid treatment. This will need to be monitored further as we move into cycle loading and stripping tests where we can observe the accumulation of metals on the resin after several cycles.

Fresh Bayer TP-207 (H form) Pb loading isotherm tests.

A single beaker Bayer TP207-H (Freshly Converted to Hydrogen Form) equilibrium Pb-loading test at an R/A ratio = 0.02 with a Fe removed Yava leachate followed by a pH 2.0 adjustment with 50-wt% HNO_3 was performed and used a comparison to the Recycle Bayer TP207-H Pb-loading isotherm test below.

Recycled Bayer TP-207 (H form) Pb loading isotherm tests.

Some loaded Bayer TP207-Na (Sodium Form) resin was stripped with 12-HNO_3 at an R/A ratio = 1.25 and thoroughly washed with deionized water. This stripped and washed resin was then regenerated with 12-wt% HNO_3 with an R/A ratio = 0.5 in 3 stages or 3 times (batch). After which, the resin wash thoroughly washed with deionized water and used in this Recycle Bayer TP207-H (Hydrogen Form) Pb-loading isotherm test. The results are summarized in Table 2.

Table 2. Loaded Bayer TP207-H (Hydrogen Form) beaker #1 equilibrium Pb-loading at an R/A = 0.02 with Fresh resin and Recycled resin.

Sample	**Pb** (ppm)	**Fe** (ppm)	**Cr** (ppm)	**Mn** (ppm)	**Ni** (ppm)	**Zn** (ppm)
Fresh - resin	246106	1267	40	19	117	127
Recycled - resin	235843	1379	80	2	121	119
Fresh - solution	5052.52	1.46	5.72	260	1.17	1.25
Recycle - solution	4998	1.93	5.79	264	1.33	1.49

Loaded Bayer TP207-H (Hydrogen Form) Pb-assays were observed to be similar with the exception of Cr and Mn. A fresh Bayer TP207-H resin seems to load Mn more than the recycled Bayer TP207-H and Cr appears to be accumulating on the resin. The results are not yet conclusive and several more cycles will be required to form a definitive conclusion.
The Pb loading isotherm for the recycled Bayer TP-207H is presented in **Figure 1**.

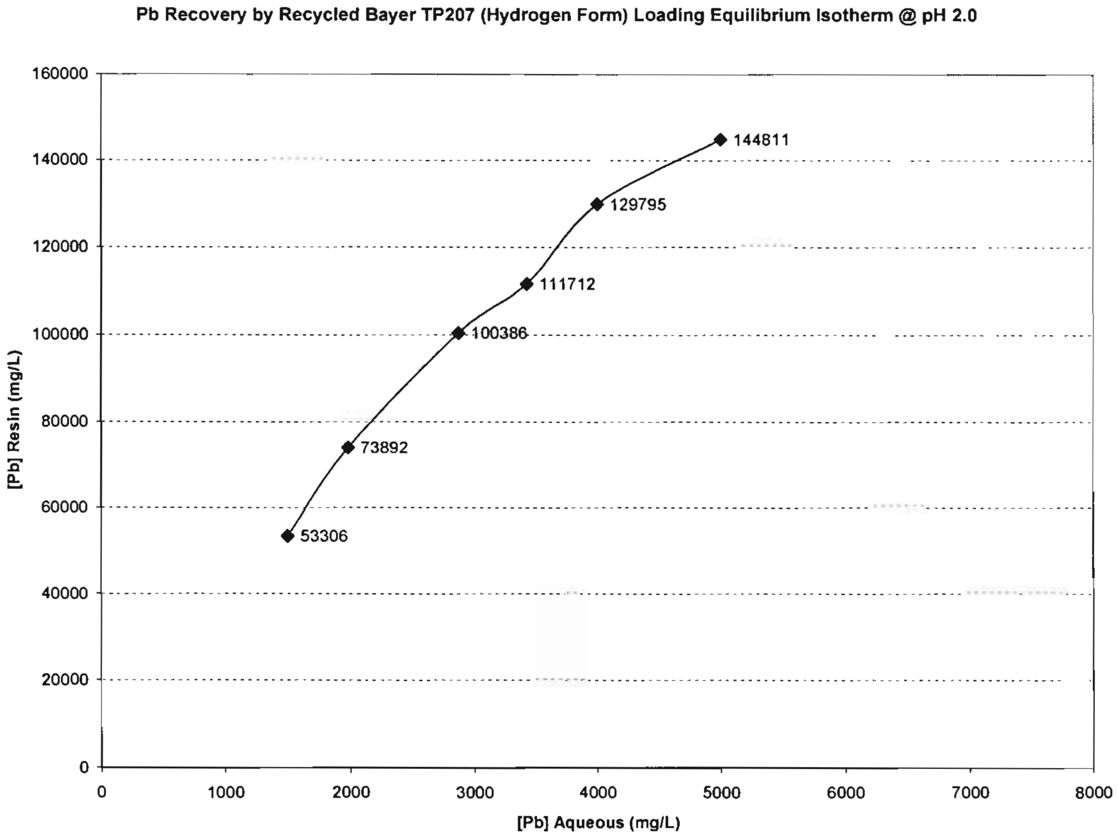


Figure 1. Recycled Bayer TP207-H equilibrium loading isotherm at pH 2.0.

Bayer TP207-Na (sodium form) stripping scope tests

Different resin/aqueous volume ratios of loaded resin and acid solution were tested in order to investigate the Pb-stripping efficiency of a 12-wt% HNO_3 on loaded Bayer TP207-Na in sodium form. The resin was obtained in a previous experiment, by loading Bayer TP207-Na resin with Cyanex 272 Fe-raffinate at pH 2.8, at an R/A ratio = 0.04. The stripping test conditions are summarized in Table 3 and results summarized in Table 4.

Table 3. Experimental conditions for Bayer TP207-Na stripping test.

	Beaker #1	**Beaker #2**	**Beaker #3**
Resin	30-mL	50-mL	100-mL
Aqueous	200-mL	200-mL	200-mL
R/A Ratio	**0.15**	**0.25**	**0.50**
Temperature	19°C	19°C	19°C
Time	4-hr	4-hr	4-hr

Table 4. Bayer TP207-Na (sodium form) stripping with 12-wt% HNO_3.

				Beaker #1	**Beaker #2**	**Beaker #3**
R/A Ratio			**Feed**	**0.15**	**0.25**	**0.50**
Pb	**Resin**	%	27.10	0.58	0.91	1.90
	Aqueous	mg/L		19868	29988	51670
	Stripped $_{resin}$	%		**97.9%**	**96.6%**	**93.0%**
Mn	**Resin**	g/t	26	<1	<1	<1
	Aqueous	mg/L		0.21	0.27	0.51
	Stripped $_{resin}$	%		**96.2%**	**96.2%**	**96.2%**
Fe	**Resin**	g/t	6232	449	780	2590
	Aqueous	mg/L		541.98	825.02	1133.22
	Stripped $_{resin}$	%		**98.4%**	**87.5%**	**58.4%**

As expected, with increasing R/A ratio, the stripping efficiency with 12-wt% HNO_3 is decreasing. At an R/A ratio = 0.25, 96.6% of Pb, 96.2% of Mn and 87.5% of Fe were removed, values that were within our stripping requirements. Thus, stripping loaded Bayer TP207-Na at an R/A ratio of 1/4 with 12-wt% HNO_3 was satisfactory. In comparison to stripping loaded Purolite C104-H, an R/A ratio of 5/4 with a 22-wt% HNO_3 was required. Unfortunately, the Bayer TP207-H resin beads degrades (by oxidation) when exposed to a strong nitric acid concentration (greater than 12-wt% HNO_3).

Bayer TP207-H (hydrogen form) and Bayer TP260-H (hydrogen form) pH scope tests

The purpose of this test was to investigate the effects of pH on Pb loading and selectivity of a new resin, Bayer TP260-H in comparison to Bayer TP207-H previously tested. The conditions used during the loading test are summarized in Tables 5 and Table 6. The R/A ratio was maintained at 0.02 (saturated) and 0.11 (unsaturated), while the pH varied from 2.8 to 3.0 (the pH of the Cyanex 272 -Fe raffinate) to 2.5 and 2.2.

The test procedure is as follows:

1. Scoop an adequate amount of Bayer TP260 or Bayer TP207 resin into a 500-mL beaker and wash thoroughly with deionized water. Decant water and fill beaker near the top. Allow resin to soak overnight.
2. Weigh and label 6 clean and dry 500-mL Pyrex beaker.
3. Measure appropriate amount of wet settled resin into each respective beaker and decant excess water.
4. Measure 300-mL of pH adjusted (with 50-wt% HNO_3) Fe-Ion exchange raffinate solution into each beaker. Take a 50-mL sample of this feed solution for analysis.
5. Place the six beakers into a gang mixer and measure the initial pH of each beaker. Allow to mix for 24 hours.
6. After 24 hours, measure pH of each beaker. Stop the gang mixer and remove beakers. Make note of any colour changes in each beaker.

7. Filter resin in Buchner funnel with filter paper and collect a sample of each filtrate for analysis. Store remaining filtrate combined in 1000-mL sample bottle and label.
8. Thoroughly wash the resin with deionized water and collect a sample of the resin for analysis. Store remaining resin in sample containers, submerged in deionized water, separately and labeled. Place the resin sample in a low temperature oven for at least 24 hours.
9. Submit feed solution, raffinate filtrate and dry resin samples for ICP metals (H2O method), Pb assay by AA/ICP and Fe assay by AA/ICP.

Table 5. Pb removal by ion exchange with Bayer TP260-H test condition.

	Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**	**Beaker #5**	**Beaker #6**
Resin	6-mL	32-mL	6-mL	32-mL	6-mL	32-mL
Aqueous	300-mL	300-mL	300-mL	300-mL	300-mL	300-mL
R/A Ratio	**0.02**	**0.11**	**0.02**	**0.11**	**0.02**	**0.11**
Initial pH	2.8	2.8	2.8	2.8	2.8	2.8
50-wt% HNO_3	None	None	3-g	3-g	4-g	4-g
pH Adjusted	2.8	2.8	2.5	2.5	2.2	2.2
Final pH	2.7	2.0	2.3	1.5	1.9	1.3
Temperature	20.5°C	20.5°C	20.5°C	20.5°C	20.5°C	20.5°C
Time	24-hr	24-hr	24-hr	24-hr	24-hr	24-hr

Table 6. Pb removal by ion exchange with Bayer TP207-H test condition.

	Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**	**Beaker #5**	**Beaker #6**
Resin	6-mL	32-mL	6-mL	32-mL	6-mL	32-mL
Aqueous	300-mL	300-mL	300-mL	300-mL	300-mL	300-mL
R/A Ratio	**0.02**	**0.11**	**0.02**	**0.11**	**0.02**	**0.11**
50-wt% HNO_3	None	None	3-g	3-g	4-g	4-g
pH Adjusted	3	3	2.5	2.5	2.2	2.2
Final pH	2.7	2.3	2.0	1.6	1.9	1.5
Temperature	20.7°C	20.7°C	20.7°C	20.7°C	20.7°C	20.7°C
Time	24-hr	24-hr	24-hr	24-hr	24-hr	24-hr

As seen in **Figures 2** and **Figure 2**, the two resins exhibit a similar behaviour. Though Bayer TP260 is a viable alternative, Bayer TP207 was chosen for further tests, for its higher loading capacity (175 g/L) and slightly better efficiency at R/A ratio = 0.02.

Figure 2. Bayer TP207H – Pb recovery as a function of pH

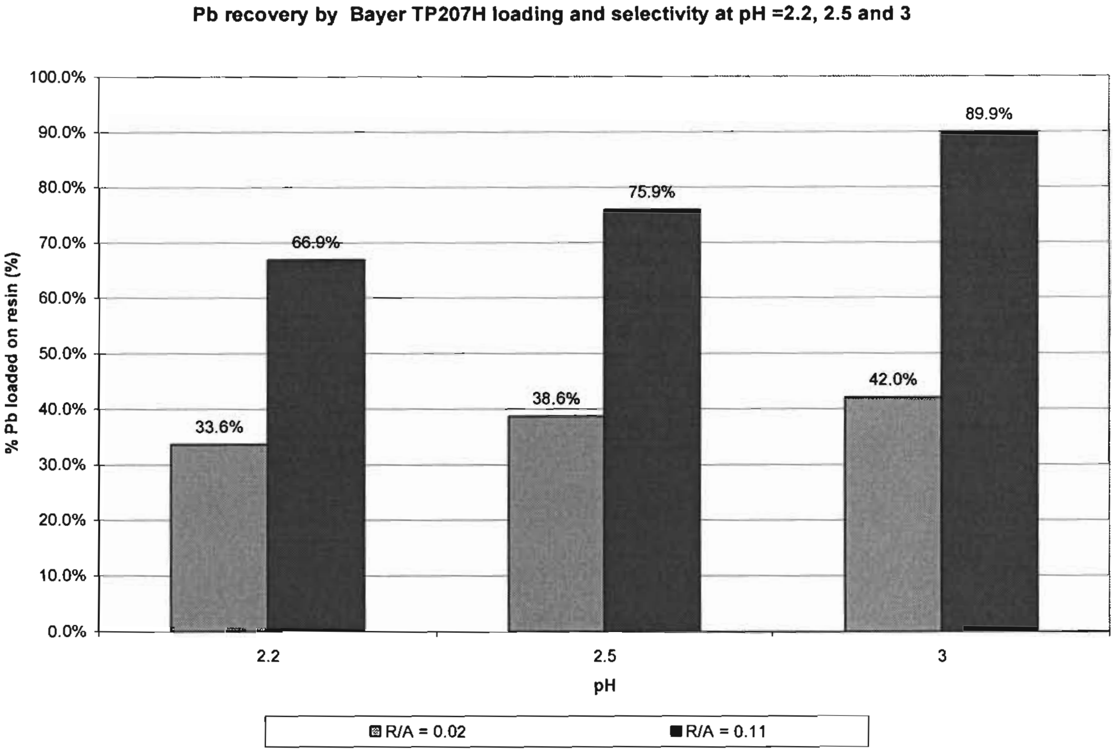
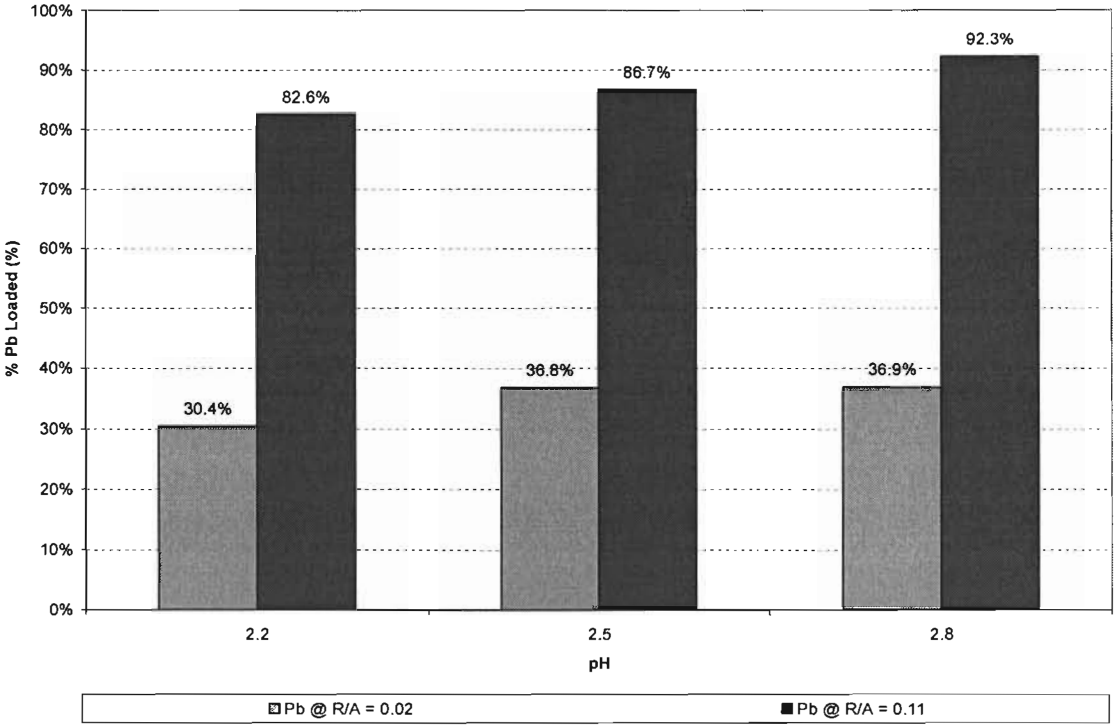


Figure 3. Bayer TP260H – Pb recovery as a function of pH

Pb Recovery by Bayer TP260 (Hydrogen Form) Loading and Selectivity at pH 2.8, 2.5 and 2.2



Loaded Cyanex 272 stripping and scrubbing – sequence test

The sequence in which the stripping and scrubbing with H_2SO_4 and HNO_3 should take place was investigated. Once the Fe loaded, Cyanex 272 is ready to be stripped using H_2SO_4. Due to co-extracted Pb in the organic phase, when treated with H_2SO_4, a white $PbSO_4$ precipitate is formed and scrubbing the organic phase with 2-M HNO_3 becomes necessary. An alternative would be to remove the co-extracted Pb first, using a 0.5-M HNO_3 followed by stripping the Fe from the Cyanex 272 with a 200-g/L H_2SO_4. With this 0.5-M HNO_3 scrubbing and 200-g/L H_2SO_4 stripping scheme, entrained H_2SO_4 solution after Fe stripping would be extremely detrimental when in contact with the pH 2.8 adjusted leachate solution (high concentration of Pb). Thus, an organic water wash to minimize H_2SO_4 carry-over would be required. Test results are summarized in Table 7.

Table 7. Analytical Result for Elution Sequence Testing.

	[Fe]$_{Org.}$ (mg/L)	**[Pb]$_{Org.}$** (mg/L)
Loaded Organic (Cyanex 272)		
Stripped (200-g/L H_2SO_4) then Scrubbed (2-M HNO_3)	15.37	<0.05
Scrubbed (0.5-M HNO_3) then Stripped (200-g/L H_2SO_4)	10.23	<0.05
Scrubbed, Stripped and the Washed		

It was concluded that if you scrub the Pb off the Fe-loaded Cyanex 272 with a 0.5-M HNO_3 before stripping the Fe with 200-g/L H_2SO_4, then the formation of $PbSO_4$ can be avoided or minimized. This is better than the alternative method of stripping the Fe with 200-g/L H_2SO_4 and dissolving the $PbSO_4$ precipitates with a 2-M HNO_3 scrub. The net result was a significant reduction in nitric acid requirement.

The 5-kg $Pb(NO_3)_2$ production pilot plant run scheduled for January 2006 will incorporate these findings to reduce the consumption of nitric acid. A wash cycle will also be used to avoid entrained sulfuric acid back into the loading cycle where it will cause $PbSO_4$ precipitation. A photograph of this experimental work is shown in Figures 4 and 5.

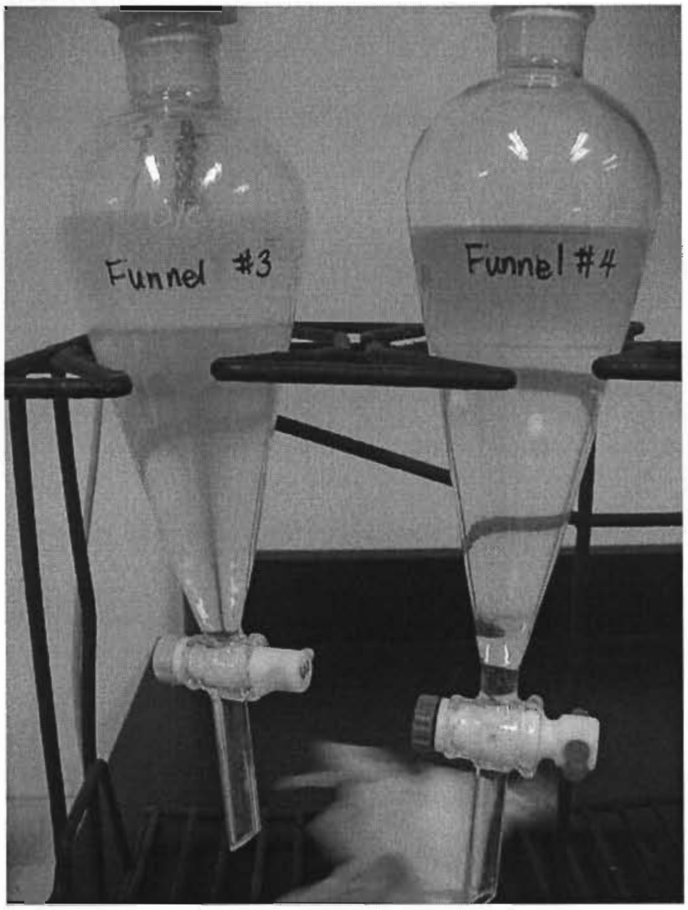


Figure 4. Funnel #3 has been stripped with 200-g/L H_2SO_4 and Funnel #4 has been scrubbed with 0.5-M HNO_3.

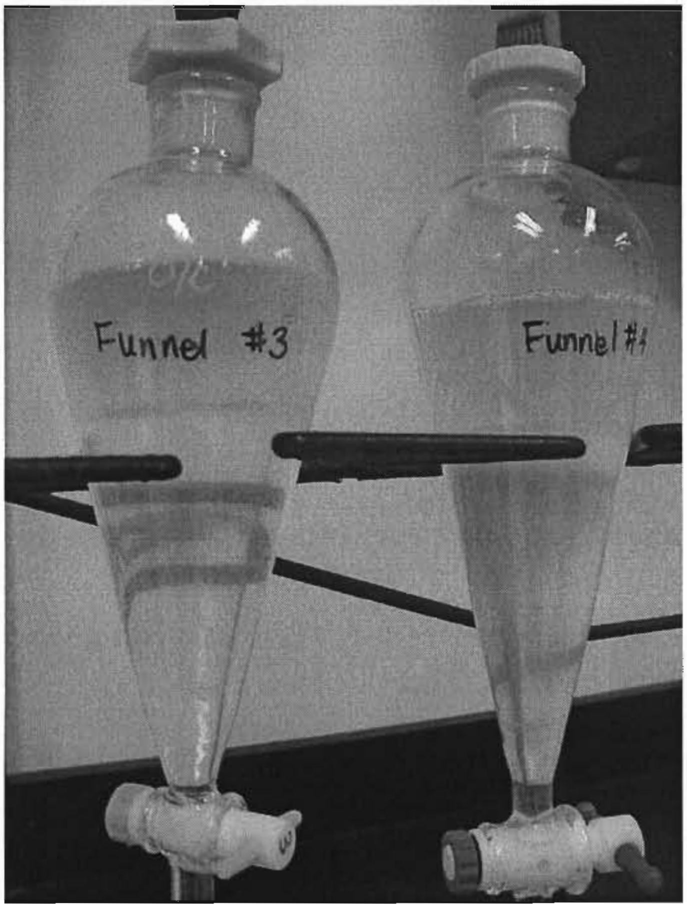


Figure 5. Funnel #3 has been stripped with 200-g/L H_2SO_4 followed by scrubbing with 2-M HNO_3 and Funnel #4 has been scrubbed with 0.5-M HNO_3 followed by stripping with 200-g/L H_2SO_4.

The stripped organic by both methods appeared very similar – opaque, pale yellow to cream in colour.

The purpose of washing the stripped Cyanex 272 organic extractant was to remove any entrained H2SO4 prior to recycling the organic to loading. Thus, this will minimize the PbSO4 formation if any H2SO4 entrainment has occurred.

Pilot Plant set up for 5-kg Lead Nitrate production.

The pilot plant was set-up such that the 5-kg Pb(NO3)2 production can be started immediately in January 2006. Piping, tubing and pumps were set up and water tested for leaks and flow. Along with the resin, Purolite C104 –H, which was conditioned with deionized water, acid solutions were prepared (200-g/L H2SO4, 0.5-M HNO3 and 50-wt% HNO3). The as-received leachate was pumped into the 1000-L stir tank, ready for pH adjustment and the wash water drum was filled.

9

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Progress Report No 9

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC. DURING JANUARY 2006

Prepared by

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and
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YAVA PILOT PLANT

MONTHLY PROGRESS REPORT NO 9 – January 2006

INTRODUCTION

In the month of January 2006, the production of 5 kg $Pb(NO)_3$ was conducted based on the procedures developed previously.

SUMMARY

1. The pilot production of 5 kg of Pb(NO3)2 was conducted.
2. The Pb(NO3)2 crystal product was submitted for purity analysis.

RESULTS AND DISCUSSION

5 Kg Pb(NO3)2 Crystal Production – Pilot Plant Test

The pilot plant production of 5 Kg of Pb(NO3)2 crystal was conducted. Appendix A summarizes the pilot plant layout and the procedures followed during this trial.

pH Adjustment, Solvent Extraction

The pH of 3 batches of as-received Yava leachate was adjusted with 50 wt% HNO_3 to 2.8. Because the 3 batches of Yava leachate possessed similar assays, for simplicity, the average of the assays of the three batches will be used as the representative Yava leachate for further calculations. The leachate assay results are summarized in Table 1.0.

Table 1.0 Analysis for Each of 3 As-Received Leachate Batches

	Pb mg/L	**Al mg/L**	**Ca mg/L**	**Cr mg/L**	**Fe mg/L**	**Mn mg/L**	**Zn mg/L**
Leachate Batch # 1	8420.49	72.70	1449.30	8.55	1129.56	227.93	14.06
Leachate Batch # 2	8198.86	70.90	1468.30	8.40	1118.16	231.88	14.17
Leachate Batch # 3	8072.68	68.80	1489.10	8.28	1097.67	229.95	13.60
Average	8230.70	70.80	1468.90	8.40	1115.10	229.90	13.90

After pH adjustment, the Yava leachate was treated with Cyanex 272 solvent extraction for Fe removal. In each batch, 160 L of leachate could be treated. As such, 12 cycles were necessary to treat all the leachate. On average, during the 12 cycles, more than 98% of Fe was removed. Other metals that could contaminate the final product, such as Zn and Al were also removed. As shown in Figure 1.0. Less than 10% of Pb was co-extracted as well. This that did not raise concerns because the treated Cyanex 272 organic phase was recycled after regeneration.

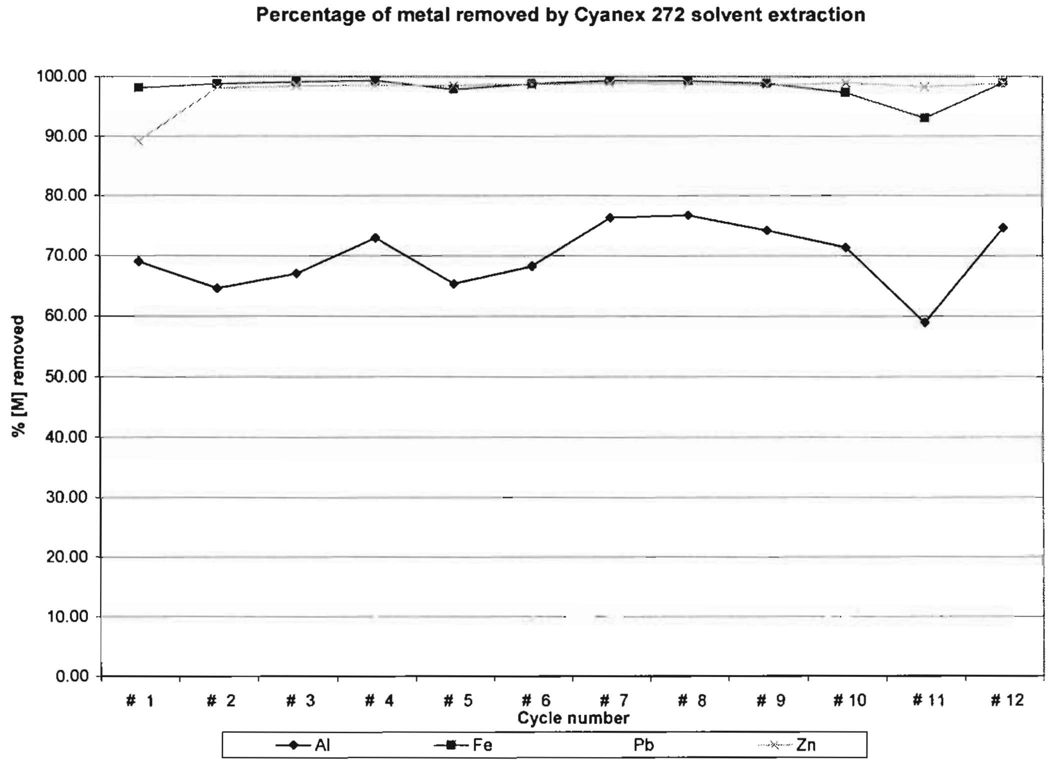

Figure 1.0 Metal removal by Cyanex 272 solvent extraction

In prior tests, the regeneration of the Cyanex 272 organic phase was done by first stripping with 200g/L H_2SO_4 followed by scrubbing with 2M HNO_3. 10 cycles of Cyanex 272 regeneration were conducted with recycled acids (allowing the stripped/scrubbed metals to accumulate in the eluent) showing regeneration was successful with no detrimental effect observed with accumulation of metals in the H2SO4 and HNO3 (for 6 cycles).

In order to obtain more experimental data during the 5 kg Pb(NO3)2 pilot production, Cyanex 272 regeneration was carried out by first scrubbing with recycled 0.5 M HNO_3 and then stripping with recycled 200 g/L H_2SO_4. The results summarized in Figure 2.0 demonstrate that both regeneration methods have similar efficiencies even after 10 cycles.

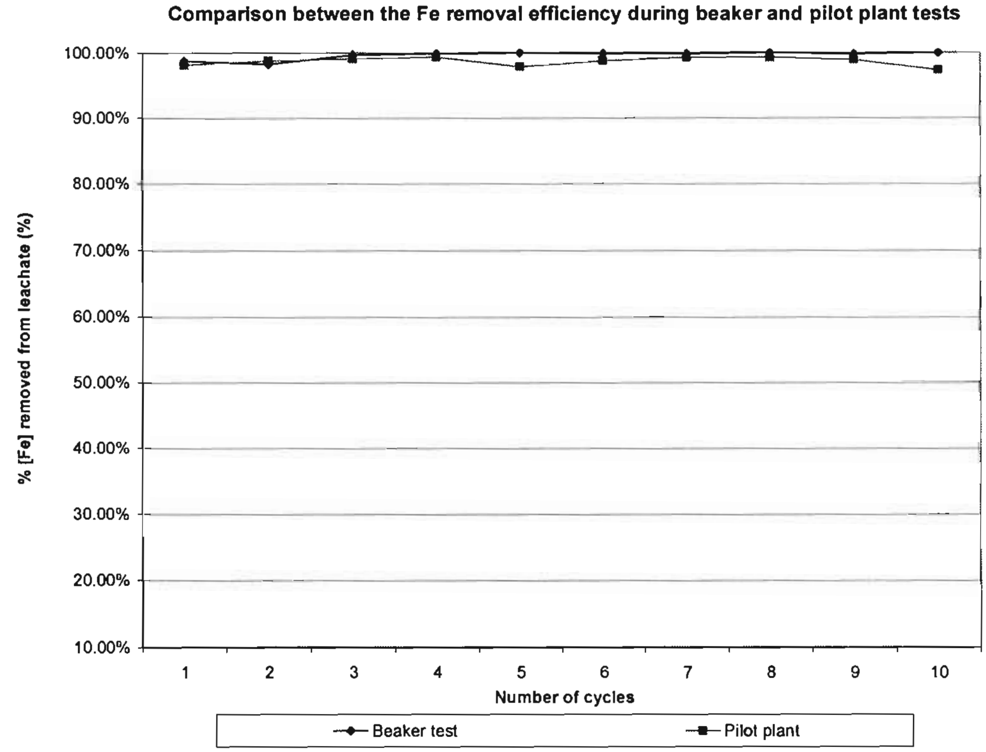

Figure 2.0 Efficiency comparisons between Cyanex 272 regenerated by H2SO4 stripping/HNO3 scrubbing (beaker test) and HNO3 scrubbing/H2SO4 stripping (pilot plant test)

Completion of 5 Kg Pb(NO3)2 Crystal Production

The batch 5 Kg Pb(NO3)2 crystal production was completed according to the procedure in Appendix A. The ICP analysis of the initial $Pb(NO_3)_2$ crystals produced are summarized in Table 2.0. Supplementary washing found to be necessary to removed Ca and better detection limit was required for Al and Fe. This work will be conducted in February 2006.

Table 2.0 The ICP analysis of $Pb(NO_3)_2$ Produced in the Pilot Plant

Element	**Spec.Limit** (ppm)	**$Pb(NO_3)_2$** (ppm)	**Element**	**Spec. Limit** (ppm)	**$Pb(NO_3)_2$** (ppm)
Pb		63.4-%	**Hg**		<3
Al	50	<100	**Mo**		<1
Sb	5	<5	**Ni**	1	<1
As	1	<5	**P**		<100
Ba		<2	**K**		<100
Bi		<2	**Sc**		<1
Cd	5	<0.2	**Ag**	10	<0.1
Ca	5	132	**Na**		<100
Cr		4	**Sr**		4
Co		<1	**Tl**		43
Cu	10	1	**Ti**		<100
Fe	5	<100	**W**		<5
La		<2	**V**		<1
Pb		635406	**Zn**	5	2
Mg		<100	**Zr**		<1
Mn	1	10	**Sn**	1	<10

Appendix A

Report to Yava on 5-Kg Lead Nitrate Crystal Production Pilot Plant Layout and Procedures

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January 4, 2006

LAYOUT

Figure 1 illustrated the overall top view for the 5-kg lead nitrate production pilot plant.

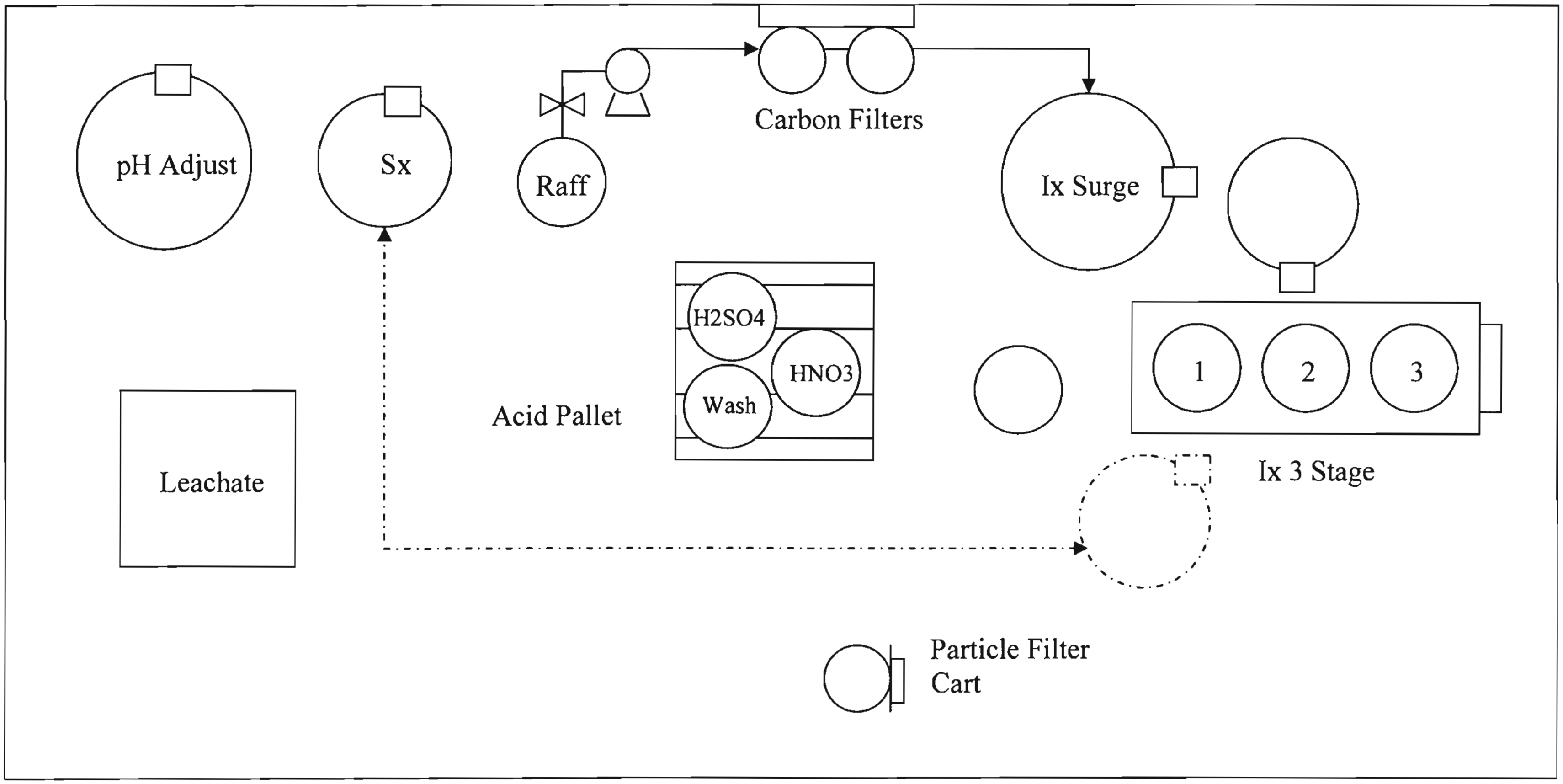


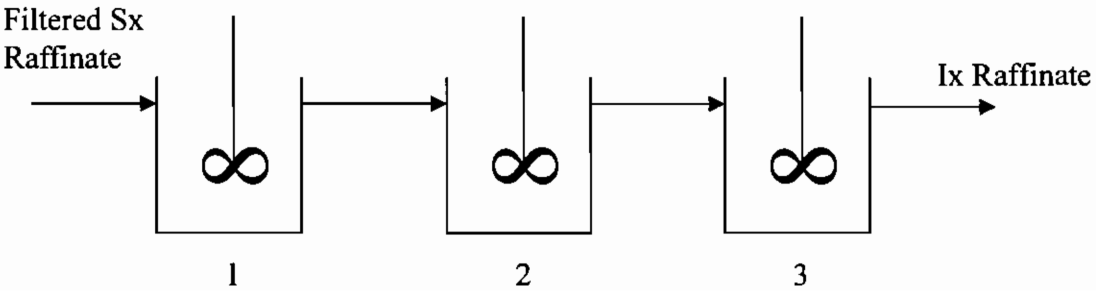
Figure 1- Top View of Pilot Plant Layout

The specifications for the pilot plant are outlined below:

- Leachate Polypropylene Tote 1000-L
- pH Adjust Polypropylene Open Head Tank 1000-L
- Sx Stir Polypropylene Open Head Tank Reactor 400-L
- Acid Pallet with Two 200-L Drum for Nitric and Sulfuric Acid Storage
- Raffinate 200-L Drum
- Gear Pump
- Dual Cartridge 20" Carbon Filter to Remove Organic
- Ix Surge Polypropylene Open Head Tank Storage
- 3 Stage Ix Drums 1, 2 & 3 200-L situated on a cart
- 20 µm Particle 20" Filter Cart w/ Diaphragm Pump to Transfer Leachate and Acid
- 20-350 rpm (1/4-hp) Variable Speed Batch Mixers

Safety Features include:

- Spill Containment around the whole plant
- Acid Spill Kit
- Acid Absorbent Mats
- Organic Absorbent Mats
- Eye Wash Station
- Coveralls
- Safety Glasses and Face Shield
- MSDS



PROCEDURES

1. Leachate Composite/Filtration

- Composite leachate feed (~8 g/L Pb) and transfer leachate solution into the 1000-L tote by pumping through particle filter.

2. pH Adjustment to 2.8 with 50-wt% HNO3

- Approximately 800-L of filtered leachate composite will be transferred into the 1000-L STR
- Approximately 15-kg of 50-wt% HNO3 will be prepared for pH adjustment
- Start impeller, take a 100-mL sample and measure the pH of the leachate solution
- Add acid to bring pH to approximately 2.8, record the amount of acid used and take a 100-mL sample of the pH adjusted leachate solution

3. A. Solvent Extraction - Fe Removal by Cyanex 272

- Prepare a approximately 160-L of a 10-vol% Cyanex 272 in Isopar-M diluent in a 400-L STR
- Transfer approximately 160-L of pH 2.8 adjusted leachate solution into the 400-L STR containing the Cyanex 272 and allow the organic/aqueous to mix gently for approximately 20-minutes
- Stop mixer and allow emulsion to completely separate
- Drain the aqueous phase out from the bottom valve into a temporary containment drum with a valve - to be organic adsorption filtered
- Capture interface solution in a 20-L carboy to be phase separated - return organic phase back into the 400-L STR

B. Organic Regeneration

- Transfer approximately 160-L of 0.5 M HNO3 400-L into the STR containing the loaded Cyanex 272
- Allow the organic/aqueous to mix gently for approximately 20-minutes
- Stop mixer and allow emulsion to completely separate
- Drain the aqueous phase out from the bottom valve into a drum
- Transfer 160-L of 200-g/L H2SO4 into the STR containing the partially regenerated Cyanex 272
- Allow the organic/ aqueous to mix gently for approximately 20-minutes
- Stop mixer and allow emulsion to completely separate
- Drain the aqueous phase out from the bottom valve into a drum
- Transfer approximately 160-L of water into the STR containing the Cyanex 272 and allow the organic/aqueous to mix gently for approximately 20-minutes
- Stop mixer and allow emulsion to completely separate
- Drain the aqueous phase out from the bottom valve into a drum
  The Cyanex 272 is now prepared for the next batch solvent extraction

4. Organic Adsorption Filtration –Cyanex 272 Aqueous Raffinate

- Drain the aqueous raffinate into the 200-L Raff drum
- Pump the raffinate solution through the Dual Cartridge 20" Carbon Filter to remove organic
- Store solution in 1000-L surge tank

5. Ion Exchange - Pb Recovery by Purolite C-104-H

- Pre-treat Purolite C-104-H by washing with deionized water
- Submerge Purolite in deionized water for a minimum of 24-hr
- Measure 20-L of Purolite C-104-H resin into 200-L Drum 1, 2 & 3
- Pump aqueous raffinate into Drum 1 and allow Pb to load for approximately 24 hours with over head mixing
- Transfer raffinate into Drum 2 and allow Pb to load for approximately 24 hours with over head mixing
- Transfer raffinate into Drum 3
- Once the resin in Drum 1 is saturated (R/A=0.02 has been reached), take the drum offline. Continue Pb recovery by using Drum 2 & 3
- When all the raffinate is processed, decant drums 1, 2 & 3
- Transfer the 20-L of Pb loaded Purolite C-104-H resin from Drum 1 into the 75-L tank
- Wash loaded Purolite C-104-H in Drum 1, 2 & 3 thoroughly with deionized water

6. Loaded Purolite C-104-H Resin Scrubbing and Washing

- Scrub loaded Purolite C-104-H by using 16-L of 3M CH3COOH for 4 hrs with over head mixing

7. Loaded Purolite C-104-H Resin Stripping with 22-wt% HNO3

- Strip loaded Purolite C-104-H resin with 16-L of 22-wt% HNO3 for 4 hours with over head mixing
- Decant the stripped solution and set aside for Pb(NO3)2 crystallization
- Transfer the stripped resin out of the drum, thoroughly wash the resin with deionized water and store resin in 20-L buckets
- Repeat resin scrubbing, washing and stripping with 20-L loaded resin for Drum
- Thoroughly wash the resin from Drum 3 with deionized water and store in a 20-L bucket as "Partially Loaded Purolite C-104-H Resin – 5 kg Pb(NO3)2 Production"

8. Pb(NO3)2 Crystallization by Evaporating Excess Water

- Evaporate 32-L strip solution, transfer solution into 3, 2-L Pyrex beakers
- Place beakers on hot plates with magnetic stir rods and turn on the heat/ mixer
- Add more strip liquor solution to each beaker as the beaker volume allows until Pb(NO3)2 crystals form
- Evaporation process is conducted in a fume hood

10

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Project No. 4040751

Progress Report No 10

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC.

DURING FEBRUARY 2006

Prepared by

Norman Chow, Kenneth Seto (VizonScitec Inc)
and
Clive Brereton, Alan Keylock (Noram Engineering and Constructors Ltd)

INTRODUCTION

The work carried out in the month of February 2006 focused on purification of the 5 Kg $Pb(NO_3)_2$ crystal sample, batch Pb load rate testing using Purolite C104H and Bayer TP207H ion exchange resin, batch Pb strip testing using loaded Bayer TP207H ion exchange resin and pilot scale Pb load testing using Purolite C104H and Bayer TP207H resin in ion exchange columns.

The 5 Kg Pb(NO3)2 crystal sample prepared from Yava leachate (initiated in the previous month) was purified using three successive water wash and recrystallization cycles to produce a high purity product requested by a potential Yava client. This sample was shipped to the client for product testing.

Batch ion exchange loading rate tests were conducted using Purolite C104-H and Bayer TP207-H ion exchange resin. The intent of this test work was to examine the relative rates in which Pb is loaded onto these resins.

A beaker equilibrium stripping isotherm test was performed on Bayer TP207H ion exchange resin preloaded with Pb. The intent of this test was to examine the maximum concentration of the strip liquor and strip efficiency as a function of R/A ratio.

Pilot scale ion exchange column tests were conducted to produce Pb loading breakthrough curves to examine loading efficiency and kinetics as a function of flow rate and resin type. Tests were conducted in a 3 inch inner diameter column at flow rates of 5 BV/hour and 12 BV/hour using Purolite C104H and Bayer TP207H ion exchange resins.

SUMMARY

1. After 3 cycles of water washing with deionized water, the 5 Kg $Pb(NO_3)_2$ crystals produced from Yava leachate achieved high purity. All metal impurities with the exception of calcium were below an arbitrarily low specification limit requested by Yava's client. The lowest Ca concentration reached was 39 ppm which the client had indicated would be acceptable. The 5 Kg Pb(NO3)2 sample was shipped to the client for product evaluation.

2. Batch Pb loading kinetic tests were performed on Purolite C104-H and Bayer TP207-H ion exchange resins at an R/A ratio of 0.04. After a 24 hour loading time the Bayer TP207 loaded about 80% of the Pb in the aqueous solution, while the Purolite C104-H loaded less than 60%.

3. Batch Pb stripping tests were performed with Bayer TP207H ion exchange resin pre-loaded with Pb. A high Pb strip efficiency (98.2% removal of Pb) was achieved at an R/A ratio of 0.10 resulting in a strip liquor concentration of 13.03 g/L Pb. As the R/A ratio increase, the Pb strip efficiency decreases and strip liquor concentration increases. At an R/A ratio of 1.25, the Pb strip efficiency was 43.4% of Pb and the strip liquor concentration was 45.496 g/L Pb.

4. Pilot scale ion exchange column load tests were conducted using Bayer TP207H at 5 BV/hour and 12 BV/hour. The breakthrough curves show that 12 BV/hour is too high of a flow rate to provide adequate contact time for efficient loading. At 12 BV/hour, Pb would report to the raffinate after a small BV had been treated. At both flow rates, insufficient BV of leachate were treated to reach breakthrough.

5. Pilot scale ion exchange column load test was conducted using Purolite C104H at 5 BV/hour. When comparing the breakthrough curves of Purolite C104H with Bayer TP207H at 5 BV/hour, the Bayer TP207H exhibited much faster loading kinetics. Pb would report to the raffinate much sooner for Purolite C104H than with Bayer TP207H.

RESULTS AND DISCUSSIONS

5 Kg Pb(NO3)2 Production from Yava Leachate

The production of 5 Kg of Pb(NO3)2 from Yava leachate was completed. Three wash cycles with deionized water were required to bring the impurity levels below an arbitrarily low specification limit provided by the potential customer. Ca was still above the specification limit, however the client had indicated that the 39 ppm Ca remaining was low and that the specifications provided were only an arbitrary guideline. ICP analysis of the Pb(NO3)2 crystals at various stages in the production process and the specification limits provided by the client are summarized in Table 1. Complete dissolution of the crystal product followed by recrystallization would have improved purity further, however, this was deemed unnecessary. A photograph of the 5 Kg Pb(NO3)2 crystal product is shown in Figure 1.

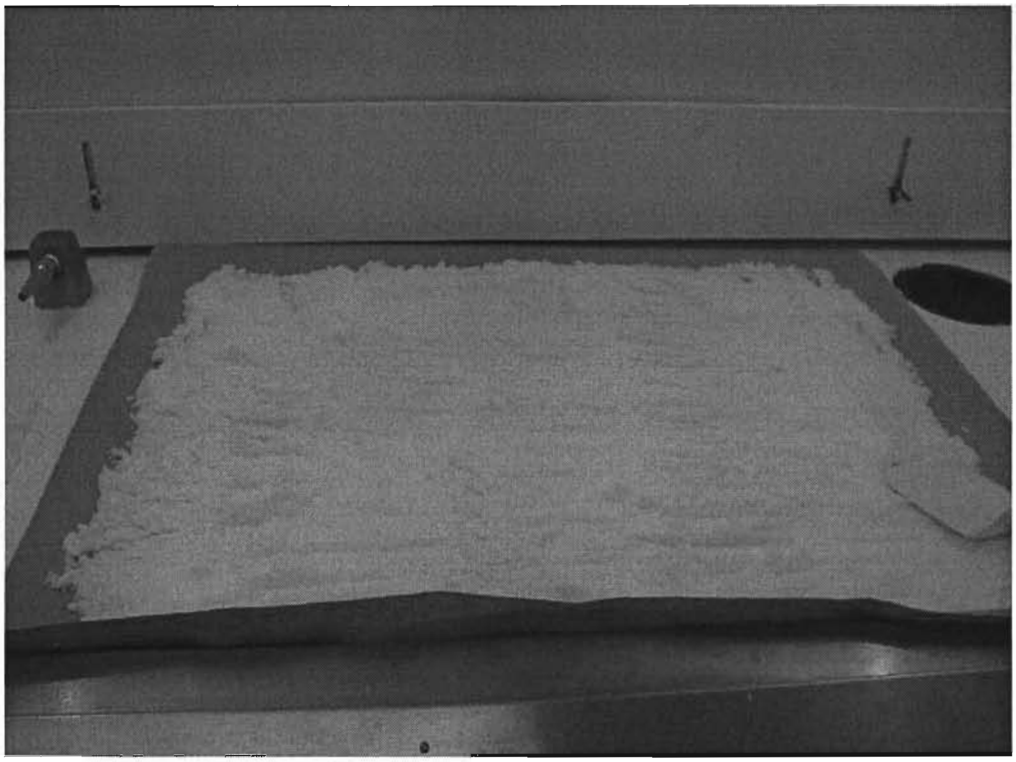


Figure 1: 5-kg $Pb(NO_3)_2$ crystals air drying on craft paper in the fume hood.

Table 1: ICP analysis for $Pb(NO_3)_2$ crystals purified by washing.

Element	Spec. Limit (ppm)	$Pb(NO_3)_2$ (ppm)	Washed $Pb(NO_3)_2$ #1 (ppm)	Washed $Pb(NO_3)_2$ #2 (ppm)	Washed $Pb(NO_3)_2$ #3 (ppm)	Element	Spec. Limit (ppm)	$Pb(NO_3)_2$ (ppm)	Washed $Pb(NO_3)_2$ #1 (ppm)	Washed $Pb(NO_3)_2$ #2 (ppm)	Washed $Pb(NO_3)_2$ #3 (ppm)
Al	50	<100	9	9	<1	**Hg**		<3	<3	<3	<3
Sb	5	<5	<5	<5	<5	**Mo**		<1	<1	<1	<1
As	1	<5	<5	<5	<5	**Ni**	1	<1	<1	<1	<1
Ba		<2	<2	<2	<2	**P**		<100	<100	<100	<100
Bi		<2	<2	<2	<2	**K**		<100	<100	<100	<100
Cd	5	<0.2	<0.2	<0.2	<0.2	**Sc**		<1	<1	<1	<1
Ca	5	132	71	43	39	**Ag**	10	<0.1	<0.1	<0.1	<0.1
Cr		4	2	<1	<1	**Na**		<100	<100	<100	<100
Co		<1		<1	<1	**Sr**		4	5	4	4
Cu	10	1	1	3	2	**Tl**		43	<100	55	<10
Fe	5	<100	19	<1	<1	**Ti**		<100	<100	<100	<100
La		<2	<2	<2	<2	**W**		<5	<5	<5	<5
Zn	5	2	<1	<1	2	**Mn**	1	10	6	1	<1
Mg	<100	**<100**	**<100**	<100	<100	**Sn**	1	<10	<10	<10	<10

Bayer TP-207-H and Purolite C104-H Batch Loading Kinetic Test at a R/A Ratio of 0.04

Equilibrium loading rate tests were performed on Bayer TP207-H and Purolite C104-H ion exchange resins. In a 20-litre HDPE bucket, 18-L of Fe removed Yava Leachate (removed with Cyanex 272 at pH 2.8) and 720-mL fresh resin were mixed gently with an over-head mixer for 24 hours while incremental aliquot samples were taken for analyses. An R/A ratio of 0.04 was selected because prior test have shown that this R/A ratio provides a good compromise between selectivity and loading capacity. The loading profiles for Pb, Fe, Mn and Al as a function of time are shown in Figures 2 and 3 for Bayer TP207-H and Purolite C104-H respectively.

Due to an over-head mixer motor failure, the Purolite C104-H equilibrium loading rate experiment was terminated after 210 minutes. Nevertheless, it is still apparent from these plots that the Pb-loading kinetics for Purolite C104-H is much lower than for Bayer TP207-H.

At the end of the 24 hour loading time with Bayer TP207-H, almost 80% of the Pb in solution was loaded onto the resin (shown in Figure 2). Along with the Pb, small quantities of Fe, Mn and Al were also co-loaded. However, since the concentrations of these metal ions are low in the Fe removed Yava leachate, the total metal impurities loaded onto the resin are low.

Figure 2: Bayer TP207-H equilibrium load rate test.

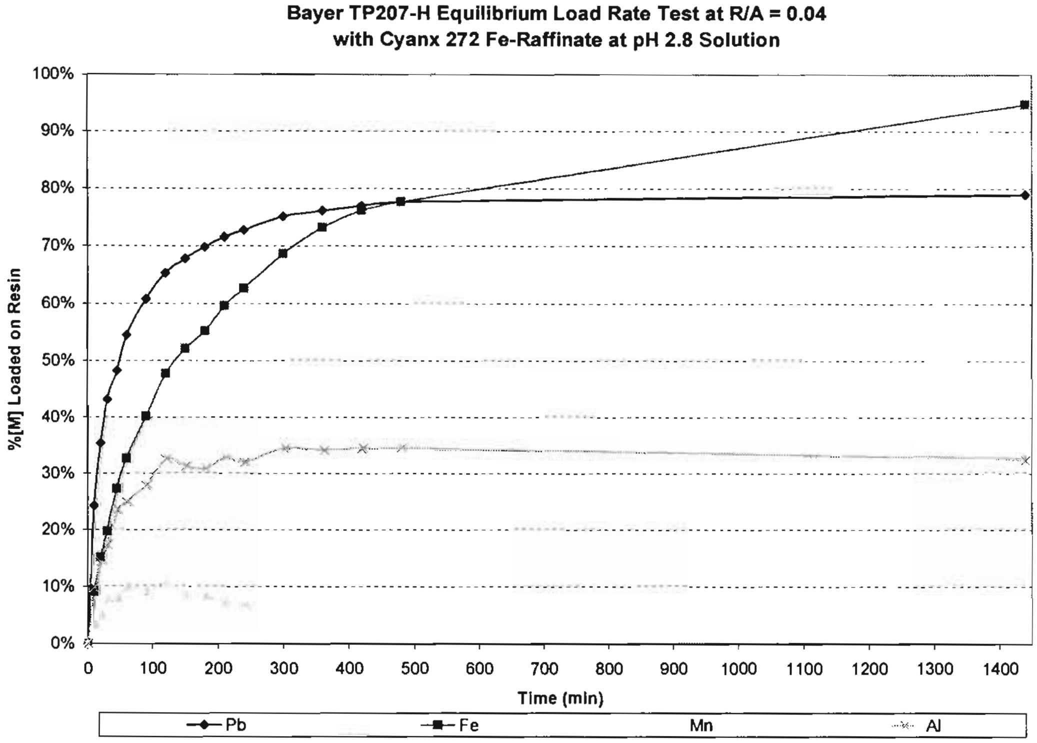
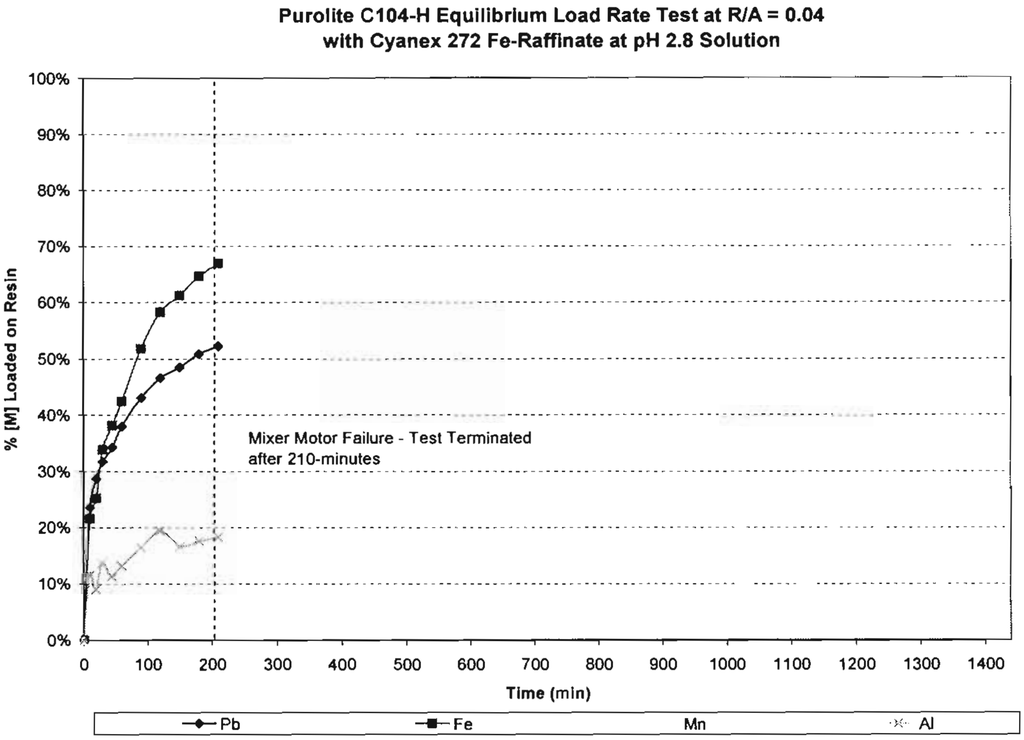


Figure 3: Purolite C104-H equilibrium load rate test.



When comparing the loading rate curves for Bayer TP207-H and Purolite C104-H, the Pb-loading kinetics were faster for Bayer TP207-H, as shown in Figure 4. At the 210 minute time interval, the Bayer TP207-H achieved approximately 72% Pb loading, while Purolite C104-H only reached about 53% Pb loading. It was also observed that Purolite C104-H loads Fe faster than Pb, even when the Pb concentration is substantially greater than the Fe concentration. As such, it is important to minimize the concentration of Fe in the feed solution with the Cyanex 272 pre-treatment.

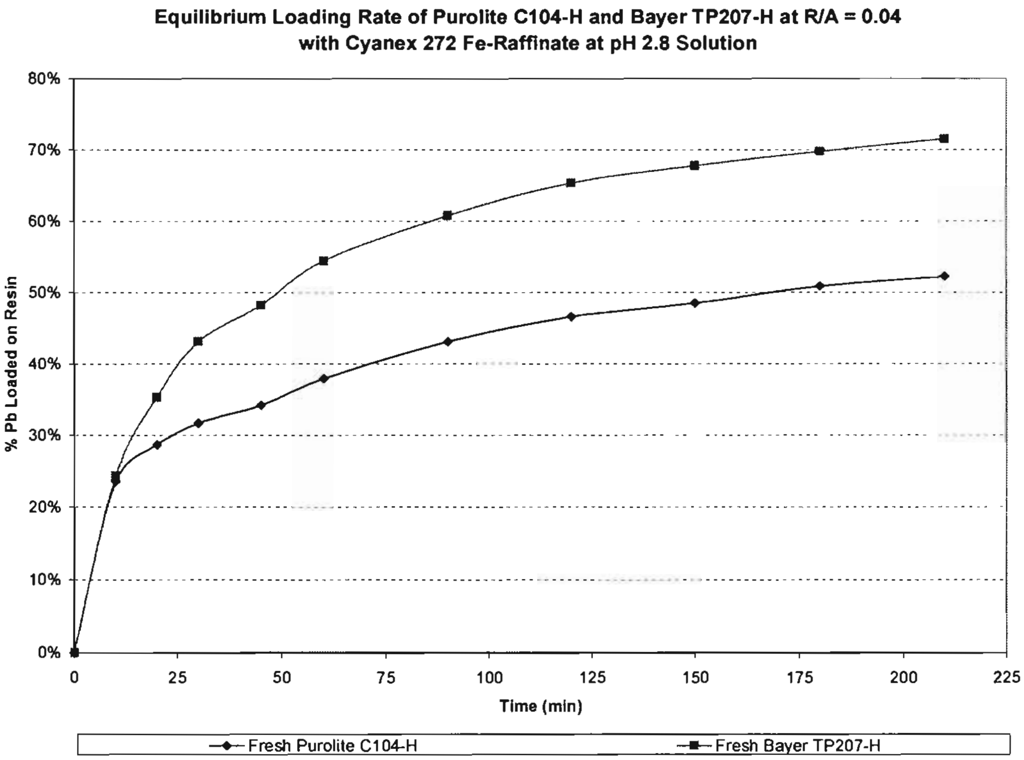


Figure 4: Comparison between the percentage Pb loaded onto Purolite C104-H and Bayer TP207-H for the first 210 minutes.

Bayer TP-207-H Stripping Equilibrium Isotherm Test.

The Pb loaded Bayer TP207-H resin produced earlier was used for the equilibrium stripping isotherm test. The Pb loaded Bayer TP207-H resin was thoroughly washed and immersed in deionized water. A 12-wt% HNO_3 strip solution was prepared from analytical grade HNO_3 (69-wt%) and deionized water. The loaded Bayer TP207-H resin volumes and 12-wt% HNO_3 strip solution volumes were measured into beakers in accordance to Table 2.

Table 2: Bayer TP207-H stripping equilibrium isotherm test conditions with 12-wt% HNO_3.

	Beaker #1	**Beaker #2**	**Beaker #3**	**Beaker #4**	**Beaker #5**	**Beaker #6**
Resin	25-mL	50-mL	80-mL	120-mL	125-mL	150-mL
Aqueous	250-mL	200-mL	160-mL	160-mL	125-mL	120-mL
R/A Ratio	**0.10**	**0.25**	**0.50**	**0.75**	**1.00**	**1.25**
Time	255-min	255-min	255-min	255-min	255-min	255-min

The six beakers were concurrently mixed gently using overhead gangue mixers for approximately 255 minutes. Solution and resin samples were taken at the end of the test and submitted for analyses. The result of the equilibrium stripping isotherm tests are presented in Table 3 and equilibrium isotherm curve is shown in Figure 5.

At a low R/A ratio of 0.10, most of the Pb (98.2%) was removed from the ion exchange resin. However, the Pb concentration of the strip liquor was relatively low at 13.03 g/L. A low strip liquor Pb concentration would require more energy to crystallize Pb(NO3)2, whereas a low strip efficiency would require more resin to recover Pb. An R/A ratio of 0.50 results in a Pb strip efficiency of 89.9% and strip liquor Pb concentration of 52.932 g/L, which appears to be a good compromise. Ultimately, this will be verified during column stripping tests at selected flow rates.

Table 3: ICP analysis for Bayer TP-207H stripping test results.

				Beaker #1	Beaker #2	Beaker #3	Beaker #4	Beaker #5	Beaker #6
	R/A Ratio		Feed	0.10	0.25	0.50	0.75	1.00	1.25
Pb	**Resin**	%	25.34	0.45	1.26	2.56	5.46	10.98	14.35
	Aqueous	mg/L		13030	31412	52932	61383	55109	45496
	Stripped $_{resin}$	%		**98.2%**	**95.0%**	**89.9%**	**78.5%**	**56.7%**	**43.4%**
Mn	**Resin**	g/t	287	<1	2.0	2.0	2.0	4.0	3.0
	Aqueous	mg/L		20.4	45.4	80.7	108.6	128.5	150.0
	Stripped $_{resin}$	%		**99.7%**	**99.3%**	**99.3%**	**99.3%**	**98.6%**	**99.0%**
Fe	**Resin**	g/t	1062	<100	164	566	1245	1208	1134
	Aqueous	mg/L		71.88	156.74	188.74	61.78	15.43	7.59
	Stripped $_{resin}$	%		**90.6%**	**84.6%**	**46.7%**	**0%**	**0%**	**0%**
Al	**Resin**	g/t	284	<100	<100	<100	<100	<100	<100
	Aqueous	mg/L		17.0	38.9	66.3	91.7	106.5	120.6
	Stripped $_{resin}$	%		**64.8%**	**64.8%**	**64.8%**	**64.8%**	**64.8%**	**64.8%**
Cu	**Resin**	g/t	36	7	13	34	42	42	41
	Aqueous	mg/L		1.62	3.58	2.67	0.56	0.11	<0.01
	Stripped $_{resin}$	%		**80.6%**	**63.9%**	**5.6%**	**0%**	**0%**	**0%**
Mg	**Resin**	g/t	130	<100	<100	<100	<100	<100	<100
	Aqueous	mg/L		6.1	12.7	23.4	31.1	36.1	43.6
	Stripped $_{resin}$	%		**23.1%**	**23.1%**	**23.1%**	**23.1%**	**23.1%**	**23.1%**
Zn	**Resin**	g/t	15	4	5	5	5	11	8
	Aqueous	mg/L		0.70	2.06	2.70	3.59	4.46	4.53
	Stripped $_{resin}$	%		**73.3%**	**66.7%**	**66.7%**	**66.7%**	**26.7%**	**46.7%**
Ca	**Resin**	g/t	833	183	188	200	208	228	200
	Aqueous	mg/L		51.8	111.3	202.3	274.9	324.2	382.4
	Stripped $_{resin}$	%		**78.0%**	**77.4%**	**76.0%**	**75.0%**	**72.6%**	**76.0%**
Cr	**Resin**	g/t	72	49	70	82	85	81	83
	Aqueous	mg/L		2.08	2.83	1.82	0.93	0.63	0.61
	Stripped $_{resin}$	%		**31.9%**	**2.8%**	**0%**	**0%**	**0%**	**0%**
Ni	**Resin**	g/t	88	18	21	43	62	84	85
	Aqueous	mg/L		5.07	10.72	14.7	13.34	8.63	7.45
	Stripped $_{resin}$	%		**79.5%**	**76.1%**	**51.1%**	**29.5%**	**4.5%**	**3.4%**
Co	**Resin**	g/t	25	9	7	9	7	11	11
	Aqueous	mg/L		1.04	2.31	4.04	5.44	6.08	6.18
	Stripped $_{resin}$	%		**64.0%**	**72.0%**	**64.0%**	**72.0%**	**56.0%**	**56.0%**

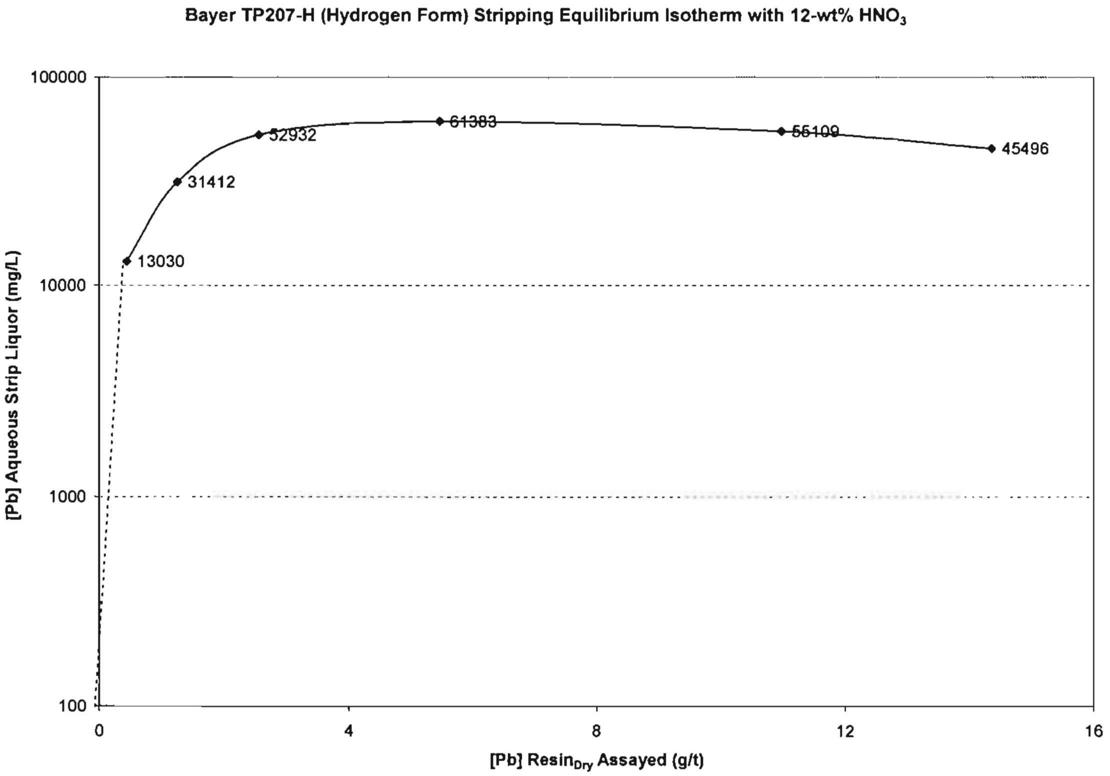


Figure 5: Bayer TP207-H equilibrium stripping isotherm with 12-wt% HNO_3.

A single stage strip configuration is desirable as long as a significant amount of metal ions are removed in a single pass with the 12-wt% HNO_3 strip solution. Should some metal ions remain tightly held by the ion exchange resin after stripping, resin regeneration tests, perhaps by using a high concentration acid should be conducted over several cycles to examine performance and robustness. Load - strip cycle testing should also be conducted with the resin of interest to determine if metal ions accumulate onto the resin after each cycle.

Pilot Scale 3 Inch Column Breakthrough Tests at 5 BV/hr and 12.5-BV/hr with Bayer TP 207H

A 3"-inside diameter clear PVC ion exchange column was constructed as shown in Figure 6. The column was filled with 1440-mL of pre-measured Bayer TP207-H ion exchange resin and back-flush to obtain a packed resin bed prior to column testing.

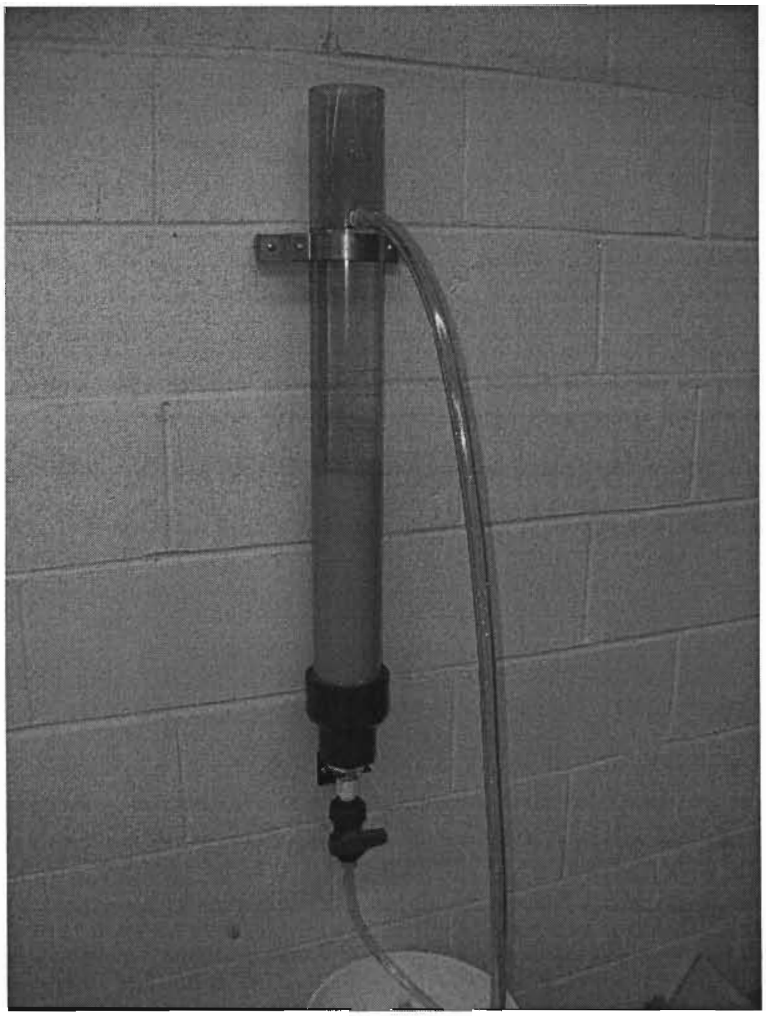


Figure 6: A single 3"-inside diameter ion exchange column for bench testing.

The feed solution was placed on a digital weight scale and the pump speed adjusted to provide the desired flow rate. The flow rate through the column was calculated using the weight loss of the feed solution over time. The ball valve at the bottom of the column was adjusted to maintain a column level of approximately 6" above the resin bed. Incremental aliquot samples were collected every 10-15 minutes for analyses.

Two different column flow rates (5-BV/hr and 12-BV/hr) were tested to gauge resin loading kinetics. The objective was to find loading conditions that would allow enough contact time between the resin and the aqueous solution for the resin to load as much lead as possible while treating as much feed solution as possible. The Pb analyses of the incremental raffinate samples are plotted against the volume of aqueous feed solution which has passed through the column in Figure 7.

As expected, lower flow rates are beneficial for column loading kinetics since the increase in contact time will typically improve Pb-loading. Figure 7 shows that at the same bed volume (BV), the Pb concentration in the raffinate solution is higher when operating at 12-BV/hr than for 5 BV/hr. A higher Pb concentration in the raffinate indicates that less Pb is loaded onto the resin.

A flow rate of 5-BV/hr and 12-BV/hr resulted in an initial column breakout of Pb (the volume of feed solution treated at which Pb is detected in the raffinate) at approximately 13-BV and 5-BV, respectively. As such, Pb loading kinetics for Bayer TP 207H are faster a flow rate of 5 BV/hr. However, given that we only have a portion of the loading breakthrough curve completed, a prediction of the breakthrough can only be obtained by extrapolation. Lower flow rates will be tested with complete breakthrough curves required to verify loading characteristics to breakthrough.

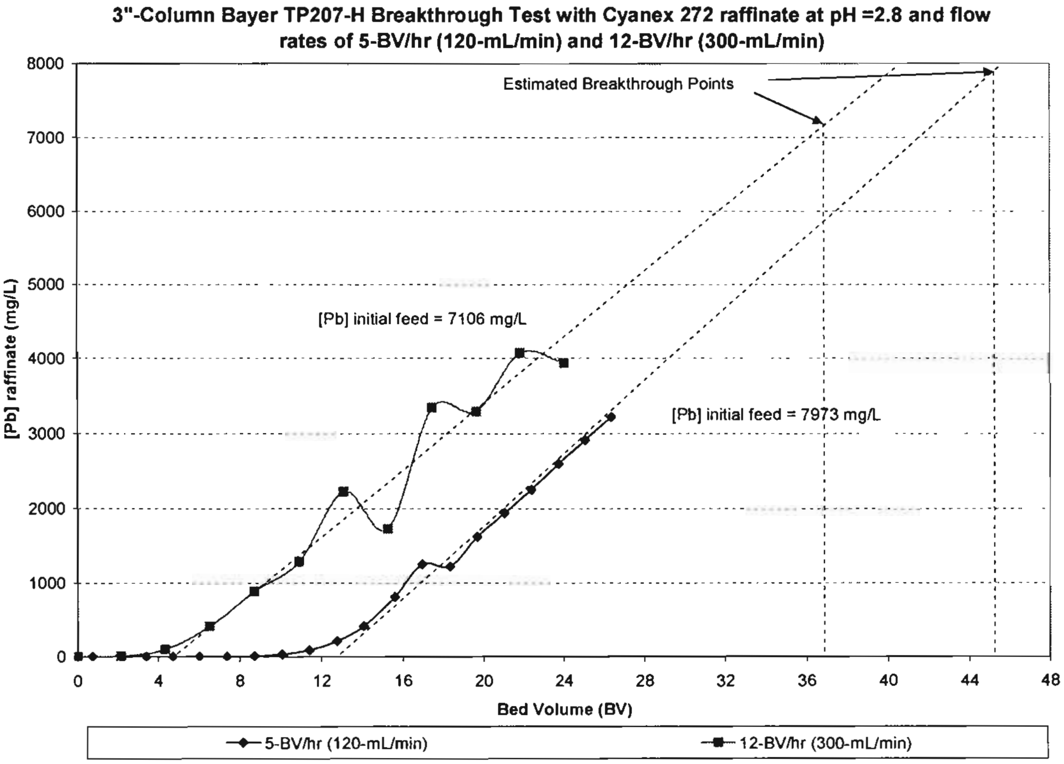

3"-Column Bayer TP207-H Breakthrough Test with Cyanex 272 raffinate at pH =2.8 and flow rates of 5-BV/hr (120-mL/min) and 12-BV/hr (300-mL/min)

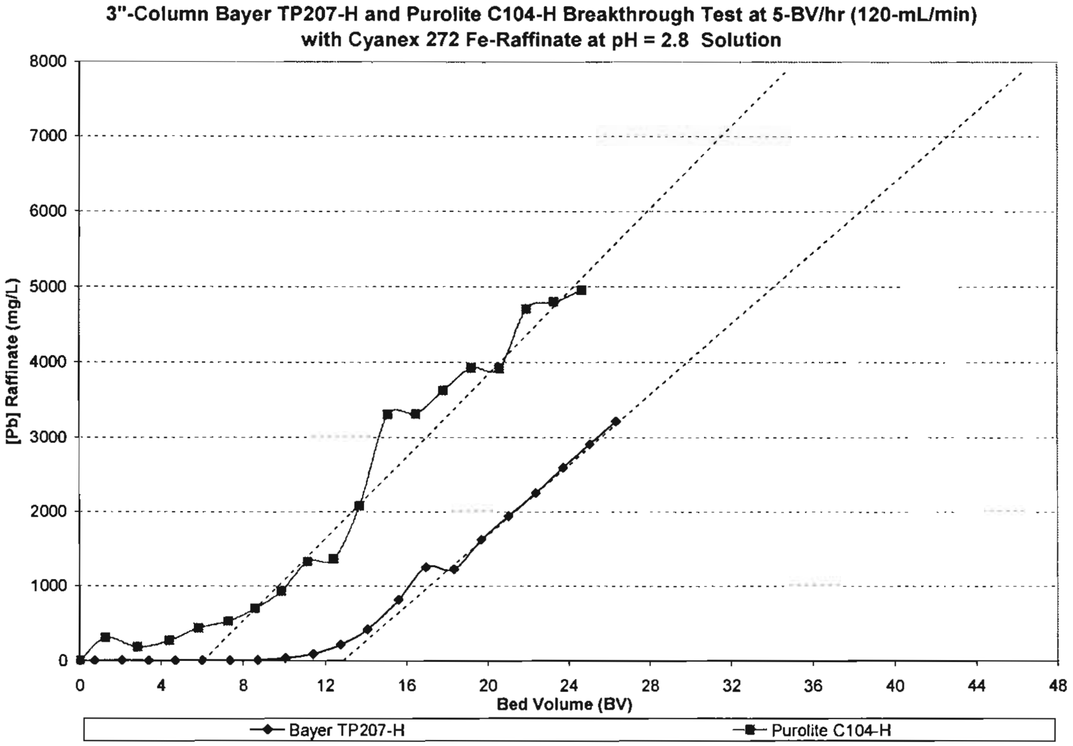
Figure 7: Pilot Scale 3 Inch Column breakthrough Curves for Bayer TP207H at 5-BV/hr and 12-BV/hr

Pilot Scale 3 Inch Column Breakthrough Tests at 5 BV/hr with Purolite C104H and Comparison with Bayer TP207H

The same experiment performed previously with Bayer TP207-H at a column flow rate of 5-BV/hr was repeated for Purolite C104-H resin. At the same column flow rate and volume of resin, less Pb was loaded onto the Purolite C104H resin (as indicated by the higher Pb concentration in the raffinate). This indicates that Pb loading kinetics are faster with Bayer TP207-H as shown in Figure 8.

Although Bayer TP207H has faster loading kinetics for Pb compared to Purolite C104-H, the Bayer TP207H can only be stripped with a maximum HNO3 concentration of 12-wt% HNO_3 (to minimize resin degradation) compared to 22 wt% HNO3 for Purolite C104H. Stripping with a high concentration HNO3 would result in a high concentration of Pb in the strip liquor.

Figure 8: Comparison between Purolite C104-H and Bayer TP207-H breakthrough curves.



Extrapolation was conducted to estimate the shape of the complete breakthrough curves for Bayer TP207H and Purolite C104H at 5 BV/hour. The large transition zone (the BV range where the Pb is detected in the raffinate and the point where the concentration of Pb in the raffinate is equal to the concentration of Pb in the feed solution) typically indicates that a multi-stage loading scheme is required.

Single and multi-stage column loading tests will be conducted at lower flow rates to complete breakthrough in the following month.

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Project No. 4040751

Progress Report No 11

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC.

DURING MARCH 2006

Prepared by

Norman Chow, Kenneth Seto (VizonScitec Inc)
and
Clive Brereton, Alan Keylock (Noram Engineering and Constructors Ltd)

INTRODUCTION

The work carried out in the month of March 2006 focused on the assembly of two new pilot scale ion exchange columns and conducting pilot scale ion exchange column tests to examine Pb separation under different test conditions.

The two ion exchange columns were manufactured from clear PVC piping and were each 5 feet height and 1.5 inches inner diameter. The columns were designed to facilitate flexible test configurations either by testing a single column individually or two columns in series. The first column in a two column series configuration is often referred to as the "lead column" and the second column is referred to as the "tail column". The multi-column configuration can improve the recovery of Pb particularly when resin loading kinetics is slow. These new 1.5 inch inner diameter columns can operate with less volume of resin compared to the 3 inch inner diameter columns tested in the previous month. This enables completed breakthrough to be achieved with lower volumes of feed solution. It is desirable to conserve the feed solution for future testing.

Pb separation tests were conducted using Purolite C104H ion exchange resin. Pb loading and stripping tests were conducted on a pilot scale using the new ion exchange columns to produce loading breakthrough curves and stripping profiles. The loading breakthrough curves and stripping profiles will be used to assess the performance of the ion exchange resin while operated under various conditions and to compare relative performance of the Purolite C104H resin to the Bayer TP207H resin (to be reported in the following month).

SUMMARY

1. Column stripping Pb loaded Purolite C104H ion exchange resin with 22 wt% HNO_3 resulted in a higher peak incremental strip liquor Pb concentration at 1.5 BV/hour than at 3.0 BV/hour (351 g/L compared to 270 g/L)
2. Column stripping Pb loaded Purolite C104H ion exchnage resin with 22 wt% HNO_3 resulted in a higher final accumulated strip liquor Pb concentration at 1.5 BV/hour than at 3.0 BV/hour (175 g/L compared to 142 g/L)

3. Pb elution approaches completion sooner at 1.5 BV/hour than 3.0 BV/hour (about 2.25 BV compared to about 3.1 BV). As such, less HNO3 eluent is consumed at 1.5 BV/hour.
4. By comparing the results of the Pb column loading test with the bench scale Pb equilibrium loading test using Purolite 104H ion exchange resin, the loading kinetics of Purolite C104H resin was too slow to efficiently load Pb at 5 BV/hour.
5. The Pb loading kinetics of the lead column in a two column carousel ion exchange process is much slower in the 2nd cycle load than in the 1st cycle load. Loading kinetics of the 2nd cycle is slower than the 1st cycle because Pb displaces other co-extracted metal ions when operated closer to the resin capacity.

RESULTS AND DISCUSSION

Assembly of Ion Exchange Columns

Two pilot scale ion exchange columns were assembled for conducting Pb separation tests with Yava leachate. Each column was 5 feet height and 1.5 inches inner diameter. A resin support screen was manufactured with a V-wire stainless steel mesh, cut to 1.5 inch diameter with a high pressure water jet and fitted at the bottom of each column. Flow control was achieved by connecting a 0.5 inch PVC ball valve at the bottom of each column, connected through a 1.5 inch to 0.5 inch PVC reducer. A hole was drilled near the top of each column and tapped to fit a 0.5 inch NPT to 3/8 inch barbed fitting for backflush overflow. The two columns were leveled and anchored to the side of a laboratory bench for easy access. A photo of the ion exchange columns are shown in Figure 1.

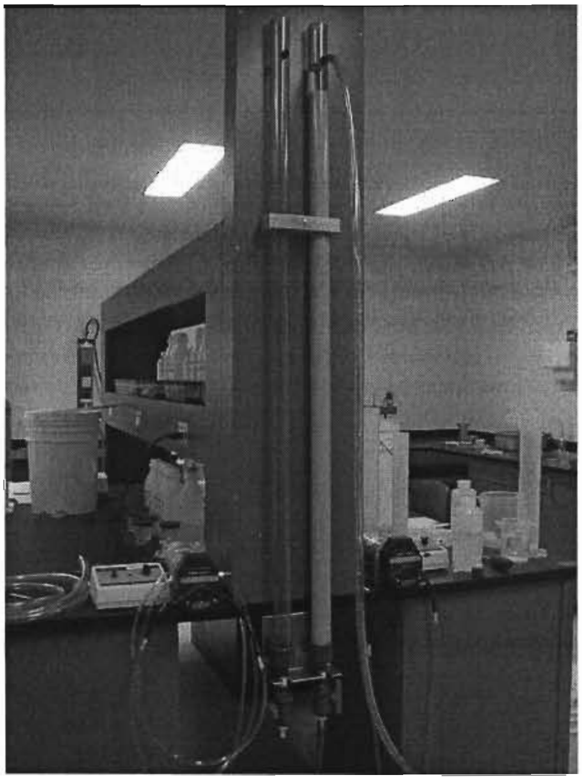

Figure 1. Two Pilot Scale Ion Exchange Columns.

Pb Separation - Ion Exchange Column Studies

Single Column Pb Stripping Tests Using Purolite C104H Ion Exchange Resin.

Pb stripping tests were conducted on loaded Purolite C104H ion exchange resin using a single pilot scale column. Tests were conducted by pre-loading Purolite C104H ion exchange resin with Pb from Yava leachate. Pb loading was conducted in a batch manner similar to the procedure described in Yava Pilot Plant Report No. 6 dated October 2005 (the concentration of Pb in the feed leachate solution was 8.6 g/L). Pb loading was conducted at an R/A ratio of 0.04 which provides a good compromise between Pb selectivity and Pb recovery (as indicated in Yava Pilot Plant Report No. 6). The loaded resin was then water rinsed with deionized water to remove excess leachate and placed inside an ion exchange column for testing. The resin volume (or bed volume) inside the column was 1000 mL for each test. The concentration of Pb loaded onto the Purolite C104H resin was 165 g/L_{resin}.

Pb stripping was conducted by pumping 22 wt% HNO3 to the top of the column and then allowing the HNO3 to flow through the resin bed and drain out the bottom of the column into a holding tank. Two flow rates (1.5 BV/hour and 3.0 BV/hour) were tested with the flow rate adjusted with the ball valve the bottom of the column. Incremental samples of strip liquor were periodically collected as the solution exited the column (prior to entering the holding tank). These samples were retained for Pb analysis by ICP.

Test conditions for the single column stripping tests using the Pb loaded Purolite C104H ion exchange resin are summarized in Table 1 (for greater clarity, flow rate are provided in BV/hr, mL/min and m/hr).

Table 1. Test Conditions for Pb Column Stripping Tests with Purolite C104H.

Resin			**Stripping solution**	**Flow rate**		
Type	**Resin Volume**	**Resin Pb-Loading**		(BV/hr)	(mL/min)	(m/hr)
Purolite C104-H	1000 mL	165-g/L-resin	22-wt% HNO_3	1.5	25	1.25
				3.0	50	2.49

The results of the Pb column stripping tests conducted on the Purolite C104H ion exchange resin are summarized on the stripping profile (graph of concentration of Pb in the strip liquor versus BV) provided in Figure 1. The Accumulated Pb Concentrations shown in Figure 1 (for 1.5 and 3.0 BV/hour) were calculated by numerical integration of the respective incremental Pb concentration curves.

By comparing the stripping profiles at 1.5 BV/hour and 3.0 BV/hour the following are observed:

- Stripping at 1.5 BV/hour results in a higher peak incremental Pb concentration than at 3.0 BV/hour (351 g/L compared to 270 g/L)
- Stripping at 1.5 BV/hour results in a higher peak accumulated Pb concentration than at 3.0 BV/hour (175 g/L compared to 142 g/L)
- Pb elution approaches completion sooner at 1.5 BV/hour than 3.0 BV/hour (about 2.25 BV compared to about 3.1 BV). As such, less HNO3 eluent is consumed at 1.5 BV/hour.

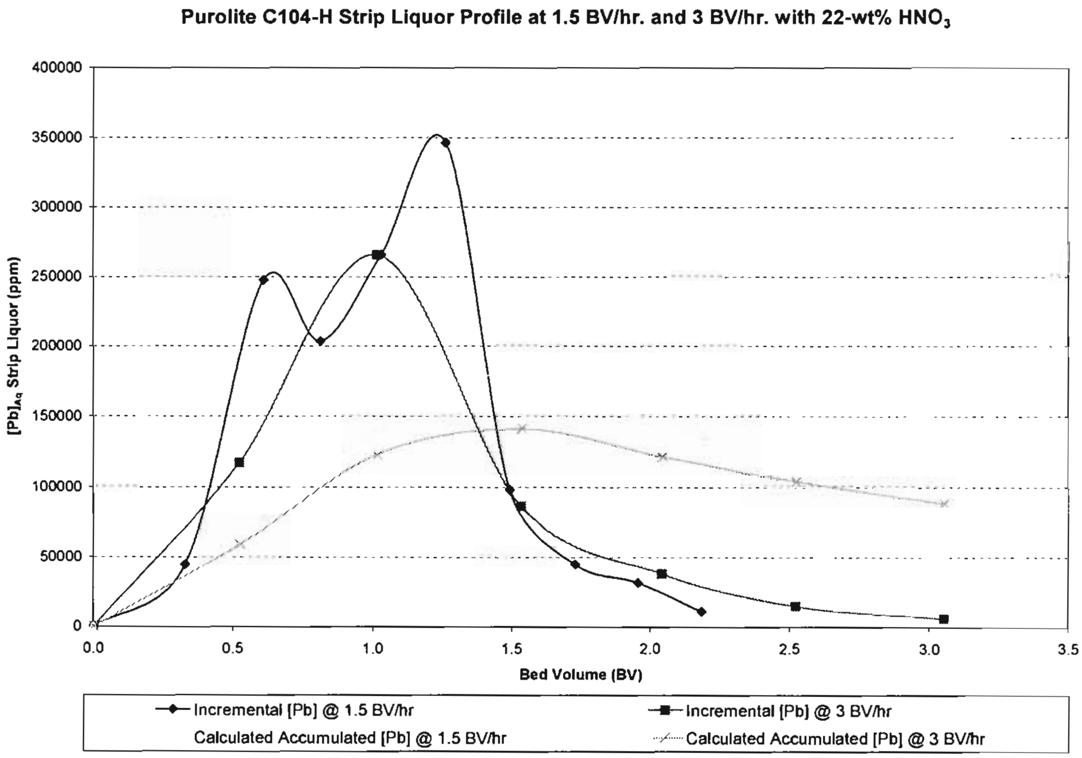

Figure 1. Stripping profile for Purolite C104-H at 1.5-BV/hr and 3.0–BV/hr

Two Stage Column Pb Breakthrough Tests Using Fresh Purolite C104H Ion Exchange Resin

Pb loading breakthrough tests were conducted by connecting two 1.5 inch inner diameter ion exchange columns in series. Each column was filled with 1000-mL of fresh Purolite C104-H resin converted from sodium form to hydrogen form with 22-wt% HNO_3 at an R/A ratio = 1.25. Each column was then backflushed with deionized water to obtain a well distributed resin bed prior to testing. The backflush stratification also assists in the removal of damaged resin, excessively fine resin particles and trapped air in the column. The excess water was drained through the bottom valve of each column, leaving a few inches of water above the resin bed.

The feed solution was placed on a weight scale so that flow rate could be calculated with the weight loss and specific gravity of the solution. The feed pump was started and adjusted to the desired flow rate of approximately 83-mL/min (equivalent to 5 BV/hour). The discharge valve of the tail column was adjusted to maintain a level of approximately 4" above each resin bed. A simplified flow diagram of this two stage column loading test with sampling locations identified is shown in Figure 1. Incremental aliquot samples were taken to obtain a loading profile for each column, shown in Figure 2. The aqueous feed solution consisted of Fe removed Yava leachate (with Fe removed by solvent extraction with Cyanex 272). The concentration of Pb in the feed solution (determined by ICP analysis) was 7.314 g/L Pb.

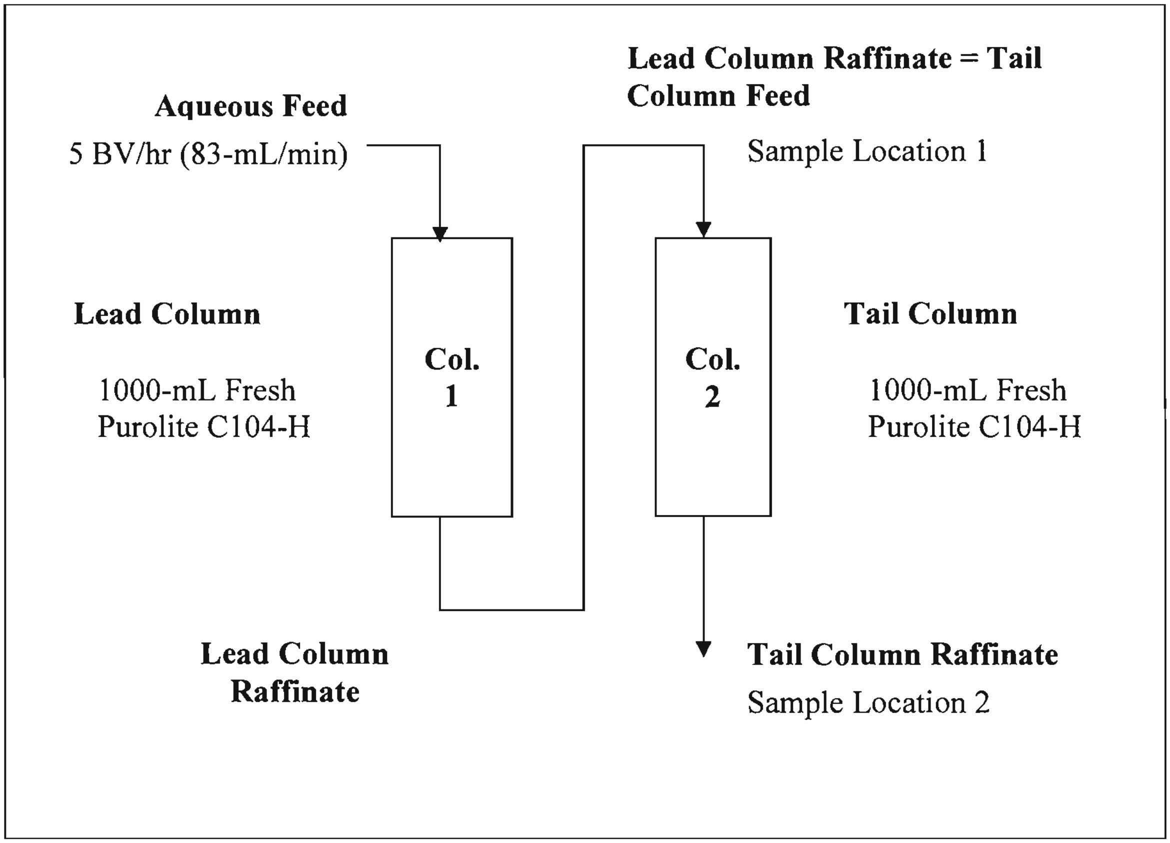

Figure 1: Flow Diagram for Two Stage Loading Test using Fresh Purolite C104-H Ion Exchange Resin – Cycle 1

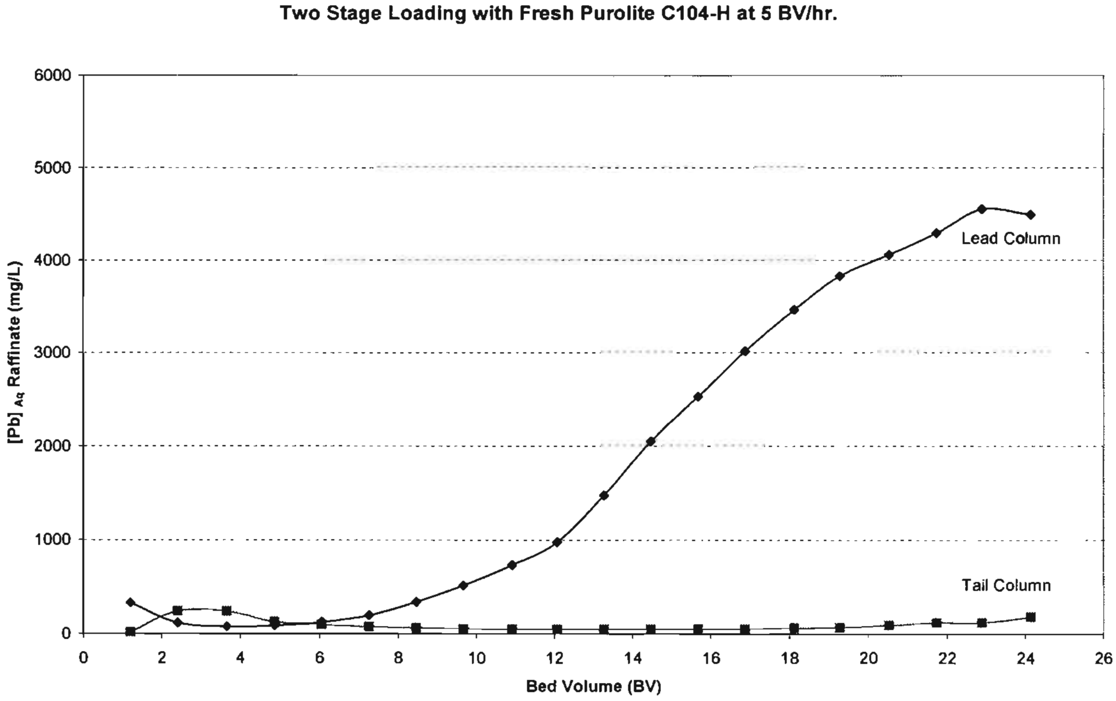

Figure 2: Purolite C104-H two stage loading at 5-BV/hr – Cycle 1

Figure 2 shows that the Lead Column raffinate concentration reaches 4.490 g/L Pb (or 2.824 g/L less than the feed concentration) after 24 BV of Yava leachate was treated with Purolite C104H ion exchange resin at 5 BV/hr. The R/A ratio for this column test (calculated as 1 BV resin divided by 24 BV leachate) was approximately 0.04. After the first load cycle, the concentration of Pb in the loaded ion exchange resin was calculated to be 135 g/L_{resin} in the lead column and 40 g/L_{resin} in the tail column.

In comparison, prior bench scale test work conducted to determine the equilibrium loading for Purolite C104H ion exchange resin at an R/A ratio of 0.04, Yava leachate feed concentration of 8.64 g/L Pb and 24 hour treatment time resulted in a raffinate with 2.880 g/L Pb (or 5.760 g/L less than the feed concentration). The low reduction in Pb concentration in solution after treatment in the column in comparison with the bench scale equilibrium test suggest that the 5 BV/hr is too high of a flow rate for efficient Pb loading with Purolite C104H.

The low concentration of Pb in the tail column raffinate indicates that the tail column is effective in capturing Pb that is missed in the first column. Note that this is only relevant for a single cycle test. Multi-column ion exchange systems typically operate in a carousel type process where the loaded lead column is stripped to remove loaded metals and then moved to become the tail column and the previously partially loaded tail column moves towards becoming the lead column. Depending on loading kinetics, solution flow rate, BV treated and number of stages it is possible that metal ions may accumulate in the ion exchange system resulting in a different loading curve for each load/strip cycle.

Two Stage Column Pb Breakthrough Tests Using Purolite C104H Ion Exchange Resin – Second Cycle in a Carousel Type Process

The two stage Pb load test in the previous section was repeated except by moving the previously partially loaded tail column to become the lead column and placing fresh Purolite C104H resin in the tail column (to simulate stripped resin). This test was conducted to illustrate the possible change in breakthrough curve for the second load cycle in a carousel type ion exchange process. As in the first cycle test, each column was then backflushed with deionized water to obtain a well distributed resin bed prior to testing. The excess water was drained through the bottom valve of each column, leaving a few inches of water above the resin bed.

Load testing was conducted with 1000 mL of resin in each column, a solution flow rate of 5 BV/hr, 24 BV of solution treated and 7.314 g/L Pb in the feed solution. Incremental aliquot samples were taken of the lead column and tail column raffinates to obtain loading profiles. A flow diagram of the two stage, 2nd cycle test is shown in Figure 3.

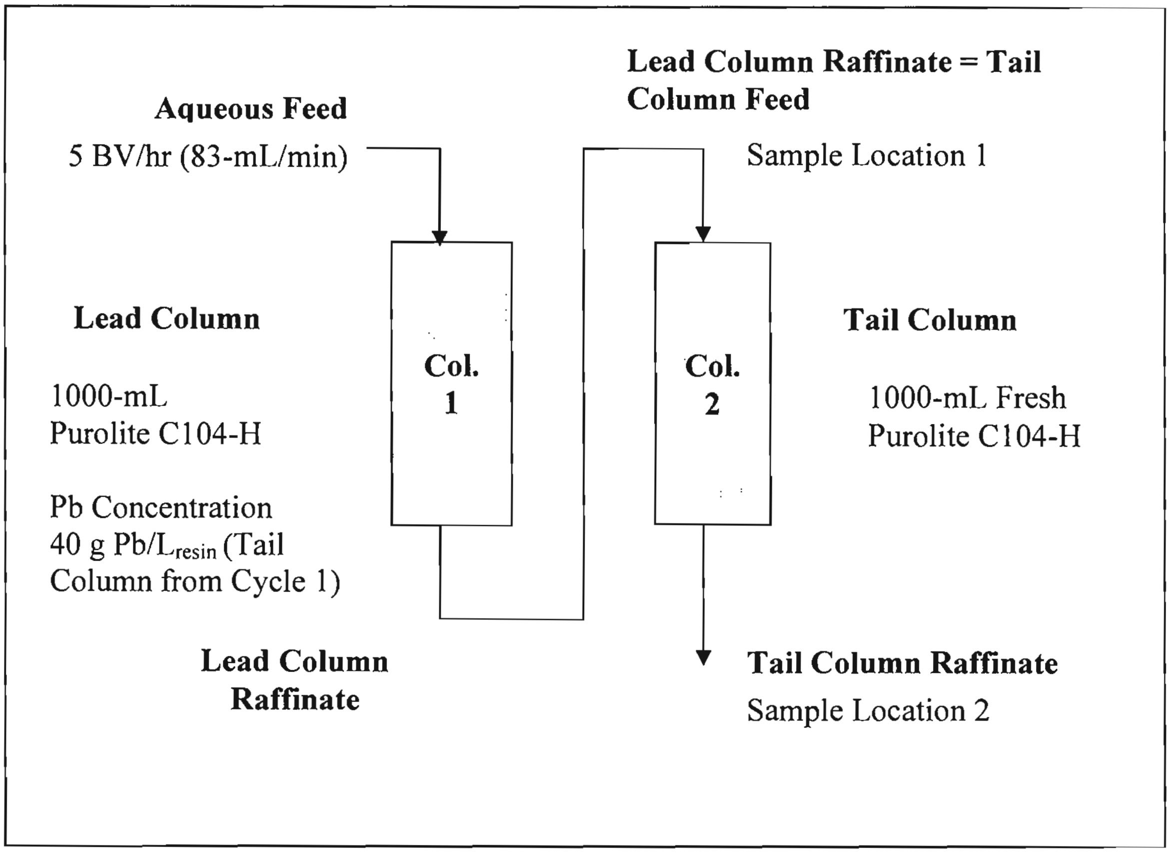

Figure 3: Flow Diagram for Two Stage Loading Test using Fresh Purolite C104-H Ion Exchange Resin – Cycle 2

The loading profile for each column in the 2nd cycle loading test is shown Figure 4 (cycle 1 profile provided for comparison). There is a significant difference in lead column loading profiles between cycle 1 and cycle 2. Cycle 1 reaches a raffinate concentration of 1000 mg/L in approximately 12 BV whereas Cycle 2 reaches a raffinate concentration of 1000 mg/L in approximately 7 cycles (where 1000 mg/L Pb raffinate concentration was selected arbitrarily to illustrate the comparison). The lower BV treated to achieve the same Pb raffinate concentration indicates that loading kinetics is much slower in Cycle 2 than in Cycle 1. This is probably due to the slower process in which Pb displaces other co-extracted metal ions when operating closer to resin capacity.

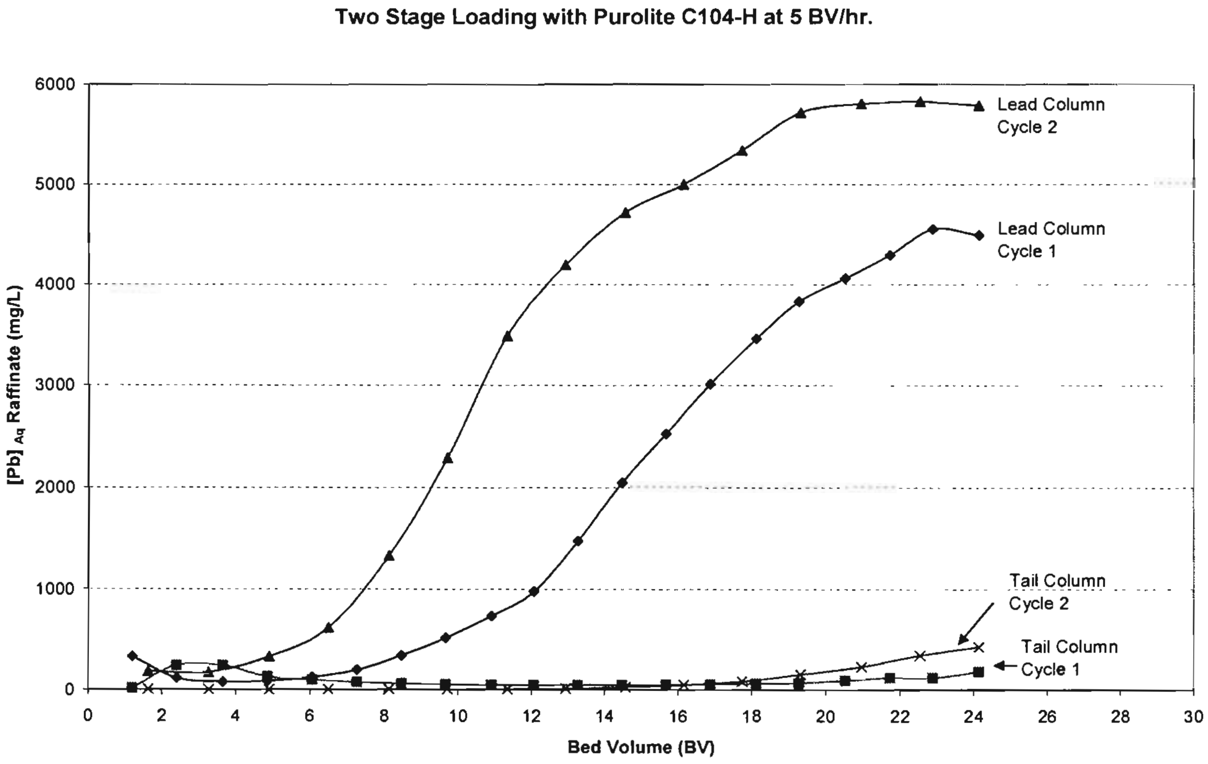

Figure 4: **Purolite C104-H two stage loading at 5-BV/hr – Cycle 1 and Cycle 2**

After the 2nd load cycle, the concentration of Pb in the loaded ion exchange resin was calculated to be 147 g/L_{resin} in the lead column and 75 g/L_{resin} in the tail column. The higher concentration of Pb in the tail column in Cycle 2 compared to Cycle 1 suggests that the loading profile will change for Cycle 3.

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Project No. 4040751

Progress Report No 12

PILOT PLANT INVESTIGATIONS FOR YAVA TECHNOLOGIES INC.

DURING APRIL 2006

Prepared by

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and
Clive Brereton, Michael Gattrell (Noram Engineering and Constructors Ltd)

INTRODUCTION

The work carried out in the month of April 2006 focused on conducting pilot scale ion exchange column tests to examine Pb separation using Bayer TP 207H ion exchange resin.

Pb separation tests were conducted using Bayer TP 207H ion exchange resin. Pb loading and stripping tests were conducted on a pilot scale using the 1.5 inch inner diameter ion exchange columns to produce loading breakthrough curves and stripping profiles. The loading breakthrough curves and stripping profiles will be used to assess the performance of the ion exchange resin while operated under various conditions and to compare relative performance of the Bayer TP207H resin to the Purolite C104H resin tested in the previous month.

This test work was conducted with a new Bayer TP207H ion exchange resin prepared with a new manufacturing technique that produces a more uniform bead size. This resin is sometimes referred to by the manufacturer as a TP207H Monoplus resin (although the official name is still TP207H) and replaces the old TP207H resin. Performance of the new TP207 resin is better than the old TP207H resin.

SUMMARY

1. Column stripping Pb loaded Bayer TP207H ion exchange resin with 12 wt% HNO_3 results in only a slightly higher peak incremental Pb concentration at 1.5 BV/hour than at 3.0 BV/hour (175 g/L compared to 150 g/L)
2. Column stripping Pb loaded Bayer TP207H ion exchange resin with 12 wt% HNO_3 results in a similar peak accumulated Pb concentration at 1.5 BV/hour and 3.0 BV/hour (62 g/L compared to 60 g/L)
3. Pb elution of loaded Bayer TP207H ion exchange resin approaches completion with a similar volume of eluent at 1.5 BV/hour and 3.0 BV/hour (about 2.8 BV). As such, stripping at 3.0 BV/hour allows removal of Pb from the loaded Bayer TP207H ion exchange resin in half the time compared to 1.5 BV/hour without noticeable reduction in strip efficiency.

4. The strip liquor Pb concentration produced with the Purolite C104H resin (tested in the previous month) is much higher than for the Bayer TP207H. Purolite C104H resin can be stripped with 22 wt% HNO3 whereas the maximum recommend HNO3 concentration for Bayer TP207H is 12 wt% to minimize resin degradation.
5. By comparing the results of the Pb column loading test with the bench scale Pb equilibrium loading test using Bayer TP207H ion exchange resin, the loading kinetics of Bayer TP207H resin was relatively close to equilibrium loading conditions at 5 BV/hour.
6. The Pb loading kinetics of the lead column in a two column carousel ion exchange process is only slightly lower in the 2nd cycle load than in the 1st cycle load. Loading kinetics of the 1st and 2nd cycle is faster using Bayer TP207H than for Purolite C104H.

RESULTS AND DISCUSSION

Pb Separation - Ion Exchange Column Studies

Single Column Pb Stripping Tests Using Bayer TP207H Ion Exchange Resin.

Pb stripping tests were conducted on loaded Bayer TP207H ion exchange resin using a single pilot scale column. Tests were conducted by pre-loading Bayer TP207H ion exchange resin with Pb from Yava leachate. Pb loading was conducted in a batch manner using an R/A ratio of 0.04, a feed solution consisting for Fe removed Yava leachate (Fe removed with Cyanex 272 at pH 2.8) with a Pb concentration of 8.31 g/L and a 24 hour loading time. The loaded resin was separated from the raffinate and then water rinsed with deionized water to remove excess leachate and placed inside an ion exchange column for testing. The resin volume (or bed volume) inside the column was 1000 mL for each test. The concentration of Pb loaded onto the Bayer TP207H resin was 150 g/L_{resin}.

Pb stripping was conducted by pumping 12 wt% HNO3 (the maximum recommended HNO3 eluent concentration recommended by the supplier) to the top of the column and then allowing the HNO3 to flow through the resin bed and drain out the bottom of the column into a holding tank. Two flow rates (1.5 BV/hour and 3.0 BV/hour) were tested with the flow rate adjusted with the ball valve the bottom of the column. Incremental samples of strip liquor were periodically collected as the solution exited the column (prior to entering the holding tank). These samples were retained for Pb analysis by ICP.

Test conditions for the single column stripping tests using the Pb loaded Bayer TP207H ion exchange resin are summarized in Table 1 (for greater clarity, flow rate are provided in BV/hr, mL/min and m/hr).

Table 1. Test Conditions for Pb Column Stripping Tests with Purolite C104H.

Resin			**Stripping solution**	**Flow rate**		
Type	**Resin Volume**	**Resin Pb-Loading**		(BV/hr)	(mL/min)	(m/hr)
Bayer TP207H	1000 mL	150-g/L-resin	12-wt% HNO_3	1.5	25	1.25
				3.0	50	2.49

The results of the Pb column stripping tests conducted on the Bayer TP207H ion exchange resin are summarized on the stripping profile (graph of concentration of Pb in the strip liquor versus BV) provided in Figure 1. The Accumulated Pb Concentrations shown in Figure 1 (for 1.5 and 3.0

BV/hour) were calculated by numerical integration of the respective incremental Pb concentration curves.

By comparing the stripping profiles at 1.5 BV/hour and 3.0 BV/hour the following are observed:

- Stripping at 1.5 BV/hour results in only a slightly higher peak incremental Pb concentration than at 3.0 BV/hour (175 g/L compared to 150 g/L)
- Stripping at 1.5 BV/hour results in a similar peak accumulated Pb concentration to 3.0 BV/hour (62 g/L compared to 60 g/L)
- Pb elution approaches completion with a similar volume of eluent at 1.5 BV/hour and 3.0 BV/hour (about 2.8 BV). As such, stripping at 3.0 BV/hour allows removal of Pb from the loaded Bayer TP207H ion exchange resin in half the time compared to 1.5 BV/hour without noticeable reduction in strip efficiency.
- The strip liquor Pb concentration produced with the Purolite C104H resin (tested in the previous month) is much higher than for the Bayer TP207H. Purolite C104H resin can be stripped with 22 wt% HNO3 whereas the maximum recommend HNO3 concentration for Bayer TP207H is 12 wt% to minimize resin degradation.

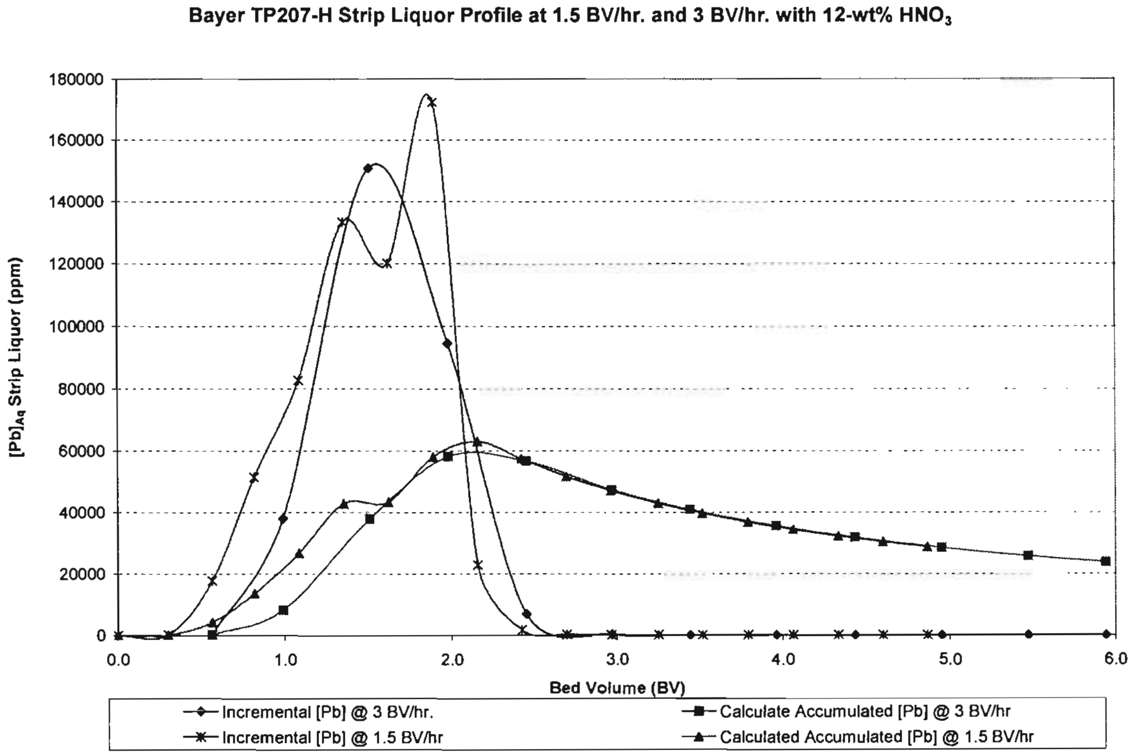

Figure 1. Stripping profile for Bayer TP207H at 1.5-BV/hr and 3.0–BV/hr

Two Stage Column Pb Breakthrough Tests Using Fresh TP207H Ion Exchange Resin

Pb loading breakthrough tests were conducted by connecting two 1.5 inch inner diameter ion exchange columns in series. Each column was filled with 1000-mL of fresh Bayer TP207H resin. Each column was then backflushed with deionized water to obtain a well distributed resin bed prior to testing. The backflush stratification also assists in the removal of damaged resin, excessively fine resin particles and trapped air in the column. The excess water was drained through the bottom valve of each column, leaving a few inches of water above the resin bed.

The feed solution was placed on a weight scale so that flow rate could be calculated with the weight loss and specific gravity of the solution. The feed pump was started and adjusted to the desired flow rate of approximately 83-mL/min (equivalent to 5 BV/hour). The discharge valve of the tail column was adjusted to maintain a level of approximately 4" above each resin bed. A simplified flow diagram of this two stage column loading test with sampling locations identified is shown in Figure 2. Incremental aliquot samples were taken to obtain a loading profile for each column, shown in Figure 3. The aqueous feed solution consisted of Fe removed Yava leachate (with Fe removed by solvent extraction with Cyanex 272). The concentration of Pb in the feed solution (determined by ICP analysis) was 7.879 g/L Pb.

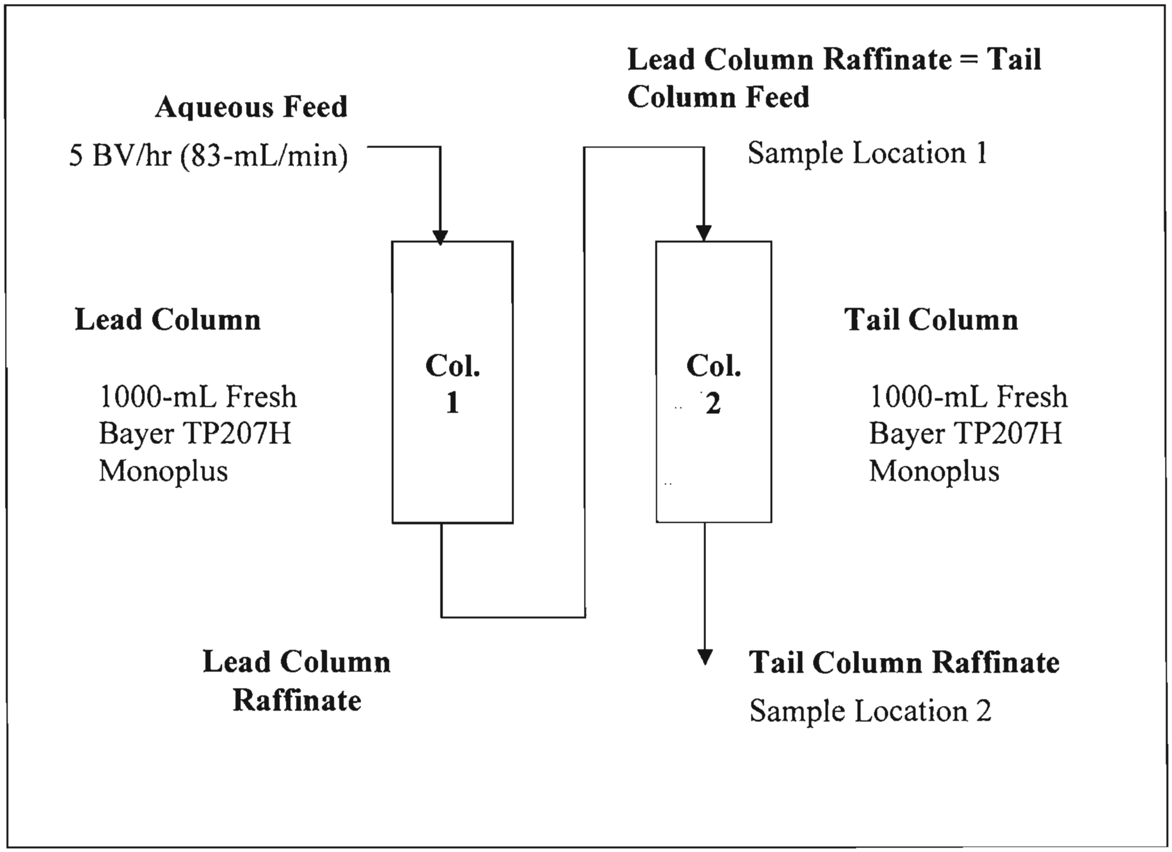

Figure 2: Flow Diagram for Two Stage Loading Test using Fresh Bayer TP207H Ion Exchange Resin – Cycle 1

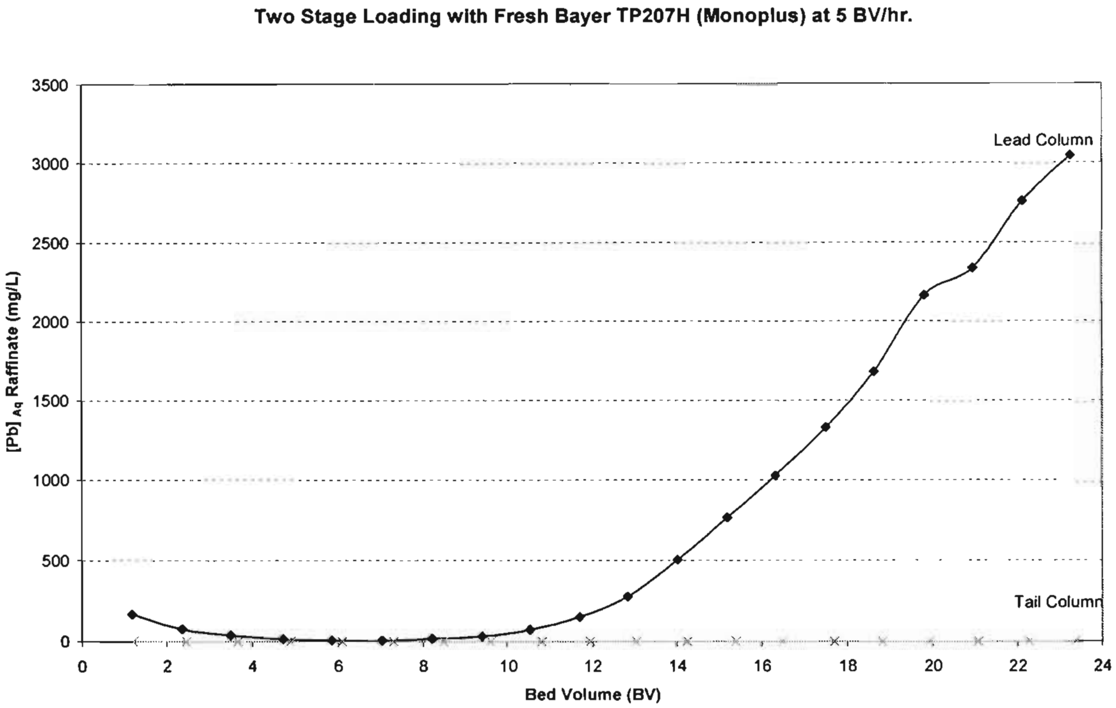

Figure 3: Bayer TP207H two stage loading at 5-BV/hr – Cycle 1

Figure 3 shows that the Lead Column raffinate concentration reaches 3.044 g/L Pb (or 4.835 g/L less than the feed concentration) after 23 BV of Yava leachate was treated with Bayer TP207H ion exchange resin at 5 BV/hr. The R/A ratio for this column test (calculated as 1 BV resin divided by 23 BV leachate) was approximately 0.04. After the first load cycle, the concentration of Pb in the loaded ion exchange resin was calculated to be 166 g/L_{resin} in the lead column and 17 g/L_{resin} in the tail column.

In comparison, bench scale test work conducted to determine the equilibrium loading for Bayer TP207 Monoplus ion exchange resin at an R/A ratio of 0.04, Yava leachate feed concentration of 8.310 g/L Pb and 24 hour treatment time resulted in a raffinate with 2.349 g/L Pb (or 5.961 g/L less than the feed concentration). The close reduction in Pb concentration in solution after treatment in the column in comparison with the bench scale equilibrium test suggest that the 5 BV/hr provide close to equilibrium Pb loading. As such, loading kinetics for Bayer TP207H is much faster than for Purolite C104H tested in the previous month.

The low concentration of Pb in the tail column raffinate indicates that the tail column is effective in capturing Pb that is missed in the first column. Note that this is only relevant for a single cycle test. Multi-column ion exchange systems typically operate in a carousel type process where the loaded lead column is stripped to remove loaded metals and then moved to become the tail column and the previously partially loaded tail column moves towards becoming the lead column. Depending on loading kinetics, solution flow rate, BV treated and number of stages it is possible that metal ions may accumulate in the ion exchange system resulting in a different loading curve for each load/strip cycle.

Two Stage Column Pb Breakthrough Tests Using Bayer TP207H Monoplus Ion Exchange Resin – Second Cycle in a Carousel Type Process

The two stage Pb load test in the previous section was repeated except by moving the previously partially loaded tail column to become the lead column and placing fresh Bayer TP207H resin in the tail column (to simulate stripped resin). This test was conducted to illustrate the possible change in breakthrough curve for the second load cycle in a carousel type ion exchange process. As in the first cycle test, each column was then backflushed with deionized water to obtain a well distributed resin bed prior to testing. The excess water was drained through the bottom valve of each column, leaving a few inches of water above the resin bed.

Load testing was conducted with 1000 mL of resin in each column, a solution flow rate of 5 BV/hr, 23 BV of solution treated and 7.879 g/L Pb in the feed solution. Incremental aliquot samples were taken of the lead column and tail column raffinates to obtain loading profiles. A flow diagram of the two stage, 2nd cycle test is shown in Figure 4.

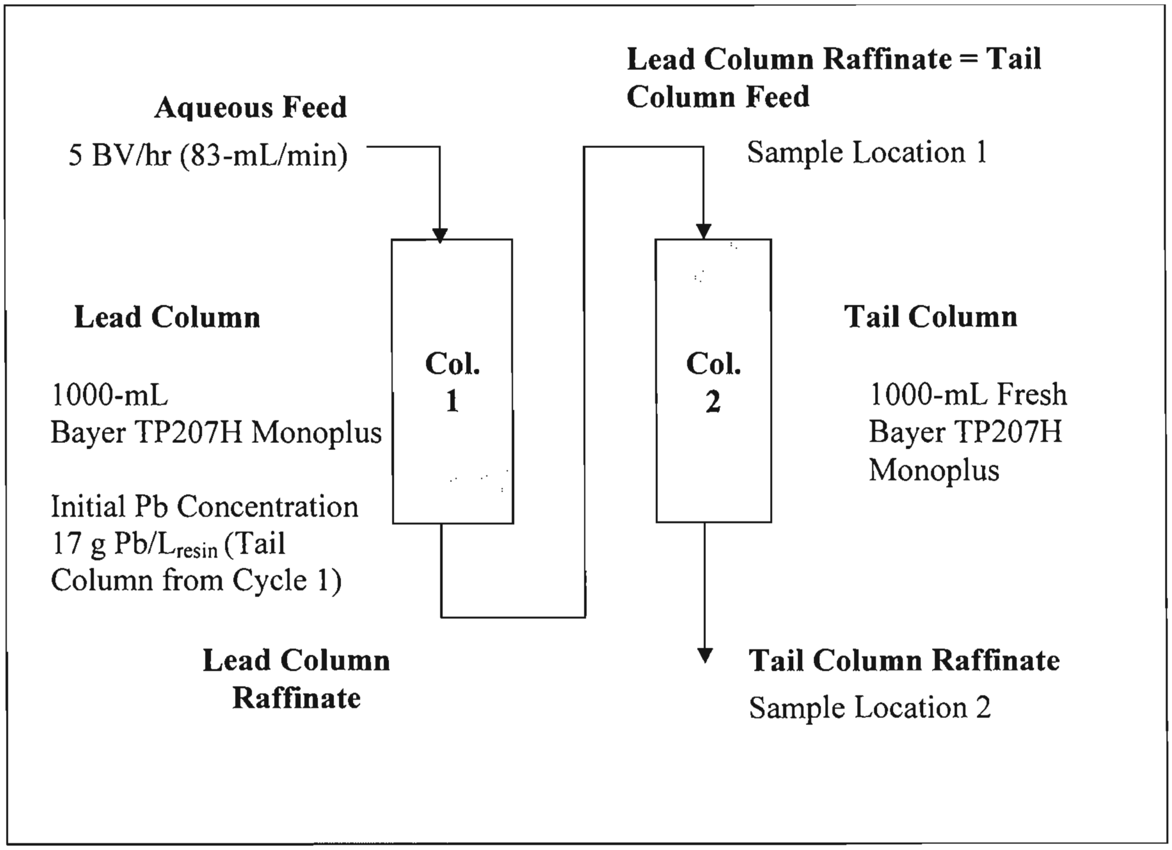

Figure 4: Flow Diagram for Two Stage Loading Test using Bayer TP207H Ion Exchange Resin – Cycle 2

The loading profile for each column in the 2nd cycle loading test is shown Figure 5 (cycle 1 profile provided for comparison). The lead column loading profiles for cycle 1 and cycle 2 using the

Bayer TP207H ion exchange resin have much greater similarity that the cycle 1 and cycle 2 loading profiles for Purolite C104H resin tested in the previous month.

With Bayer TP 207H cycle 1 reaches a raffinate concentration of 1000 mg/L in approximately 16 BV whereas Cycle 2 reaches a raffinate concentration of 1000 mg/L in approximately 12 cycles (where 1000 mg/L Pb raffinate concentration was selected arbitrarily to illustrate the comparison). The BV treated to achieve the same Pb raffinate concentration is only slightly lower for cycle 2 than for cycle 1. This indicates that loading kinetics is only slightly slower in Cycle 2 than in Cycle 1.

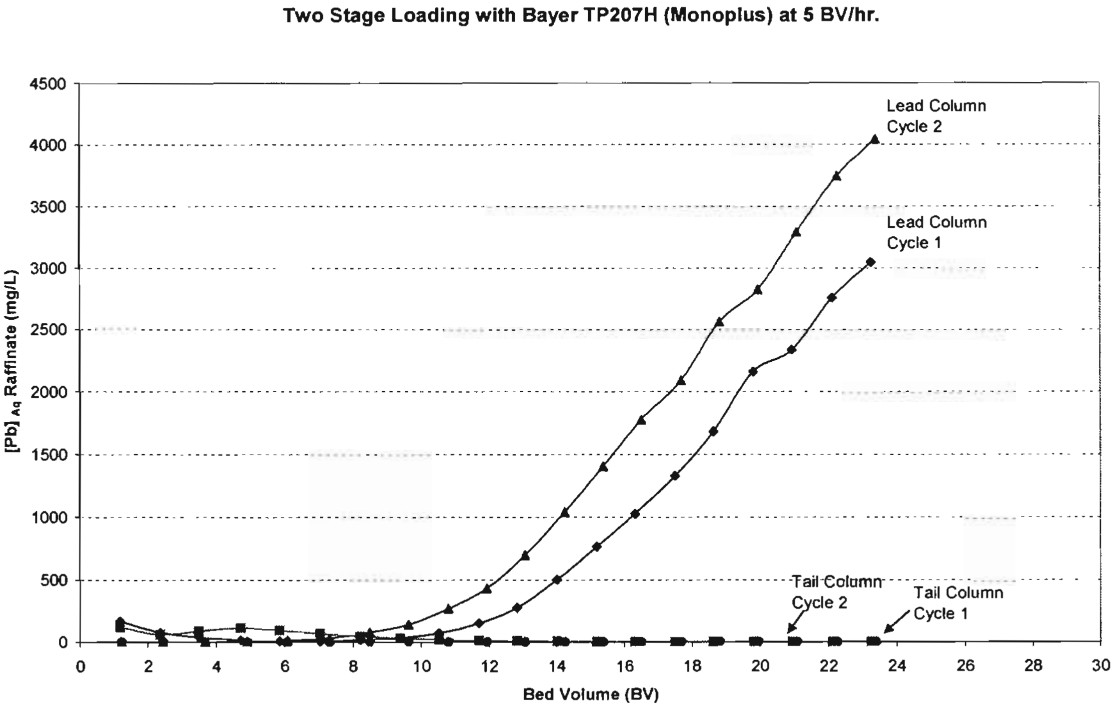

Figure 5: Bayer TP207H two stage loading at 5-BV/hr – Cycle 1 and Cycle 2

After the 2nd load cycle, the concentration of Pb in the loaded ion exchange resin was calculated to be 177 g/L_{resin} in the lead column and 26 g/L_{resin} in the tail column. In comparison, Bayer TP207H loads Pb much faster than for Purolite C104H tested in the previous month. In addition, loading kinetics for subsequent cycles in carousel type ion exchange process does not degrease as much for Bayer TP 207H as it does for Purolite C104H.

Appendix G

APPENDIX G

LIST OF REPORTS OF INVESTIGATIONS PERFORMED BY ELECTROSYNTHESIS INC.

1

Research & Technology

Progress Report - Electrochemical Salt Splitting for Recycle of Ion Exchange Regenerants

Date: December 23, 2005

Report for:

Yava Technologies Inc.

Prepared by:

Dennis Chai, Scientist
J. David Genders, President

Electrosynthesis

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5 Future Work **10**

1 Summary

This interim report will cover initial testing of the electrochemical salt-splitting of synthetic $NaNO_3$ and Na_2SO_4 solutions using bipolar membrane technology to produce pure acid and base products. To date, we have examined the relationships between the solution strengths, cell voltage, current efficiency and water transport. We have shown that in general, as product concentrations decline, current efficiency, cell voltage and water transport all increase. We have also demonstrated the longer term stability of the system (over a period of 25hours for the $NaNO_3$ system and 51hours for the Na_2SO_4 system). There was some accumulation of a brownish colored precipitate in the flow channels of the feed compartments during the Na_2SO_4 experiments and this resulted in reduced flows and slightly higher cell voltage. When the flows were re-adjusted the cell voltage returned to normal levels. Addition of a filter to the feed loop should remove any insolubles in the solution and thus avoid this problem.

Future work will involve testing of the sodium acetate system under similar conditions.

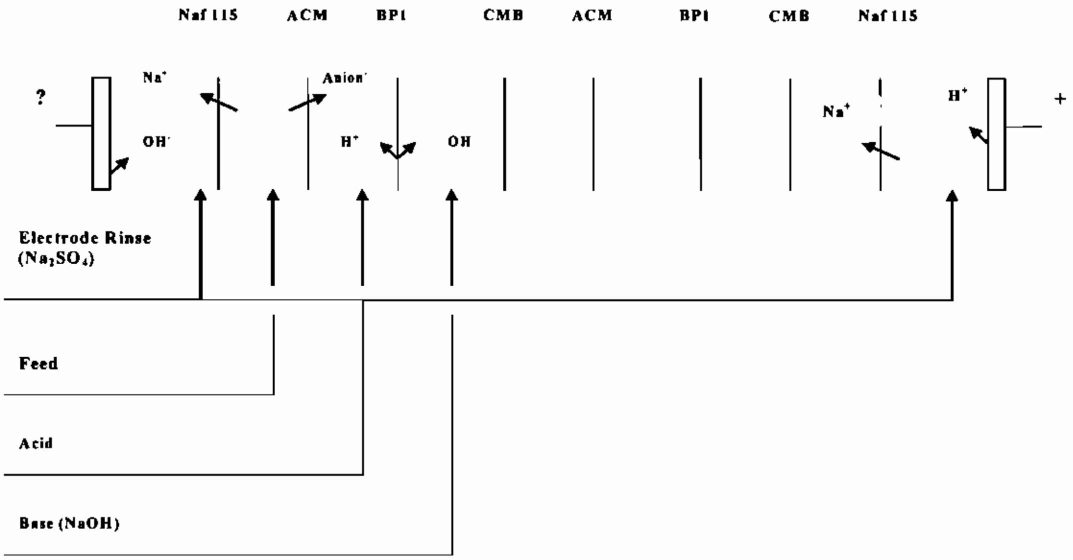
2 Introduction

Yava Technologies Inc. (Yava) would like to recycle regenerants for several ion exchange steps that are part of a metal extraction process currently being developed by Yava in conjunction with Noram and Vizon. The Electrosynthesis Company Inc. (ESC) was contracted to demonstrate electrochemical salt-splitting utilizing bipolar membranes to produce pure NaOH and acid (or a mixed acid and sodium acetate stream in the case of sodium acetate salt-splitting) for the regeneration of ion exchange resins. IN the initial phase 1 of this contract we are using reagent grade starting materials; the proposed phase 2 will investigate longer term performance of the system using "real" salt solutions produced from regeneration of ion exchange resins.

3 Experimental

The experiments were carried out in an ED-1BP Electrodialysis cell equipped with a platinized titanium anode and a nickel cathode. Each membrane set consisted of a Neosepta CMB cation exchange membrane, a Neosepta BP-1 bipolar membrane, and a Neosepta ACM anion exchange membrane. Five membrane sets were used for a total operating surface area of 0.05m^2. The gaskets were made of EPDM and the turbulence promoters were made of polypropylene. A diagram of the membrane configuration (with two out of five sets) is shown below.

ED-1BP Membrane Configuration



The feed compartment consisted of a 10 liter glass reservoir and an Iwaki WMD-30LFX-115 magnetic drive centrifugal circulation pump. Feed solution was fed from a 30 gallon tank by a peristaltic pump to the glass reservoir at approx. 500 ml/min (20 min reservoir turnover time) and then overflowed by gravity back to the 30 gallon tank. This ensured that the feed concentration changed very slowly with time. Solution was pumped from the glass reservoir to the electrodialysis cell and then back to the glass reservoir. The pH of the solution was monitored using a Jenco #3676 pH controller and pH electrode. When sodium sulfate was used as the feed, the pH was controlled at above 4.5 by the addition of NaOH fed through a solenoid valve (activated by the pH controller) to the top of the reservoir. The pH of the solution was controlled in order to ensure that the anionic species were present as SO_4^{2-} as opposed to the HSO_4^- species. The temperature, inlet pressure and flow rate of the feed solutions were all monitored and recorded.

The acid compartment consisted of a 1 liter glass reservoir and similar circulation pump. Dilution water was added to the reservoir at a measured rate and the diluted acid product overflowed by gravity to a product collection tank. The inlet pressure and flow rate of the solution were all monitored and recorded. The base compartment consisted of a reservoir and circulation pump similar to that used for the feed loop. Dilution water was also added to the base compartment at a measured rate and the diluted product (NaOH) overflowed by gravity into a collection tank. The inlet pressure and flow rate were monitored and recorded. To allow for safe overnight operation, several safety shutdowns were incorporated into the system, including low solution flow rate and spill detectors.

The anolyte and catholyte rinse streams were combined and were made up of sodium sulfate (5% by weight, 1 liter). The electrode reactions are as follows:

Anode: $2\,H_2O \rightarrow 4\,H^+ + O_2 + 4\,e^-$

Cathode: $2\,e^- + 2\,H_2O \rightarrow 2\,OH^- + H_2$

Overall: $2\,H_2O \rightarrow O_2 + 2H_2$

Combining the anolyte and catholyte maintains pH neutrality in the electrode rinse stream.

The cell was operated at 80 mA/cm^2 (8 amps total current), 35^{0}C, with circulation rates initially set at approx. 100 ml/min per individual frame (as recommended by the manufacturer) for a total of 500 ml/min for each compartment.

3.1 Sampling and Analysis

Samples of acid, base and feed solutions were taken frequently and analyzed. Both acid and base samples were analyzed for strength using a simple acid/base titration. In addition, the acid samples were analyzed for Na^+ contamination and the base samples for anionic contamination by ion chromatography. The feed samples were also analyzed by ion chromatography for anionic strength (NO_3^-, SO_4^{2-}).

4 Results and Discussion

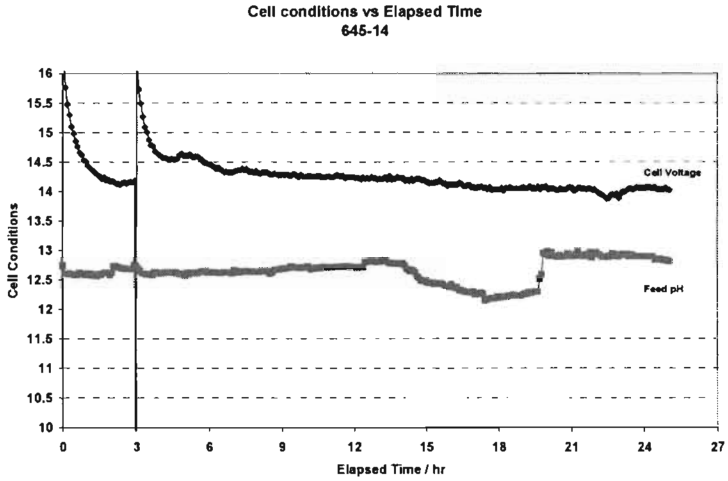
A summary of the results obtained for salt-splitting of $NaNO_3$ can be found in Table 1 and Na_2SO_4 can be found in Table 2.

4.1 Salt-splitting of $NaNO_3$

Experiments were performed at several different base and acid product strengths. The results indicate that as the NaOH product strength was increased from 5.6% to 7.5% (exp. 645-06 and 645-12) the current efficiency for its production dropped from 72.7% to 60%. This is primarily due to back-migration of OH^- ions across the cation membrane into the feed compartment. Water transport on the base side varied between 6.1 to 6.7 moles H_2O/mole Na^+. IC analysis for NO_3^- contamination indicated an average of approx. 44mM, corresponding to between 1-2% membrane inefficiency.

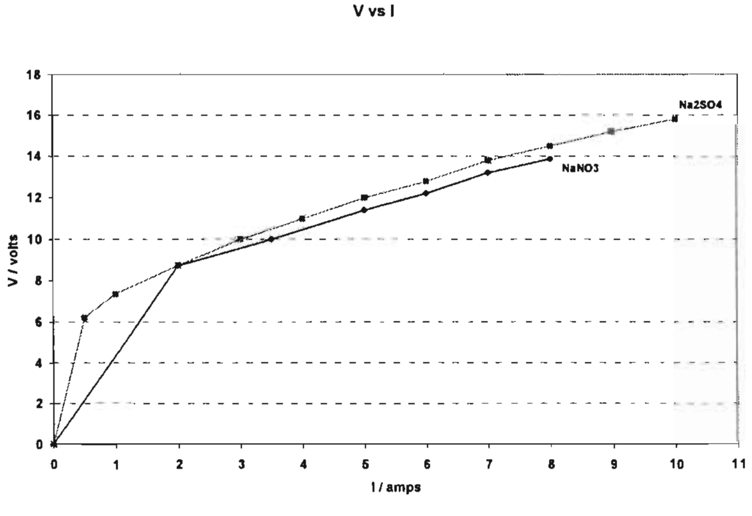
For the same experiments, the acid product (HNO_3) strength was raised from 7.8% to 9.9% and this resulted in a current efficiency decline from 79.3% down to 74.4% (due to proton back-migration across the anion membrane). Water transport varied between 1.8 to 2.0 moles H_2O/mole NO_3^-. IC analysis for Na^+ contamination indicated an average of 233 ppm (< 1% membrane inefficiency)

The cell voltage dropped slightly (down from 14.3V to 14.1V) as the product strengths were increased, reflecting the increased conductivity of the solutions. In one of the experiments (645-14), the cell was run for an extended period of time (25hrs) to evaluate its stability. The cell voltage and feed pH profile are shown below.



The cell voltage stabilized at 14.1 volts. Current efficiencies and water transport numbers measured near the beginning and end of the run were similar (see Table 1). The feed pH remained alkaline throughout the run, reflecting the fact that the there was more back-migration of hydroxyl ions than of proton into the feed compartment. A V/I curve was generated and is shown in Figure 1 below.

V/I curve for ED Stack.



The curve shows that at current densities of up to 80 mA/cm^2 (8 Amps) the system is not yet mass transport limited and could theoretically be run at higher current densities (this is not usually recommended by the manufacturer as it can result in excessive shunt currents in the cell stack resulting in permanent damage to the membranes).

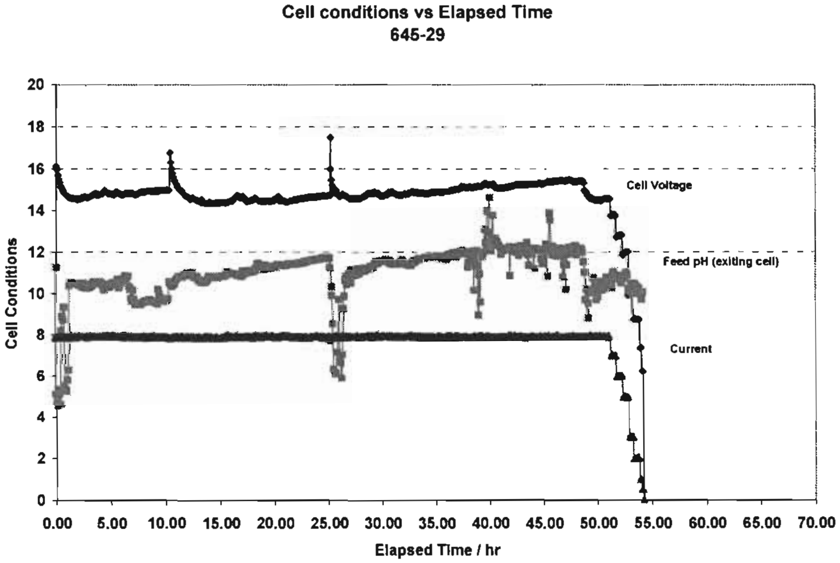
4.2 Salt-splitting of Na_2SO_4

Experiments were run at nominal base product strengths of 6% and 5% and nominal acid product strengths of 8%, 5%, and 4%. The results (Exp. 645-25 to 645-29e) indicate that as the base (NaOH) product strength was dropped from 5.9% to an average of 5.0%, the current efficiency rose from 70.8% up to an average of 72.6%. IC analysis for SO_4^{2-} contamination indicated an average of approx. 9mM, corresponding to approx. 1% membrane inefficiency.

For the same experiments, decreasing the acid (H_2SO_4) strength from 4.8% down to an average of 4.25%, the current efficiency rose from 70% up to an average of 74.7%. Na^+ contamination of the acid product averaged 228 ppm (<1% membrane inefficiency). One experiment was conducted at acid strength of 8.3% (exp.645-20) and an acid current efficiency of 56.2% was obtained. No data for the performance of the base side was used from this run as a small amount of cross-flow from the feed to the base side was suspected (as shown by higher than normal contamination with SO_4^{2-}).

The cell voltages were slightly higher than for $NaNO_3$ salt splitting and were also generally higher when lower strength product was made and when the feed concentrations were similar. The cell voltage rose from 13.8 V at 6% base and 5% acid product to 14.4 V at 5% base and 4.25% acid product. A V/I curve was generated and is shown in Figure 1 above. It also indicates that the system is not yet mass transport limited at current densities of up to 100 mA/cm^2 (10A).

An experiment was conducted to evaluate the effect of feed concentration on the water transport across the membranes. Starting with feed strength of 1.16M Na_2SO_4 (Exp. 645-29b) the experiment was run continually for 51 hours and the current efficiencies and water transport numbers calculated as the feed was slowly depleted down to 0.61M (Exp. 645-29e). The results of Exp. 645-29a were not included so as to allow some time for equilibration of the membrane, particularly after it had been run with low concentration feed in the previous two experiments (645-25 and 645-27). The product concentrations were kept constant throughout the run. In addition, it was necessary to manually dilute the feed solution on two occasions in order to reduce its concentration in a timely manner. A diagram of the cell voltage and feed pH profile is shown below. The results show that the feed pH remained alkaline throughout the run again indicating that there is more hydroxide back-migration than proton back-migration. The cell voltage was relatively constant at first, but rose slowly as time progressed. This voltage increase was due to a drop in the flow rate of the feed solution over time. When the feed flow rate was re-adjusted, the cell voltage dropped back to normal levels (at 49hrs elapsed time). During the course of the sodium sulfate salt-splitting work, the backpressure in the feed compartment increased significantly over time. Upon opening the cell for inspection, it was found that a brownish precipitate had accumulated in the feed entry flow channels, and this was the likely cause of the reduced flow/higher inlet pressures observed. It is assumed that this was insoluble material introduced with the sodium sulfate and it should be possible to avoid this problem by installing a filter in the feed loop.



The results (Table 2) also show an increase in the water transport as the feed solution was depleted. The water transport across the cation membrane increased from 4.1 to 5.6 moles H_2O/ mole Na^+ and the water transport across the anion membrane increased from 3.7 to 4.8 moles H_2O/ mole SO_4^{2-}.

5 Future Work

We will reconfigure the cell and rebuild with new gaskets and membranes to test the salt-splitting of sodium acetate solutions. It will produce NaOH and a mixed stream of acetic acid and sodium acetate. As outlined in the proposal, we will run experiments at several feed and product concentrations and evaluate their effects on voltage, current efficiency and water transport. We will also analyze for acetate contamination in the base product.

Table 1

Electrochemical Salt-splitting of $NaNO_3$ with Bipolar Membranes

Run	Feed	Base Strength (%w/w)	Base CE %	Base Water Transport Moles/mole Na^+	Acid Strength (%w/w)	Acid CE %	Acid Water Transport Moles/mole NO_3^-
645-06	**1.1M $NaNO_3$**	5.6	72.7	6.7	7.8	79.3	2.0
645-12	**0.77M $NaNO_3$**	7.5	60	6.1	9.9	74.4	1.8
645-14	**1.10M $NaNO_3$**	6.3	71.8	5.1	9.8	73 5	1.8
645-14 cont'd	**0.83M $NaNO_3$**	6.1	67.5	4.7	9.8	73.7	2.2

Table 2

Electrochemical Salt splitting of Na_2SO_4 with Bipolar Membranes

Run	**Feed**	**Base Strength (%w/w)**	**Base CE %**	**Base Water Transport Moles/mole Na^+**	**Acid Strength (%w/w)**	**Acid CE %**	**Acid Water Transport Moles/mole SO_4^{2-}**
645-20	**1.0M Na_2SO_4**	-	-	-	8.3	56.2	4.3
645-22	**0.87M Na_2SO_4**	-	-	-	4.7	69.8	4.8
645-25	**0.92M Na_2SO_4**	5.9	70.8	5.6	4.8	70.0	3.0
645-27	**0.96M Na_2SO_4**	5.3	71.4	5.5	4.25	74.0	4.8
645-29a	**1.20M Na_2SO_4**	5.2	75.0	4.6	4.28	74.9	4.8
645-29b	**1.16M Na_2SO_4**	5.1	72.2	4.1	4.3	74.5	3.7
645-29c	**0.97M Na_2SO_4**	4.89	69.5	3.8	4.2	75.0	3.8
645-29d	**0.88M Na_2SO_4**	4.89	74.4	5.1	4.2	74.3	4.2
645-29e	**0.61M Na_2SO_4**	4.80	73.2	5.6	4.3	75.6	4.83

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June 29

NOVA SCOTIA
Natural Resources

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Map
Refs. ____________

Form 10 - Statement of Assessment Work Expenditure

(pursuant to the *Mineral Resources Act*, S.N.S. 1990, c. 18, s. 43(1))

(Complete as necessary to substantiate the total claimed.) May 1, 2005 - April 30, 2006 Final

pecial Re: Licence No. 1/03 Date of issue July 16, 2003

	Type of Work		Amount Spent ($)
1	Prospecting	____ days	
2.	Geological mapping	____ days	
3.	Trenching/stripping/refilling	____m² / ____m³	
4.	Assaying & whole rock analysis	____ #	12,232
5.	Other laboratory	____ #	
6.	Grid:		
	(a) Line cutting	____ km	
	(b) Picket setting	____ km	
	(c) Flagging	____ km	
7.	Geophysical surveys **Airborne:**		
	(a) EM/VLF	____ km	
	(b) Mag or Grad	____ km	
	(c) Radiometric	____ km	
	(d) Combination	____ km	
	(e) Other ____	____ km	
8.	Geophysical surveys **Ground:**		
	(a) EM/VLF	____ km	
	(b) Seismic soundings	____ #	
	(c) Magnetic/telluric	____ km	
	(d) IP/resistivity	____ km	
	(e) Gravity	____ km	
	(f) Other Hydrogeological	____ km	142,379
9.	Geochemical surveys		
	(a) Lake, stream, spring		
	(i) Water	____ samples	
	(ii) Sediments	____ samples	
	(b) (i) Rock	____ samples	
	(ii) Core	____ samples	
	(iii) Chips	____ samples	
	(c) (i) Soil	____ samples	
	(ii) Overburden	____ samples	
	(d) Gas	____ samples	
	(e) Biogeochemistry	____ samples	
	(f) Sample collection	____ days	
	(g) Other ____	____	
10.	Drilling:		
	(a) Diamond (# holes/m)	____ / ____ m	
	(b) Percussion (# holes/m)	____ / ____ m	
	(c) Rotary (# holes/m)	____ / ____ m	
	(d) Auger (# holes/m)	____ / ____ m	
	(e) Reverse circulation (# holes/m)	____ / ____ m	
	(f) Logging, supervision, etc.	____ days	
	(g) Sealing (# holes)	____ #	
11	Other (describe)		
	Pilot plant operations		657,263
		Subtotal	811,874
	Overhead costs		
12	Secretarial services		
13	Drafting services		
14.	Office expenses (rent, heat, light, etc.)		
15	Field supplies		
16.	Compensation paid to landowners		500
17.	Legal fees		
18	Other (describe)		
		Subtotal	500
		Grand total	$ 812,374

List the names of the persons who conducted the work reported in the previous table and the dates during which the work was performed.

Name	Address	Dates Worked
SGS Minerals	185 Concession St. Lakefield, ON K0L 2H0	MAR/06
CETI	6 Southland Cr. Ottawa, ON K1G 5E4	MAY - AUGUST/05
Robert Boudreau	1175 St. Emmanuel Terrace Orleans, ON K1C 2J6	SEP/05 - APR/06
J.D. Crossley	1560 King Vaughn Road Maple, ON L6A 2A7	OCT/05 JAN-FEB/06
John Labrecque	1084 Candlewood St. Ottawa, ON K4A 5E9	OCT/05
Jacques Whitford	3 Spectacle Lake Dr. Dartmouth, N.S. B3B 1W8	MAY/05 - APR/06
Golder Associates	2390 Argentia Rd. Mississauga, ON L5N 5Z7	APR/06
Vizon SciTec Inc	3650 Westbrook Mall Vancouver, B.C. V6S 2L2	MAY/05 - APR/06
Centaur Mining Exploration Ltd	20 York Valley Cr. Toronto, ON M2P 1A7	May/05 - Apr/06
Madhav Dahal & Associates	265 Markland Dr Unit 706 Toronto, ON M9C 1R5	May/05 - Apr/06
Electrosynthesis Co.	72 Ward Rd Lancaster, NY. 14086	Oct/05 - MAR/06
IBC Advanced Technologies	856 East Utah Valley Dr. American Fork, UT 84003	Dec/05
Lakefield Research Ltd	185 Concession St. Lakefield, ON K0L 2H0	July/05
NORAM ENGINEERING & CONSTRUCTORS LTD.	200 GRANVILLE ST. #400 VANCOUVER, B.C. V6C 1S4	MAY/05 - APR/06

I hereby certify that the information in this form is true and correct, that it has not before been submitted for assessment work credit and that it is the total of all work conducted on the licence during the past licensed year.

As PRESIDENT I am duly authorized to make this certification.
(position in company or licensee)
Dated at TORONTO in the Province of ONTARIO on July 24/06
Name and address of licensee: YANA TECHNOLOGIES INC.
20 YORK VALLEY CR., TORONTO, ON M2P 1A7

Signature

For further Information, contact the Registrar of Mineral and Petroleum Titles at 1-902-424-4068.