ARD/AMD POTENTIAL
OF THE
WASTE MANAGEMENT AREA

GOLDCORP INC.

RED LAKE MINE DIVISION
BALMERTOWN, ONTARIO

FINAL REPORT

NOVEMBER, 1997

ARD/AMD POTENTIAL OF THE WASTE MANAGEMENT AREA
GOLDCORP INC., November, 1997

TEXT: **GOLDFIN1.WPD** **[29.5]** COPY: GOLDFIN2.WPD [29.6]

TABLES

Table 1:	[29.6]	GC-DATA3.XLS	
Table 2:	[29.6]	GC-DATA1.XLS	GC-ABA1
Table 3:	[29.6]	GC-DATA1.XLS	HUMIDITY
Table 4:	[29.6]	GC-DATA1.XLS	SITE DESCRIPTIONS
Table 5:	[29.6]	GC-DATA1.XLS	Phyto and Peri I.D.
Table 6:	[29.6]	GC-DATA1.XLS	Water Sampling
Table 7:	[29.5]	TAILTON.WB2	
Table 8a:	[29.5]	TABLE8.WPD	integrated in text
Table 8b:	[29.5]	TABLE8.WPD	integrated in text
Table 8c:	[29.5]	TABLE8.WPD	integrated in text
Table 9:	[29.5]	MINERALS.WB3	
Table 10:	[29.5]	SEQUENT2.XLS	
Table 11:	[29.6]	SOLSUM.XLS	
Table 12:	[29.6]	GC-DATA1.XLS	Solid Slurries
Table 13:	[29.6]	GCEXTR.WB2	
Table 14:	[29.5]	PIEZOWS1.WB2	

FIGURES

Figure 1:	[29.6]	GC-DATA1.XLS	NP vs AP graph
Figure 2:	ZIP	PARTAQ.CDR	converted through wpg file to PARTAQ.WPD to print
Figure 3:	[29.6]	GCT.WB3	
Figure 4:	[29.5]	PIEZOWS1.WB2	
Figure 5:	[29.5]	PIEZOWS1.WB2	
Figure 6:	[29.5]	PIEZOWS1.WB2	
Figure 7:	[29.5]	PIEZOWS1.WB2	
Figure 8:	[29.5]	PIEZOWS1.WB2	
Figure 9:	[29.5]	PIEZOWS1.WB2	
Figure 10:	[29.5]	PIEZOWS1.WB2	
Figure 11:	[29.5]	PIEZOWS1.WB2	
Figure 12:	[29.5]	PIEZOWS1.WB2	

MAPS

Map 1:	ZIP	GOLDCORP.CDR, MAP1.WPD
Map 2:	ZIP	COMBO-1.CDR, MAP2.WPD
Map 3:	ZIP	MAP3.CDR, MAP3.WPD
Map 4:	ZIP	MAP4.CDR, MAP4.WPD

All cdr files (including figure2) are zipped to MAPCDR
All word peerfect graphic files are zipped to MAPWPD

APPENDIX A (TABLES) table of contents in : APPENDA.WPD

Table 1:	[29.5]	TAILTTON.WB2	
Table 2:	[29.6]	GC-DATA1.XLS	SOLIDS ICP
Table 3:	[29.6]	GCT.WB3	
Table 4:	[29.6]	GTTABLE2.WB2	
Table 5:	[29.5]	SEQUENT2.XLS	Methods1
Table 6:	[29.6]	GCEXTR.WB2	
Table 7:	[29.6]	GC0797.WB2	
Table 8:	[29.6]	GC0797S.WB2	
Table 9:	[29.5]	PIEZOWS1.WB2	
Table 10:	[29.5]	PIEZOS2.XLS	
Table 11:	[29.5]	SEQUENT2.XLS	
Table 12:	------	photocopy	
Table 13:	[29.5]	TABLE13.WPD	

APPENDIX B (FIGURES) table of contents in: APPENDB.WPD

All figures (1 to 35) are in the file:
[29.5] PIEZOWS1.WB2

SUMMARY

As part of the decommissioning plans of Goldcorp Inc., the potential of acid generation from the tailings was addressed with acid base accounting tests. The occurrence of acid generating minerals along with neutralizing minerals is extremely varied and the limited number of ABA tests lead to uncertain results. Boojum Research Ltd. was retained to assess the acid generation potential further.

From a field investigation of the surface water on the tailings it was concluded, that within one year of cessation of tailings discharge, the physical/chemical conditions of the ponded water resembles those of the surrounding fresh water and supports considerable biological activity. The pH in ponded water on the tailings ranged from 7.0 to 8.6 and in the sediments/tailings pH values as high as 9.1 were measured. The electrical conductivity was low, with values ranging between 200 µmhos/cm to 1000 µmhos/cm. These surface waters would facilitate ecological approaches to decommissioning.

To determine a reliable estimate of the quantity of acid generating minerals in the tailings, data were extracted from mineralogical and milling records in addition to chemical analysis reported for tailings. The data consistently produce an average of 1.5% S, 6.7% Fe and 2.7% Al. Sequential extraction of the tailings for mineral association produced a weight loss of 42%, the remaining 68% of the tailings are totally inert. Of the digestible fraction, 29% consisted of alkalinity generating minerals and 21% of potentially acid generating minerals, thus the neutralizing potential mass is equal to that of the acid generating mass.

Through the extraction of weathering products which had formed in the tailings the ongoing oxidation, leading to acid generation and the concurrent neutralization was determined. Tailings material collected from old (32 years) and new (10 years) tailings ponds was leached with distilled water. The leach solutions were all above pH 7 and alkalinity between 50 to 210 $mg \cdot L^{-1}$ $CaCO_3$ equivalent. The tailings material clearly has remaining neutralization potential left. In the leachate from visually oxidizing tailings, 36%

of S in the solids was mobilized with water, along with 14% Ca after 10 years of exposure in the tailings pond. In the visually unoxidized sample exposed for the same time only 2.3% of the S and 2.2% of the Ca could be liberated by distilled water. Acid generation and its concurrent neutralization appears to be localized in pockets of the tailings pond. The rate of acid generation and neutralization has not been determined. It can be concluded that effluent problems normally associated with acid generating tailings will not be encountered for the Goldcorp Inc. tailings area. The concentrate stored on the tailings is acid generating. It is at present unclear, if some piezometer water quality is affected by this material.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF MAPS

LIST OF PLATES

1.0 INTRODUCTION

Boojum Research Ltd. was requested to review the Acid-Base accounting data generated during an Environmental Testing Program carried out by CESL Engineering on three samples (fresh raw tailings, old deposited tailings and mine waste rock) which were provided for testing by GOLDCORP on November 4, 1994. The three samples were tested using Acid-Base accounting techniques generally accepted by the industry. A brief commentary of the results is presented to provide the background to the approach chosen in this study, which has the objective to clarify the acid generation question specifically for the long term for the GOLDCORP waste management area.

1.1 Review of Acid-Base Accounting Tests

Data were received at Boojum Research Limited by fax on January 30, 1997. Analytical data for old tailings, new tailings and waste rock samples were provided. Acid base accounting data were supplied for each of the 3 samples. Complete humidity cell experimental data were provided.

1.1.1 Whole Rock Analyses by ICP

Arsenic concentrations in assays of whole samples of old tailings, new tailings and waste rock are 1,615; 1,293 and 2,286 $mg \cdot kg^{-1}$ (0.16, 0.13 and 0.23%) respectively, in the old and new tailings and waste rock samples (Table 1). Old tailings contain more arsenic than new tailings. Waste rock contains the highest concentrations of As. Metal concentrations, including Co, Cu, Ni, Pb, Zn and Cr, range from 22 $mg \cdot kg^{-1}$ to 243 $mg \cdot kg^{-1}$ (Table 1). Overall, arsenic is likely the primary contaminant of concern apart from acid generating potential, while metals are found at relatively low concentrations.

Table 1: GOLDCORP INC.
ICP Analyses of Solids subjected to acid base accounting

Element	Old Tailings	New Tailings	Waste Rock
Al	16200	18400	15200
As	1615	1293	2286
Ba	57	118	108
Ca	28600	27900	31100
Co	25	22	28
Cu	114	115	109
Fe	74800	78500	67000
K	4000	6200	5400
Li	27	41	38
Mg	18400	24400	23500
Mn	1358	1510	1356
Na	700	800	700
Ni	147	131	172
P	330	410	350
Pb	95	176	37
S, total	22500	14600	19300
S as SO_4	1400	700	1700
Sb	61	70	63
Sr	94	100	99
Ti	700	1100	900
V	197	236	186
Zn	98	92	70
W	26	32	25
Cr	215	243	229

N.B. All measurements are in $ug.g^{-1}$

All elements at or below 10 ug.g-1eliminated. See details in Appendix

Hawley, J.R., 1979/1980. The Chemical Characteristics of
Mineral Tailings in the Province of Ontario. MOE Report.

1.1.2 Acid Base Accounting

The old tailings sample reported the lowest NNP value, compared to the new tailings sample and the waste rock sample. This could be interpreted as an indication that the neutralizing potential in the waste material diminishes with time. The mineralogy of the deposit indicates that arsenopyrite is present, and the likelihood that some form of AMD is taking place is quite high. However, a net acidic generating potential from the waste will depend on the distribution and weatherability of the acid neutralizing material in the wastes.

Paste pHs all exceed pH 7, ranging from 7.25 to 8.55 (Table 2). Sulphur concentrations in rock samples tested were moderate, and acid production estimates (AP, in kg $CaCO_3 \cdot t^{-1}$) due to sulphide mineral oxidation, anticipated from the S concentrations, are similar for the three whole tailings and waste rock samples, ranging from 43.4 to 65.9 kg $CaCO_3 \cdot t^{-1}$. Total sulphur concentrations are 1.46%, 1.93% and 2.25% in new tailings, waste rock and old tailings, respectively. Iron concentrations are 5.7%, 7.35% and 7.48% in waste rock, new tailings and old tailings, respectively.

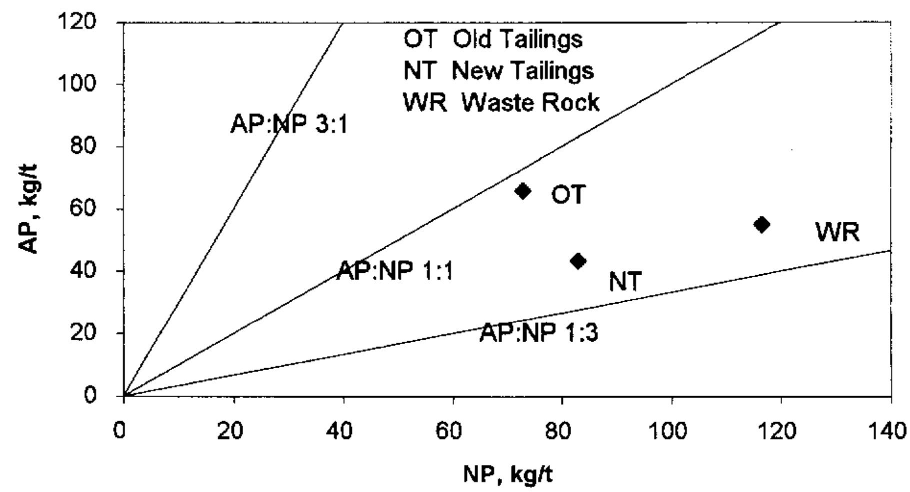
Table 2: GOLDCORP INC. Modified Acid Base Accounting Data

	kg/t NP	kg/t AP	Code	Paste pH	% S (T)	% S (SO_4)	Net kg/t NP	NP/AP
Old Tailings	72.8	65.9	OT	7.25	2.25	0.14	6.9	1.10
New tailings	83	43.4	NT	8.55	1.46	0.07	39.6	1.91
Waste Rock	116.5	55	WR	8.44	1.93	0.17	61.5	2.12

The neutralizing potentials (NPs) of these samples are consistently higher (72.8 to 116.5 kg $CaCO_3 \cdot t^{-1}$ than their respective APs. The old tailings contain the least NP, while waste rock contains the highest NP. Net Neutralizing Potentials (NNP = NP - AP) are accordingly positive, ranging from 6.9 to 61.5 kg $CaCO_3 \cdot t^{-1}$. The highest NNP was calculated for the waste rock sample, while the lowest NNP was calculated for the old tailings sample

(Table 2). Waste material with NNPs between -20 and +20 kg $CaCO_3 \cdot t^{-1}$ are generally considered in that class of rocks where prediction of their acid generating behaviour is ambiguous, as they are neither clearly acid generating nor non-acid generating. The ratios of AP to NP of rocks for which the test results are uncertain lie between 1:1 and 1:3; all three samples fall in this category (Figure 1). Rocks with a AP:NP ratio less than 1:3 can be classified as non-acid generating.

Figure 1: GOLDCORP Red Lake Mine
Neutralizing Potential (NP) vs Acid Potential (AP)



1.1.3 Humidity Cell Data

The pH of Humidity Cell leachates were consistently greater than pH 7 for the three rock samples tested. Therefore, it is likely that most metals mobilized from the rock samples had precipitated as hydroxides and carbonates. Metal precipitates were likely retained within the humidity cells or retained on filter papers prior to analysis of leachates by ICP. The concentrations of all metals in filtered leachates were all very low. Most of the S in the samples was pyrite (1.39 to 2.11%) and only a small fraction was present as sulphate (0.07 to 0.14%). It should be noted that all humidity cell metals and sulphate concentration data are based on analyses of filtered samples.

Pyrite oxidation occurred in all three humidity cells, suggested from the consistently elevated sulphate concentrations in the leachates. The old tailings sample contained 2.25% S at the outset of the humidity cell test. Based on the cumulative sulphate load ($\sum$ [concentration in assays x leachate volume]), the old tailings sample's sulphate content was diminished from 2.25%, by at least 0.62%, such that no more than 1.63% S remained in the old tailings sample after 273 days of leaching (Table 3). Approximately 28% of the original S content in this sample was leached over 273 days. The S content of the new tailings (1.46%) and the waste rock (1.93%) samples was diminished by only 0.12% and 0.02%, respectively. The final S content of these samples are expected to be 1.34% and 1.91%. The equivalents of 8% and 1%, respectively, of the samples' S content were leached over the 273 days test period.

Table 3: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids and Humidity Cell Summary data.

		OLD TAILINGS			NEW TAILINGS			WASTE ROCK		
	units	Total in Solids	Leached in 273 d (Filtered)	Remaining in Solids	Total in Solids	Leached in 273 d (Filtered)	Remaining in Solids	Total in Solids	Leached in 273 d (Filtered)	Remaining in Solids
S	%	2.25	0.62	1.63	1.46	0.12	1.34	1.93	0.02	1.91
As	$ug.g^{-1}$	1615	6.7	1608	1293	6.7	1286	2286	3.69	2282
Fe	%	7.48	0.0004	7.48	7.35	0.0007	7.35	5.70	0.0001	5.70
Ca	%	2.86	0.72	2.14	2.79	0.14	2.65	3.11	0.04	3.07

More S may have oxidized and leached than can be calculated from the cumulative sulphate load as it can precipitate and remain in the columns for example, gypsum.

While sulphate, one of the two main products of pyrite oxidation, reported to leachates, iron, the other main product, was at very low concentrations or non-detectable in the filtered leachates (Table 3). Iron oxidation in the neutral pH range would be immediately followed by its precipitation as oxides/ hydroxides and or carbonates. These particles will also adhere to rocks in the cells or be captured on the filter papers, given their size as summarized in Figure 2. These processes likely account for the low or non-detectable concentrations of iron in the filtered leachates using filtration size of 0.45um.

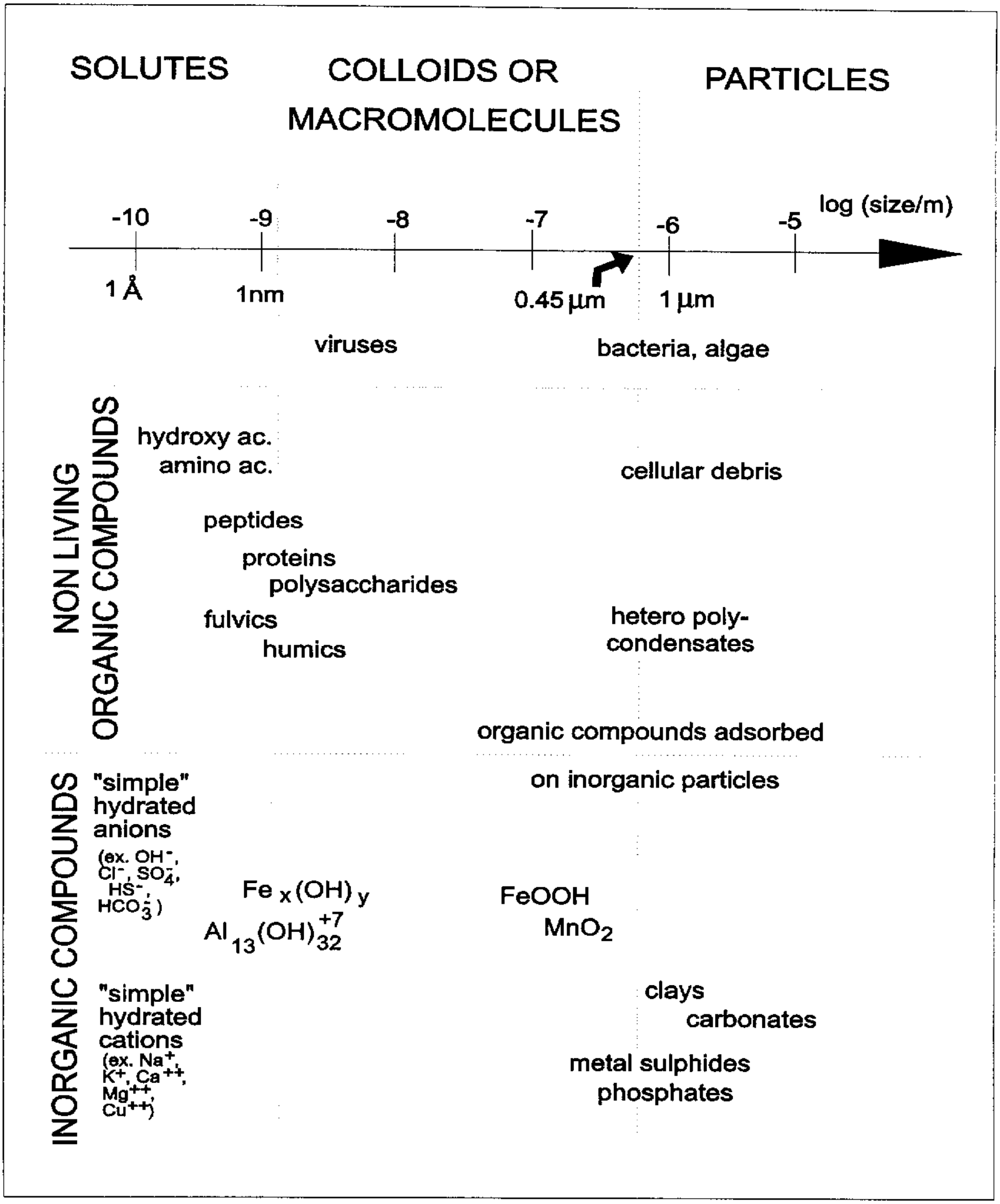


Figure 2: Nature and size domain of the important particles of aquatic systems. (From: Environmental Particles, Buffle J., Leeuwen H.P., 1992)

Very little As reported in filtered leachates from the old tailings, new tailings or waste rock samples. No more than approximately 7 mg of As were present in the total volume of filtered leachate produced in 273 days, compared to 1,293 to 2,286 mg of As originally present in the 1 kg samples.

Although the results are interesting in general, the extensive tests of the three samples do not allow a definite conclusion on the potential for acid generation. It should be stated that the sample selection and the approach was justified, as it is reasonable to assume that the ore body mined by GOLDCORP is of the same mineralogy as that of Campbell Mines, which also did acid base accounting tests. The acid base accounting of the waste materials from Campbell Mines indicated that no acid generation problem exists, and the assumption of similar mineralogy of the two mines is very reasonable. The uncertainty associated with the results was further addressed by this study, using field and historical data from the site.

Boojum Research Ltd. was retained to assess the long term geochemical behaviour of the waste management area and review the background information of the site to facilitate effective long term environmental waste management of the tailings areas.

2.0 METHODOLOGY AND APPROACH

2.1 Distribution and Quantities of Sulphidic Waste Materials

The mining and milling history of the Arthur White Mine, later referred to as the Dickenson Mine and now known as the GOLDCORP Mine, was reconstructed to determine the amount of tailings and their distribution in the waste management area. This information provides estimates of the quantity of tailings during the time of the roaster operation, the location of these low sulphur tailings, and the quantity of post-roaster shut down tailings, along with the locations of these sulphur-containing tailings deposits. These results are summarized in Section 3.1.

2.2 Mineralogy of Waste Materials

In order to make some assessment of the mineralogy, a literature review was carried out, focussing on reports of minerals which can contribute alkalinity when they weather. These minerals can be expected to provide neutralisation of any acid generated, if they weather at the same rate as sulphidic materials oxidize and generate acid. To ascertain the mineral association of the existing waste material, a sequential extraction test was run with tailings material. These samples were subjected to a Sequential Extraction test for gold association, which identifies the mineral fractions (see Appendix for detailed methods). Extraction with acetic acid reports the calcite fraction; extraction with hydrochloric acid (HCl) reports dolomite and iron oxides. Cold nitric acid digestion solubilizes the arsenopyrite and some pyrite and secondary copper minerals. Finally, hot nitric acid extracts all remaining pyrite and sulfides. This information is summarized in Section 3.2 along with other information from the literature.

2.3 Elemental Composition of Waste Materials

A summary of the elemental composition of solid waste materials was prepared for which chemical analysis have been reported. The ore body is very heterogenous with respect to the distribution of sulphidic minerals. The tailings generated since 1948 are the results of various gold extraction processes. Extensive drilling, sampling and analytical work would be required to obtain representative samples for determining ranges of elemental compositions of the waste material. Even with this information at hand, it would be difficult to obtain an assessment of the acid generation potential, given the heterogeneity of the mineralogy. A summary of all reported elemental concentrations in the tailings will provide an adequate inventory of the sulphidic or acid generating mass and those elements which might contribute to neutralisation. These results are presented in Section 3.2.

2.4 Characteristics of Tailings Slurries

2.4.1 Mass Balance Leaching of Tailings

A pragmatic approach to assessing the acid generation potential has been taken by using samples of unoxidized and oxidized tailings material and subjected these to leach tests. Oxidized tailings were considered those which showed no signs of iron deposition, and oxidized tailings where those which did, as shown in Plates 1 and 2, respectively.

Such visually different locations were encountered throughout the tailings deposit in the exposed area of the secondary pond and the old tailings areas during the field trip carried out by Boojum Research in 1997. The samples with which the leach tests were done originated from the primary pond and are depicted in Plate 3.

Plate 1: Unoxidized tailings pit in the Old Tailings pond in the vicinity of OT-B1 (Map 1).

Plate 2: Oxidized tailings pit in the Old Tailings pond in the vicinity of OT-B1 (Map 1).

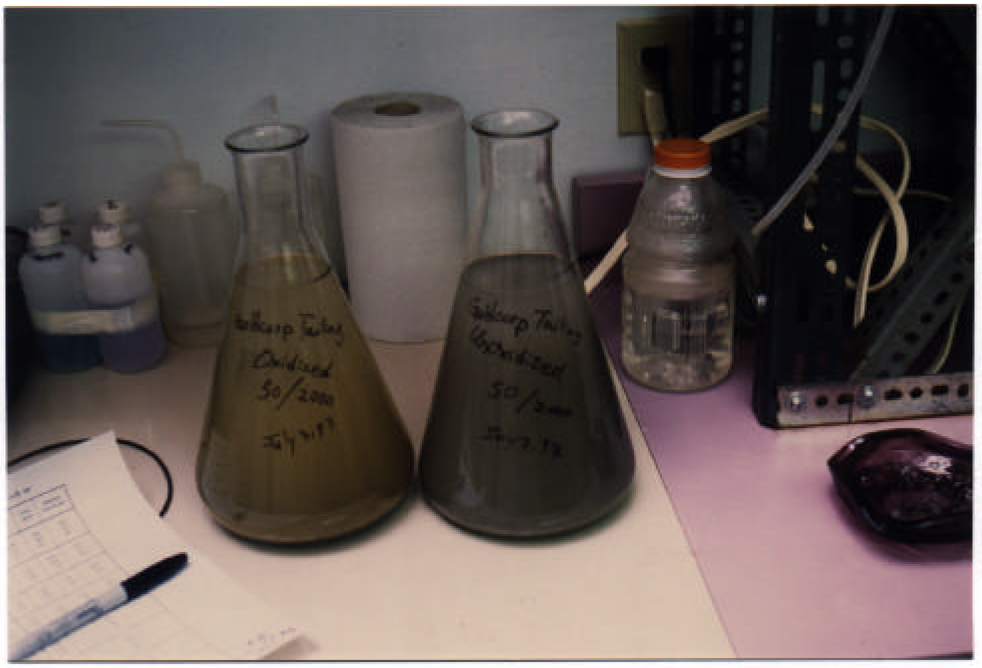


Plate 3: The slurries prepared with 2 L distilled water for mass balance leaching

The detailed methodology for these leaching tests is described below:

Procedure 1: Setting up 50/50, 50/100 and 50/150 experiments : Supernatant was prepared by measuring 50 mL wet volume for each tailings sample and then adding 50 mL, 100 mL and 150 mL distilled water respectively to achieve wet volume to distilled water volume ratios of 1:1, 1:2 and 1:3, corresponding 50 /50, 50/100 and 50 /150. The slurries were stirred for 2 hours on the magnetic stirrer and the pH of the supernatant was measured after letting the sample settle for several hours. These steps are referred to as a decant cycles, of which 22 decant cycles were performed. Essentially the tailings were leached successively as the supernatant obtained from decant cycle 1 was decanted and then a volume of distilled water was added according to the original sample wet volume: distilled water volume ratio. The new slurry was again placed on the magnetic stirrer for an additional 2 hours and again allowed to settle for several hours before measurement of pH. The first 18 decant cycle supernatants were pooled into one sample for measuring pH, conductivity and alkalinity. Supernatant samples were stored separately from decant cycles 19 to 22 to determine the same parameters.

Procedure 2: Setting up 50/2000 experiments: The supernatant was prepared by measuring a 50 mL wet volume for each sample as described in Procedure 1. A volume of 2000 mL of distilled water was added to the 50 mL wet sample and the pH was determined immediately. The slurry was placed on the magnetic stirrer for 2 hours, then allowed to settle for several hours. The supernatant was decanted and the pH, conductivity and alkalinity were measured. The supernatant was filtered through 0.45μm and acidified with nitric acid to pH 1 and submitted for chemical analysis by ICP-25 and fro sulphur to MDS laboratories.

Following decanting of the supernatant, a second volume of 2000 mL distilled water was added to the same tailings sample. The slurry was stirred for 2 hours, then allowed to settle for several hours. The supernatant's pH, conductivity and alkalinity were then measured. The supernatant was filtered and acidified and also submitted for analysis for IAP-25 and sulfur analysis.

2.4.2 Tailings Leaching of Potential Oxidation Products

In order to determine what materials might have weathered in the exposed waste material, 200 mLs of tailings slurries were slurried with 200 mLs of distilled water by stirring, and pH, conductivity, acidity and alkalinity were measured following stirring for 24 hours. The supernatant was decanted and replaced with 200 mLs of new distilled water. This cycle was repeated for 5 cycles. Tailings used in this procedure were collected from tailings areas exposed for different length of time since discharge. The determinations were carried out by GOLDCORP staff on site (Noel Mejia). All leach test results are discussed in Section 3.3.

2.5 Piezometer Data Collected by GOLDCORP

To relate the leach tests to the actual tailings deposits and their weathering characteristics, the chemical analysis of the piezometers which have been sampled by GOLDCORP have been examined. These piezometer waters should best reflect the weathering of elements in the tailings and their movement is determined by the hydrology of the tailings.

This information is used together with the hydrological conditions of the waste management area, described in 1994 by Denis Netherton Engineering. Since that time, approximately 350,000 tonnes of tailings have been added to the waste management area. These results are presented in Section 3.4.

3.0 RESULTS AND DISCUSSION

On June 17th and 18th 1997 a field investigation was carried out by Boojum Research of the Goldcorp tailings area. On all open water bodies on the tailings pH and electrical conductivity were measured. The surface water or ponded water in all areas of the tailings ranged from 7.0 to 8.6 and in the tailings forming the sediment generally the pH was higher than in the overlaying water with pH values as high as 9.1, as for example in the primary pond (Table 4). The conductivity of the water is low when compared to other tailings ponds. The tailings ponds do not show any sings of acid production. Sampling locations are indicated on Map 1.

The pond water was assessed for the presence or absence of algal groups indicating the ecological status of the water (Table 5). Thin algal mats, formed by blue-green algae and by diatoms, generally covered the tailings/sediment surface. When running water was encountered on the tailings generally attached filamentous algae, such as Ulothrix grew in these small streams. The algal populations are typical for alkaline waste waters. The occurrence of these algal groups within a year of cessation of discharge of tailings indicates that recovery of the waste water is very fast. The possibility to utilize biological

Table 4: GOLDCORP INC. Limnological Survey, June 18, 1997

LOCATION	DEPTH	pH	Cond uS/cm	Temp. °C
Primary Pond				
	tailings, dry, higher up	7.0	1121	17.4
	middle, near channel in liquified brown layer, 0.3 m depth, 0.2 m thick			
(post off stand pipe)				
	Surface	8.3	940	15.6
	Bottom (0.6 m)	8.5	935	15.5
near second post off cross-road (metal gauge off road)				
	Surface water	7.8	757	15.2
	dredge sample supernatant	8.5		
Shallow end				
	surface	8.6	934	16.2
	top 10 cm of tailings sediment (1.5 m)	7.9	715	16.1
Dry tailings (wet depression)				
	Tailings detression hole (no oxidized layer)	8.4	980	17.8
	In tailings	9.1	510	16.8
Tailings				
Eckman dredge sample				
	surface water	7.5	730	15.5
New Tailings				
mouth of gabion weir				
	gray layer top 0.08 m			
	brown/reddish layer at 0.2 m			
	gray tailings below			
Channel in Tailings				
	water	8.5	968	19.0
	in tailings	7.7	457	17.5
Abandoned Tailings Near Concentrate Pile				
tailings profile				
	Several layers: oxidized band visible	too dry for analylitical determination		
	Piezometer pumped dry: get fresh sample from bottom			
	Standing water in piezometer	8.6	633	9.0
	After 26 min.	8.7	687	9.9
	0.5 m hole: more oxidized; difficult to get sample	6.9	693	16.5

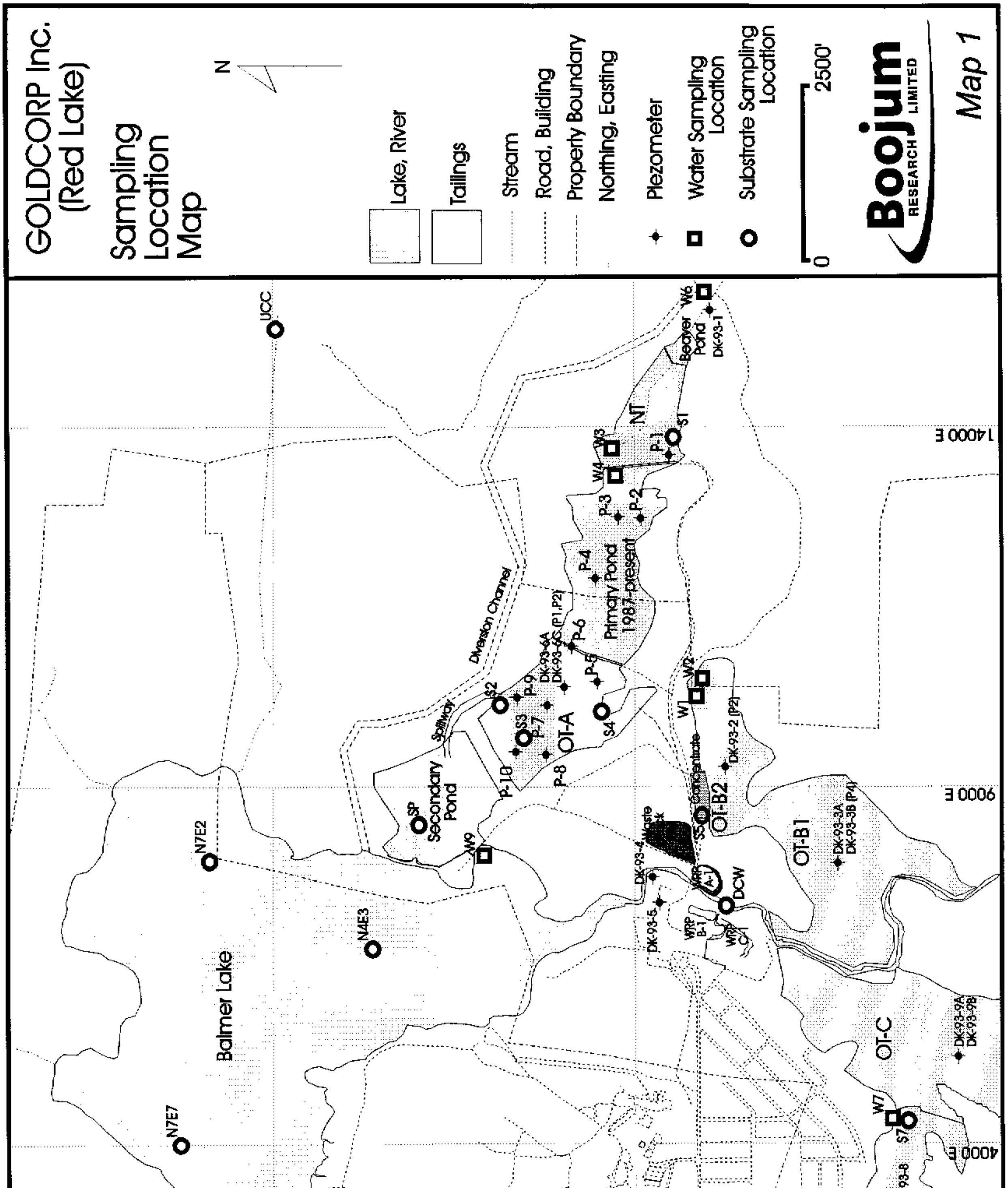

GOLDCORP Inc.
(Red Lake)
Sampling
Location
Map
N
Lake, River
Tailings
Stream
Road, Building
Property Boundary
Northing, Easting
Piezometer
Water Sampling Location
Substrate Sampling Location
0
2500'
Boojum RESEARCH LIMITED
Map 1
Balmer Lake
N7E7
N7E2
N4E3
UCC
Secondary Pond
SP
W9
Spillway
Diversion Channel
P-10
P-8
P-7
S3
S2
P-9
OT-A
S4
P-5
P-6
DK-93-6A
Primary Pond
1987-present
P-4
P-3
P-2
W4
W3
NT
P-1
S1
Beaver Pond
W6
DK-93-1
W1
W2
DK-93-2 (P2)
Concentrate
OT-B2
S5
DK-93-4
DCW
WRP A-1
WRP B-1
DK-93-5
OT-B1
DK-93-3A
DK-93-3B (P4)
OT-C
DK-93-9A
DK-93-9B
W7
S7
OT-D
DK-93-8
W8
S6
14000 E
9000 E
4000 E
10000 N
5000 N
Red Lake

Table 5: GOLDCORP INC. Phytoplankton Samples Descriptions, June 18, 1997

LOCATION		SPECIES	cell density cell.L^{-1}
Primary Pond			
	W3 surface sample		
		Fragillaria sp., single-celled pennate diatom	2.10E+06
		coccoid chrysophyte cysts	2.70E+06
		Dinobryon sociale	1.00E+06
		Dictyospharium sp.	few
		naviculoid diatoms	few
	W3 sediment surface		
		pennate diatoms (no *Eunotia* or cyanobacteria)	
	W4 seepage through dam		
		Ulothrix filaments	abundant
		pennate diatoms	few
		Cosmarium desmids	few
		Oscillatoria (cyanobacteria) filaments, motile	few
		Euglena sp.	few
		chrysophytes	few
Secondary Pond			
	W9 seepage through dam		
		Spirogyra filaments	abundant
		Fragillaria, colonial pennate diatoms	few
		rotifers	few

20 mL were reduced to 1 mL from which 50 µL were counted or inpecrted microscopically

polishing processes exists, should water quality improvements be required. The tailings pond water in 1997 reflect these conditions which would be encountered after one year of shut down.

From the investigation of the surface water and the exposed dry tailings, acid generation did not appear to be evident. The field visit was broadened to gain an overview of decommissioning conditions for the entire site. Water was collected throughout the drainage basin containing the old and new tailings and pH, electrical conductivity and Eh was determined (Table 6). The measurements from the waste management area are compared to background undisturbed water. The Beaver Pond which forms a head of water above the primary tailings pond was considered as background (Plate 4).

Table 6: General Chemistry of Goldcorp Inc. Surface water Samples collected June 17-19, 1997

Sampling Location	Location Description	Sample Type	pH	Cond. uS/cm	Eh mV	Temp °C
W1	Old tailings 'B' Site	seepage	7.03	790	336	20.8
W2	Old tailings 'B' Site	run-off pool	6.91	1181	335	22.8
W3	Primary Pond	pond water	7.20	214	356	22.0
W4	Primary Pond First Berm	seepage	6.69	236	355	22.0
W5	Secondary Pond	discharge	7.17	299	345	22.3
W6	Beaver Pond / Background	seepage	7.48	132	344	22.7
W7*	Abandoned, Revegetated Tailings Behind Delta Apartments	?	8.94	860		25.5
W8*	Abandoned, Revegetated Tailings at Hwy 12	?	7.65	967		24.4

* - collected by Noel Mejia

Plate 4: Beaver Pond Outflow Area with minnow toxicity test.

The values obtained indicate that the differences of the inactive tailings pond water and background undisturbed water with respect to the measured parameters is negligible. The higher conductivity on run-off pool on the old tailings area is very reasonable, as this water was obtained from a pool on the tailings, which had accumulated after rain. In such puddles evaporated salts are a common occurrence. The Eh of the water, the redox potential, is a measure of reactivity of the water. The values suggest that the water on the tailings pond is very similar to value of the undisturbed Beaver Pond water (Table 6).

In several areas of the tailings surface iron precipitates leave the typical yellow - brown stains which suggest that iron has been released. Throughout the tailings area, small pits were dug with a shovel to a depth of about 0.5 m in several locations. Thin layers or bands of iron precipitates were generally found in those areas where tailings beaches were allowed to form during pond operation. However, the occurrence of iron precipitation the result of iron mineral or pyrite oxidation, was not only limited to areas of tailings pond beaches. Similar oxidation pockets were found at random throughout the tailings area. With a summary of the mining milling history of the operation, which was assembled from the literature and with the help of Goldcorp staff an understanding of the distribution of oxidizing material (principally sulphidic material) in the tailings deposit was gained.

3.1 Mine / mill and waste management history

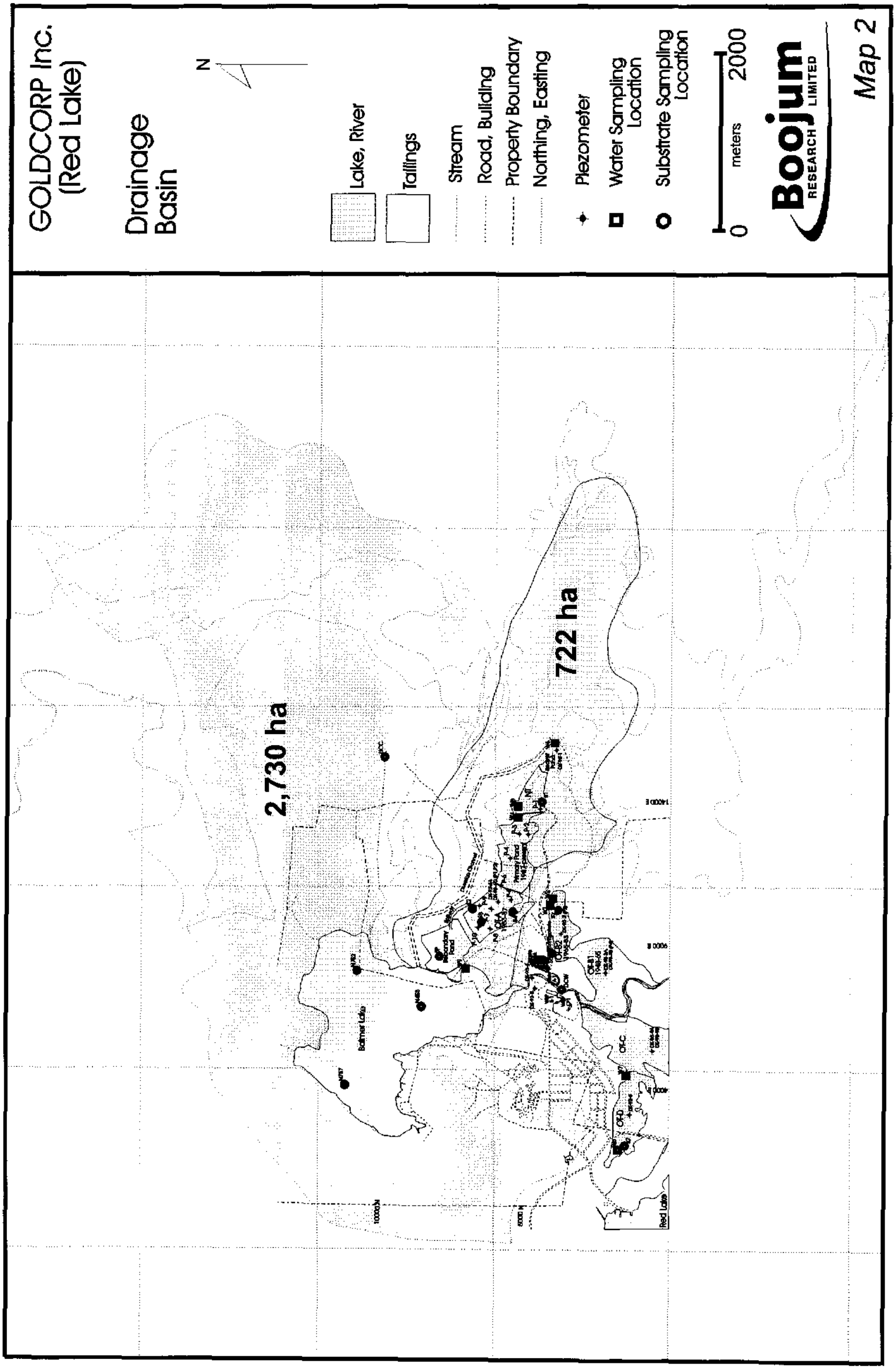
The history of the site is presented below in point form summarizing the key events, relevant to mine waste generation, their location and the mineralogy.

1. **Property**

   Arthur H. White Mine, previous names, Golden Eagle Mine, Red Lake Mine, Renamed in 1982, now owned by Goldcorp.

2. **Location**

   32 patented claims in Balmer township, Red Lake, Ontario, bordered by Campbell Red Lake Mine, Placer Dome. The mine and mill site is located at the mouth of Balmer Lake (Map 2) which covers about 299 ha in a drainage basin of 2,730 ha.


GOLDCORP Inc.
(Red Lake)
Drainage
Basin
N
Lake, River
Tailings
Stream
Road, Building
Property Boundary
Northing, Easting
Piezometer
Water Sampling
Location
Substrate Sampling
Location
0
meters
2000
Boojum
RESEARCH LIMITED
Map 2
2,730 ha
722 ha
Balmer Lake
Red Lake

The drainage basin in which the tailings are located covers an area of about 277 ha, all draining through Balmer Lake past the mine site through Balmer Creek.

3. Metals Recovered

Gold and Silver (Arsenic was recovered from 1971-1980)

4. History of Mine Development and Operation.

1945-1948	Diamond drilling and mine development.
1946-1948	Shaft sunk to 543 ft.
1948 Dec.	Milling operations started at a rate of 125 tpd
1951	Mill capacity increased to 350 tpd; A flotation circuit and a 14 tpd fluid bed roaster were installed. The roaster calcine was leached with the flotation tailings.
1954	Flotation concentrate exceeds roaster capacity. Flow sheet changed to direct cyanidation of whole ore. Milling rates increased to 430 tpd.
1959	Mill expansion to 470 tpd. Installed re-grind for flotation feed. Installation of backfill plant.
1970	Started underground production of Robin Red Lake property.
1980	Roaster shutdown. Sale of sulphide concentrate to smelter.
1981	Mill expansion to 700 tpd, running under utilized
1988	Start stockpiling flotation concentrate.

5. Geology and Mineralogy

The gold deposits occur in The Red Lake greenstone belt. Host rocks are tuffs of intermediate (andesite, diorite) to mafic (basalt) composition. Mineral assemblage overall is actinolite, biotite, muscovite and albite. Subordinate quantities of epidote, quartz, almandine, garnet, pyrrhotite and ilmenite are dispersed throughout the rocks. The area has experienced intense alteration in the form of carbonization (calcium, magnesium, iron), silicification and sericitization. All rocks have an appreciable content of carbonaceous material. Gold occurs in quartz veins in volcanic rocks or in areas of mixed volcaniclastic and chemical sediments. The mineralogy is quite variable in various zones of the mine.

A staff report in 1973 stated that "the normal ankerite has the following composition: Ca CO_3.Mg CO_3. Fe CO_3 ore consists of vein lenses of a cherty-grey quartz with considerable ankerite, mineralized in descending order of pyrite, pyrrhotite, arsenopyrite, sphalerite, magnetite and stibnite, with very minor amounts of chalcopyrite and pentlandite In some areas there is considerable silicification of the wall rock in fine fractures, with which are associated the higher gold values. Sulphide make up about 3% of the mill feed."

Other references indicate 4% or 5% sulphide in the ore at other times. When Knopp et al (1989) tested a flotation column in the plant in 1988 they reported that the flotation feed averaged 2.16% sulphur and 0.52% arsenic, indicating 3.63% and 1.13% pyrite (or pyrite equivalent) and arsenopyrite, respectively, for a total of 4.76%. From the information reviewed, no definitive estimate of the mineralogical composition of typical mill feed can be made There is considerable variation in the mineralogy in various ore zones, particularly with regard to relative abundance of sulphide and carbonates, the key minerals relating to the acid generation process.

6. Mining Method

Underground, cut and fill, methods were used from 1959 to 1980, followed by blasthole stopping in 1981. The latter resulted in high dilution, and led to the reintroduction of cut and fill in 1982. Primarily cut and fill underground mining, with sand backfill produced from the mill since 1959. According to the Canadian Mines Handbook (1996-97) in earlier operations some ore was mined by shrinkage stopping. In 1996 cut and fill stopes produced 80% and longhole stopes 20% of the ore.

7. Mine waste.

There are three waste rock piles (A, B, and C) located south west of the mill site, above Balmer Creek. The following are estimated volume for each Pile A 82,392 cu yds, pile B 34,270 cu yds and pile C contains 15,000 cu yds. Most mine waste would come from underground development work and are probably low in sulphur. No obvious seepage path were noted on the surface from the waste rock piles.

In addition there are a number of small waste piles located south of the Mill area close to the control weir and roads around the site are generally built with mine waste rock. Most of the mine waste has originated from underground development work and is probably low in sulphur. Drainage either runs off the waste rock piles or would reach Balmer Creek as ground water. No surface run-off channels were noted.

8. Mine water usage

In 1993 the volume of water pumped from the mine is estimated around 40 gal/min. The majority of this water originated from backfilling. The mine site surface drainage is directed toward a pump house and to a containment basin.

9. Milling process

1948	Process consisted of crushing, grinding, cyanidation and gold recovery by zinc precipitation (Merrill-Crowe process).
1951	Mill capacity increased to 350 TPD. Added gravity concentration (jigs) and amalgamation of jig concentrate. A flotation circuit was added to recover gold-bearing sulphide prior to cyanidation. Flotation concentrates were roasted in a new fluid bed roaster and roaster calcine was cyanided along with the flotation tailings. The jig concentrate was amalgamated. Roaster off-gases were vented to the atmosphere.
1954	Milling rate was increased to 430 TPD. Circuit was changed to cyanide the ground ore before flotation. The flotation concentrate was roasted and the calcine returned to the leach circuit.
1954-1963	Milling capacity increased to approximately 525 TPD.
1959	A grinding mill was added to re-grind repulped cyanidation tailings prior to flotation resulting in a finer flotation feed, from the previous 58% - 200 mesh to 92% - 200 mesh. A backfill plant was installed to provide sands for mine backfill.
1971-1980	Mill didn't operate at capacity.
1980	Roasting plant shut down.
1981-1983	Mill was expanded to 800 tpd by the addition of a new grinding mill

and other changes.

1984 An arsenopyrite - gold concentrate was recovered by flotation and shipped to Boliden in Sweden.

1988 Shipments to Boliden ended. The arsenopyrite concentrate continued to be recovered, but was stored in a separate area close to the main tailings.

The fact that the old tailings were generated to be roasted it can be reasonably assumed that those tailings are low in sulphur as sulphide flotation works very effectively. The old tailings have been discharged to areas designated at OT-D, OT-C and most of the tailings in area OT- B1 (Map 1). In the tailings part designated as OT-B2 where discharge of tailings ceased in 1983, some higher sulphur tailings may be present. In this area the concentrate was stored since 1988, after shipment to Boliden ceased in 1984.

Since 1983 tailings were discharged into an area between the Secondary and Primary pond. The Secondary pond borders Balmer Lake separated by a dam with a decant structure. In principle, the secondary pond was expected to function as a polishing pond for the active tailings area to degrade cyanide and settling of suspended solids. Water management during the seasonal run-off events was difficult and a freshwater diversion channel was recently installed to divert water around the three main tailings ponds and the secondary pond. Since 1987 tailings are discharged into the upper portion of the Primary pond.

Using the milling rates tailings tonnages are estimated for the different areas. The areas are indicated on Map 3 and the tailings distribution by area is presented in Table 7. The detailed production data are reported in the Appendix A. Relating the tonnage of tailings to the time the roaster was operating up to 1980, producing low sulphur tailings it is suggested, that about 2.9 million tonnes of tailings are waste material which do not present a concern for acid generation. Those appear to be distributed over a relatively large area of about 120 ha. The sulphide flotation process works relatively effective which would reduce the pyrite content of these old tailings to very low. Therefore acid generation from

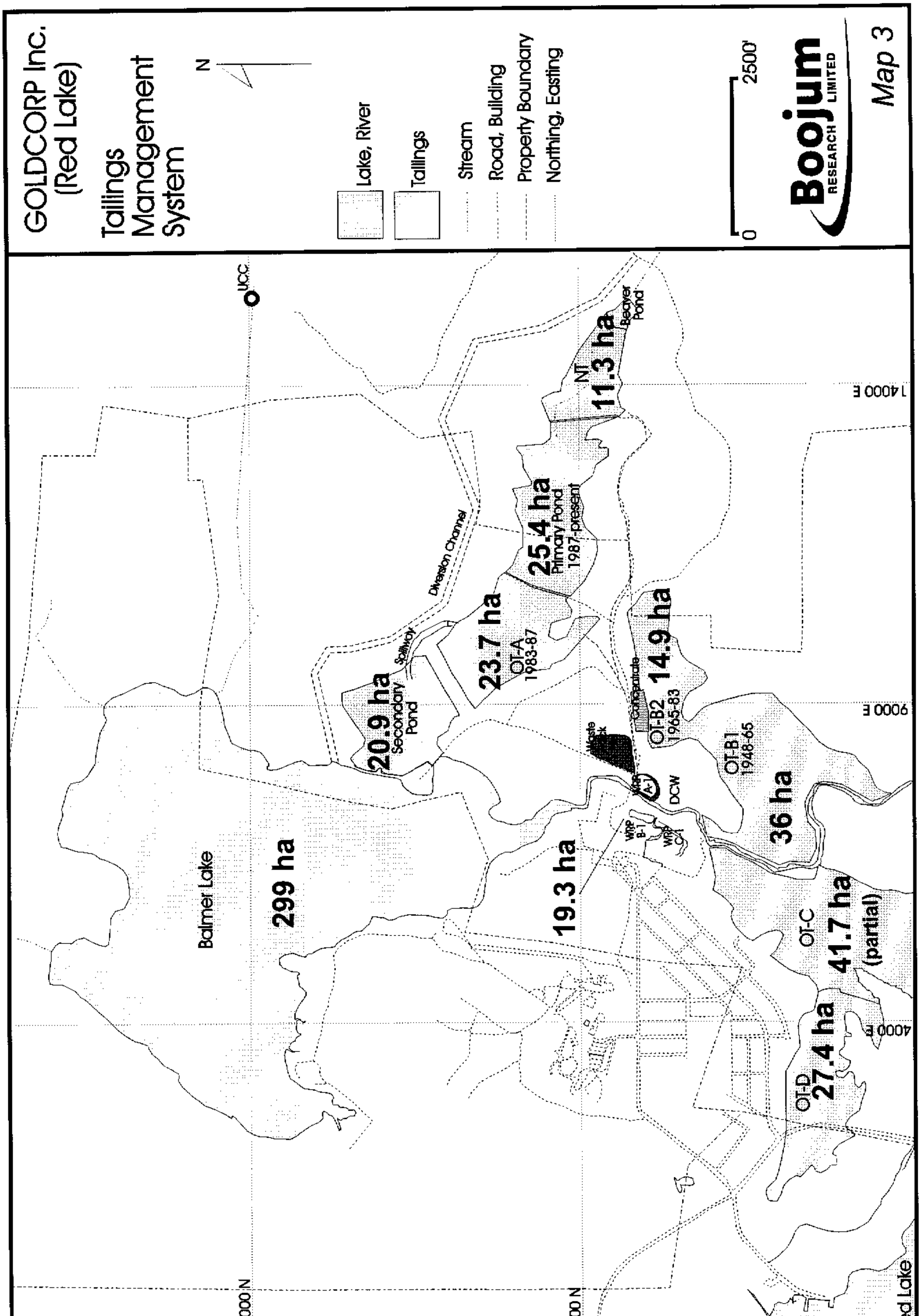

GOLDCORP Inc.
(Red Lake)
Tailings
Management
System
N
Lake, River
Tailings
Stream
Road, Building
Property Boundary
Northing, Easting
0
2500'
Boojum
RESEARCH LIMITED
Map 3
UCC
Beaver Pond
NT
11.3 ha
14000 E
Diversion Channel
25.4 ha
Primary Pond
1987-present
23.7 ha
OT-A
1983-87
Spillway
20.9 ha
Secondary Pond
Concentrate
14.9 ha
OT-B2
1965-83
9000 E
OT-B1
1948-65
36 ha
DCW
Balmer Lake
299 ha
19.3 ha
OT-C
41.7 ha
(partial)
4000 E
OT-D
27.4 ha
10000 N
5000 N
Red Lake

Table 7: Tailings Distribution by Area, 1949-1996

Years	Location	Area ha	Tailings Deposited tonnes	t/ha
1949-1965	OT-B1, OT-C, OT-D	105.1	1,867,105	17,765
1966-1982	OT-B2	14.9	1,401,665	94,071
1983-1987	OT-A	23.7	593,833	25,056
1987-1996	Primary Pond	36.7	1,301,696	35,469
1951-1980	Pre-Roaster	120.0	2,972,936	24,774
1981-1996	Post-Roaster	64.0	2,102,503	32,852

this pre-roaster material is not expected at all. This if there is a potential to generate acid, then this can only be found in tailings deposited post roaster shut down 1980. Up to 1996 post roaster tailings production was about 2,1 million tonnes of tailings. Those have been deposited over a smaller area then pre- roaster time.

Considering the tailings deposition in more detail with respect to recovery of the surface and coverage by volunteer vegetation some interesting observations can be made. Table 7 presents for the different areas where tailings have been deposited, a tonnage per area. The area OT-B2 apparently has the highest t/ha ratio and is not covered by vegetation. However it appeared that the areas OT-D, OT-C and OT-B1 is relatively well covered with vegetation. Although a detailed assessment of the vegetation cover was not carried out during the field trip in 1997, the factors relating to the natural revegetation should be defined and utilized for the decommissioning approach for the site. In the area OT-B2 at the edges of the pond, where run off occurs, vegetation has rooted to a limited degree suggesting that run-off water plays a role in vegetation cover.

3.2 Mineralogy and chemical composition of the tailings

From the geology of the ore, the pyrite concentrations vary. Published plant operations data are used to arrive at sulphur concentrations which have likely reported to the tailings. Based on Knopp and Contini (1988) 1.67% sulphur reports to the tailings (Table 8a) and using the data of Wan Weert et. al in 1988 it could be expected that the tailings contain 1.36% S (Table 8b). These results would indicate a relatively consistent pyrite content, which does not reflect the large variability of the ore mineralisation, where concentrations up to 5% sulphide are reported. It appears that the milling process evens out the occasional high sulphide ores. The most recent data are provided by Goldcorp for the years 1991 to 1994 and produce an average of 1.45% S (Table 8c).

The most important mineral estimate, in addition to the sulphur concentration, are concentrations of neutralizing minerals and their quantities, such as carbonates. To estimate those, data from Kerrick et al (1981) who did extensive analysis of mineral samples were utilized (detailed data are copied in the Appendix A, Tables 12a to 12c). Although several assumption have to me made using the Loss On Ignition values to obtain an estimate of regularity of volatile fraction in the rock, ratios of Ca oxide and Mg oxides have been calculated. In these samples the calcium oxide fraction ranges from 2.6% to as high as 10.3% and for magnesium oxide the range is 1.5% to 5.0%. Using ratio of L.O.I to iron oxide a fairly consistent ratio is obtained ranging from 0.1 to 0.8. For the ratio of L.O.I over S the range is less consistent with values between 1.1 to 30.8. These differences may, suggest, that there are non- sulphide iron minerals in the ore (Table 13, Appendix A).

A list of minerals reported in the geology for the Goldcorp ore body is given in Table 9. The reported minerals do suggest, that iron containing minerals are present. However with the exception of ankerite, there is no other carbonate mineral reported and the weatherability of these minerals is not well known.

Table 8a: Production Sample Assays

	Assays oz/t,%				Distributions		
	Wt%	Au	As	S	Au	As	S
Flotation Feed	100.0	0.059	0.56	2.56	100.0	100.0	100.0
Cleaner Concentrate	2.7	0.816	6.53	34.80	37.0	32.3	40.5
Final Tailings	97.3	0.038	0.39	1.57	63.0	67.7	59.5

Table 8b: Metallurgical Results

	Flotation Feed			C1 Conc.			Final Tails		
Date	Au oz/t	S %	As %	Au oz/t	S %	As %	Au oz/t	S %	As %
April, 20	.037	2.06	.35	.72	35.7	6.6	.024	1.55	.23
April, 25	.026	1.60	.33	.39	29.2	6.0	.018	1.18	.21
April, 26	.029	1.90	.40	.41	38.1	6.3	.017	1.26	.24
April, 27	.032	2.00	.40	.48	31.8	7.0	.022	1.38	.27
April, 28	.031	2.01	.44	.48	26.8	6.5	.020	1.42	.26
Average	.031	1.91	.38	.50	26.0	6.5	.020	1.36	.24

Table 8c: Tailings grades: Arsenic and Sulphur

Date		% Arsenic	% Sulphur
1994	January	0.26	1.52
	February	0.21	1.29
	March	0.30	1.57
	April	0.46	1.96
	May	0.23	1.63
	June	0.17	1.45
	July	0.27	1.55
	August	0.36	1.57
	September	0.33	1.78
	October	0.30	1.57
1993	January	0.24	1.50
	July	0.18	1.19
	October	0.24	1.22
	November	0.24	1.36
	December	0.24	1.22
1992	January	0.27	1.47
	February	0.29	1.58
	March	0.36	1.70
1991	June	0.27	1.39
	July	0.24	1.20
	August	0.24	1.20
	September	0.26	1.24
	October	0.26	1.40
	November	0.26	1.21
	December	0.27	1.43
	Minimum	0.17	1.19
	Maximum	0.46	1.96
	Average	0.27	1.45

Table 9: Mineral Composition

Actinolite:	$Ca_2(Mg,Fe)_5(OH)_2(SiO)_2$
Albite:	$NaAlSi_3O_3$
Almandine:	$Fe_3Al_2(SiO4)_3$
Andradite:	$CaFe_2^{+3}Si_3O_{12}$
Ankerite:	$2CaCO_3 \cdot MgCO_3 \cdot FeCO_3$
Arsenopyrite:	FeAs
Biotite:	$K(Mg,Fe)_3(Al,Fe)Si_3O_{10}(OH,F)_2$
Chalcopyrite:	$CuFeS_2$
Chlorite:	$4H_2O \cdot 5MgO \cdot Al_2O_3 \cdot 3SiO_2$
Epidote:	$H_2O \cdot 4CaO \cdot 3(Al,Fe)_2O_3(SiO_2)_6$
Grossularite:	$Ca_3Al_2Si_3O_{12}$
Hydrogrossularite:	$Ca_3Al_2Si_3O_8(SiO_4)_{1-m}(OH)_{4m}$
Ilmenite:	$FeTiO_3$
Magnetite:	Fe_3O_4
Muscovite:	$KAl_2(AlSi_3O_{10})(OH)_2$
Pentlandite:	$(FeNi)_9S_8$
Pyrite:	FeS_2
Pyrope:	$Mg_3Al_2Si_3O_{12}$
Pyrrhotite:	Fe_7S_8
Quartz:	SiO_2
Spessartite:	$Mn_3Al_2Si_3O_{12}$
Sphalerite:	ZnS
Stibnite:	Sb_2S_3
Uvarovite:	$Ca_3Cr_2Si_3O_{12}$

In general the acid base accounting tests rest of the following principle. From the geological and mineralogical record it is evident that carbonate minerals are present in the Goldcorp tailings. The neutralizing capacity of the rock will depend on its weatherability. It is assumed that the acid-base accounting of the three Goldcorp samples was performed according to generally accepted procedures, where acid-generating potential (AP) is calculated as follows:

$$AP = \%sulphur \times 31.25$$

Both AP and neutralizing potential (NP) are expressed as kg $CaCO_3$ per tonne of sample. The neutralization potential (NP) of the rock is calculated based on the mass of sample, and volume and normality of the acid added.

These procedures assume that the neutralization potential of the rock is mainly derived from carbonate and hydroxide minerals which dissolve during the test. However, the NP value reflects only the immediate alkalinity released within the time span of this test, in contrast with the AP value, which is based on the total sulphur content. Both these values are unrelated to the rate at which the acidity might be generated in the tailings or the waste rock.

The approach to acid-base accounting uses an AP value representative of acidity generated in the long term, combined with an NP value estimated from a short term test measuring that alkalinity immediately available. Using these values, the Net Neutralizing Potential (NNP) is calculated as follows:

$$NNP = NP - AP$$

Using this approach, it is concluded that the waste rock material will be acid generating when the acid-generating potential (AP) exceeds the neutralizing potential (NP), and the net neutralizing potential (NPP) value is negative. Although there is currently no other approach available to determine to NPP, since this approach does not incorporate an estimate of the acid generation rate predictions based on the NNP can be sometimes problematic. From a chemical point of view, the validity of the NP evaluation procedure may be debatable, particularly when geological information concerning the waste material indicates that carbonate minerals are essentially absent. Although this is not the case for Goldcorp tailings, in addition to carbonate other minerals can be present which might possess acid-consuming capacities. A brief discussion on the acid consuming capacities of minerals is given below.

Water acidity as measured by titration with sodium hydroxide addition, consists of both proton ions and metal ions (Fe, Al, Cu, Mn, Zn, etc.). When minerals dissolve in acidic water, the acid radicals will form weak acid and consume proton ions, as follows:

$$MeS + H^{+} = Me^{+2} + HS^{-}, \qquad MeS + 2H^{+} = Me^{+2} + H_2S$$
$$MeCO_3 + H^{+} = Me^{+2} + HCO_3^{-}, \qquad MeS + 2H^{+} = Me^{+2} + H_2CO_3$$
$$MeSiO_3 + H^{+} = Me^{+2} + HSiO_3^{-}, \qquad MeS + 2H^{+} = Me^{+2} + H_2SiO_3$$

Therefore, upon dissolution, all minerals of sulphide, carbonate and silicate can consume proton ions and increase the pH of water. In the sequential extraction carried out on the Goldcorp tailings material, the first step using acetic acid the pH increase noted in the extracts reflects this consumption of protons (Appendix A, Table 11). This could be considered as the weathering step. When metal oxides dissolve into water, they too increase the pH of water, as follows:

$$Fe_2O_3 + 6H^{+} = 2Fe^{+3} + 3H_2O$$

However, minerals, dissolving in water and through this process consuming protons (neutralizing water H^{+}), also increase the metal concentrations in the water. If the minerals are those listed previously the acidity of the solution remains unaltered, as in the first extraction step with acidic acid. The major difference with respect to the water quality, modified by mineral dissolution, is that the acidity has changed from proton acidity to metals acidity.

However, if dissolution of minerals has resulted in increased concentrations of metals such as Ca, Mg, K and Na, then the proton (H+) acidity of the water has truly decreased. Therefore, the ability of a mineral to provide neutralization potential, to counteract the acid-generating material, might be derived from minerals which contribute K and Na, as well as Ca and Mg, during dissolution.

In Table 9 the minerals which are reported to occur in the Goldcorp ore are listed, but can not be considered as a complete list. From this list, those minerals which, through weathering, might contribute to acid consumption, are

Dolomite type minerals:

$$CaMg(CO_3)_2$$

Chlorite silicates of general formula:

$$A_{5\text{-}6}Z_4O_{10}(OH)_8,\ A=Al,Fe^{+2},Fe^{+3},Li,Mg,Mn^{+2},Ni;\ Z=Al,B,Fe^{+2},Si$$

Biotite:

$$K(Mg,Fe^{+2})_3(Al,Fe^{+3})Si_3O_{10}(OH,F)_2$$

and minerals from the *Mica* group, a group of mica-clay minerals of general formula of:

$$(H_3O)(Al,Mg,Fe)_2(Si,Al)_4O_{10}[(OH)_2,H_2O]$$

In addition some alkalinity can be expected from weathering of Feldspars and silicates of general formula:

$$XZ_4O_8,X=Ba,Ca,K,Na,NH_4,Sr;\ Z=Al,B,Si$$

and Montmorillonite:

$$(Na,Ca)_{0.33}(Al,Mg)_2Si_4O_{10}(OH)_2 * nH_2O.$$

The partial list of minerals presented in Table 9 suggest that minerals are present which have a minerals' chemical formulations producing potential neutralizing capacity, and they do frequently co-occur with the major acid generating metals Al and Fe. The minerals listed above could contribute to acid consumption, but their relative dominance in the different geological formations is not known. The overall effect of their dissolution during the acid generation process is, therefore, difficult to estimate.

In order to obtain some data which would produce some indication on the mineral association in the tailings, sequential extractions were carried out, a procedure described by C.L Brierely (1995) as an indirect chemical method to estimate mineral composition of a sample. Generally it is used on ore samples, to determine the gold association. The detailed methodology is given in the Appendix A in Table 5.

The results from these tests are presented in Table 10. The weight loss reported from this test indicates, that in the tailings material, there is a reasonably large fraction of the sample associated with Calcite, Dolomite and iron oxides reporting as high as 21% in the Old tailings which are pre-roaster shut down and the new tailings report about 16% associated with dolomite or iron oxide fraction. Based on the percentages of the last two extraction steps, which are taken to be iron rich minerals, this constitutes in this test an average of about 9% of iron rich minerals. Of the mineral fractions which were extracted in this sequential extraction a total of about 43% of the weight was extracted. This extractable fractions appears reasonable as Kerrich et al (1981) report up to 60% of Silica oxide. Of the extractable fraction (43%) generally 29% could be interpreted as potentially neutralizing minerals and 21% of acid generating minerals (Table 10). These considerations suggest, that the tailings have a significant neutralisation capacity.

Assuming that the old tailings collected close to Highway 125 represent tailings which can be expected to have been deposited between 1948 and 1965 the surface grab sample s used in the test would have weathered on the surface for at least 32 years. The differences in the percentages of the mineral associations with respect to the time of exposure between the old tailings exposed for 32 years and the new tailings maximum exposed to weathering for 2 years is minimal. In addition, the sample which was collected in the primary pond under water, was not different to that exposed to rain and air in the same pond.

Table 10: Summary of GOLDCORP Sequential Extraction Data: Weight Loss with Extractions

Area Location	NT S1 New Tailings 1987-present	OT-A S2 Primary Pond Exposed Tailings	OT-A S3 Primary Pond Below Water Sampling Stn	OT-A S4 Old Tailings 1983-1987	OT-D S6 Abandoned Tailings @ Hwy 125	OT-D S7 Old Tailings 1965-83
A: Calcite	6.0%	7.7%	7.6%	8.7%	8.7%	4.7%
B: Dolomite, Iron Oxides	16.7%	17.9%	18.4%	16.9%	21.2%	15.4%
C: Arsenopyrite,pyrite,2° Cu minerals	8.8%	11.8%	11.9%	10.2%	6.9%	9.6%
D: Pyrite, Sulphides	8.2%	7.3%	6.5%	7.0%	8.4%	9.3%
Sum	39.8%	44.6%	44.4%	42.8%	45.2%	38.9%

These results do suggest, that the mineral associations in the tailings have not been altered with exposure to weathering for 32 years. It could also suggest, that as the oxidizing products are formed, they are neutralized and the ratio has not changed in 32 years of exposure. Although this interpretation is only an indirect indication of weathering ie. the long term stability of the material, other considerations can be derived from the results. The extraction steps where supplemented with pH measurements and determination of acidity in the original sample, in the extracts and in the washing water. All extraction steps reported increases in pH after 6 h of rolling (Appendix A, Table 11). This indicates that the material has remaining neutralizing capacity after the exposure to weathering. This approach to testing the alkalinity generation potential appears to be useful. Further development of experimental protocol using different strength of acidic acid, will lead to more selective release of neutralizing potential. The acidities reported with the experiments are very high, as full strength acidic acid (17.5 N) is used in this standard test. The main fact derived from these data is, that even after exposure to weathering for various periods of time, neutralizing potential remains in the tailings.

The composition of the tailings with respect to the elements which might contribute to alkalinity release and acidity generation can also be used. All published chemical analysis are used to arrive at average concentrations, along with minium and maximum concentrations (Table 11). The detailed analyses are given in the Tables 2a to 2g in Appendix A.

The average concentrations of the major elements in the waste management area are very similar, considering three groups of samples. First an average was calculated for all solid chemical analysis from 36 samples as the first group (all samples), Second the average are calculated form samples which are reported as sediment samples collected in Balmer Creek and Balmer Lake and third those which are tailings or other mine related materials. This is not surprising to find, that all three groups are very similar since during the early operations, both Balmer Lake and Balmer Creek likely received some tailings. However, if these elemental concentrations are compared to those materials which had been used for the acid base accounting test (Table 1), both the total S and Fe concentrations of these samples are above the average concentrations which can be found in solids material associated with the wastes. It is therefore argued, that the net acid production value from these tests were higher than would be expected from a larger number of samples.

Table 11 also summarized the minium and maximum concentration reported in the data. The range of concentrations for each element covers an order of magnitude, reflecting the variability reported for the ore body and from the geology. Thus further acid base accounting tests, would produce increased data set, with the a large variability as seen in the elemental composition. It would contribute little to increasing the certainty in relation to the acid generation question. Given this variability an pragmatic approach was used, collecting tailings from surfaces exposed to weathering for different length of time to determine the weathered products which would be transported by rain, simulated with distilled water. Such leaching tests will produce data, which indicate what has weathered to date, after 10 years.

Table 11: Minimum, Maximum, Average (in $\mu g/g$) and N for solid samples

ALL SAMPLES				
Element	Minimum	Average	Maximum	N
Al	3,400	30,456	51,000	36
Ca	6,000	30,973	50,200	15
Fe	8,100	63,644	110,000	36
K	270	4,675	9,100	15
Mg	3,800	20,080	30,000	15
Mn	200	1,258	2,000	36
S	2,400	11,500	24,600	15

BALMER CREEK AND BALMER LAKE SOLIDS				
Element	Minimum	Average	Maximum	N
Al	15,000	31,640	47,000	25
Ca	10,000	29,429	38,000	7
Fe	15,000	62,040	93,000	25
K	1,500	4,257	6,400	7
Mg	5,200	22,743	30,000	7
Mn	300	1,295	2,000	25
S	3,000	8,129	18,000	7

NOT BALMER CREEK AND BALMER LAKE SOLIDS				
Element	Minimum	Average	Maximum	N
Al	3,400	27,764	51,000	11
Ca	6,000	32,325	50,200	8
Fe	8,100	67,291	110,000	11
K	270	5,040	9,100	8
Mg	3,800	17,750	29,000	8
Mn	200	1,175	1,600	11
S	2,400	14,450	24,600	8

3.3 Weathering products in tailings leached with distilled water

In Table 12 the results of distilled water rinses, stirring for 24 h between each rinse, are reported for tailings collected from 1 year old surfaces (S1, fresh tailings) and S6 and S7 collected from the oldest tailings area. The sampling locations are given in Map 1. In this set of tests a sample from the concentrate pile was included, to arrive at a quantity of acid generated exposed to weathering, representing worst case scenario for pyritic material exposed. The concentrate had been stored since 1988, when shipment to Boliden stopped. However, the length of time the sample which was collected was exposed could not be determined. Therefore a conservative assumption that the exposure was about 5 years.

The pH of the fresh tailings slurries was the highest with 8.7 in the first rinse, and had dropped by the 5th rinse about one pH unit. In comparison the old abandoned tailings stayed at the same pH throughout all 5 rinse cycles. The alkalinity in all the rinse waters was low with the highest value measured for tailings collected below the water. The conductivity in all the samples decreased with rinsing cycles to various degrees. The low alkalinity of the rinse water either suggests, that either no oxidation has taken place or as it is generated, it is neutralized, and all alkalinity is used up.

The acid generation in the material would not be noted as it will be neutralized and in the pore water, which is washed out with the tailings rinses. No net alkalinity can be observed, as reflected in the low alkalinities in the rinse cycles (Table 12). A second approach was taken to assess the oxidation products which have formed in the tailings, by leaching both apparent oxidized tailings and unoxidized tailings for a longer period of time.

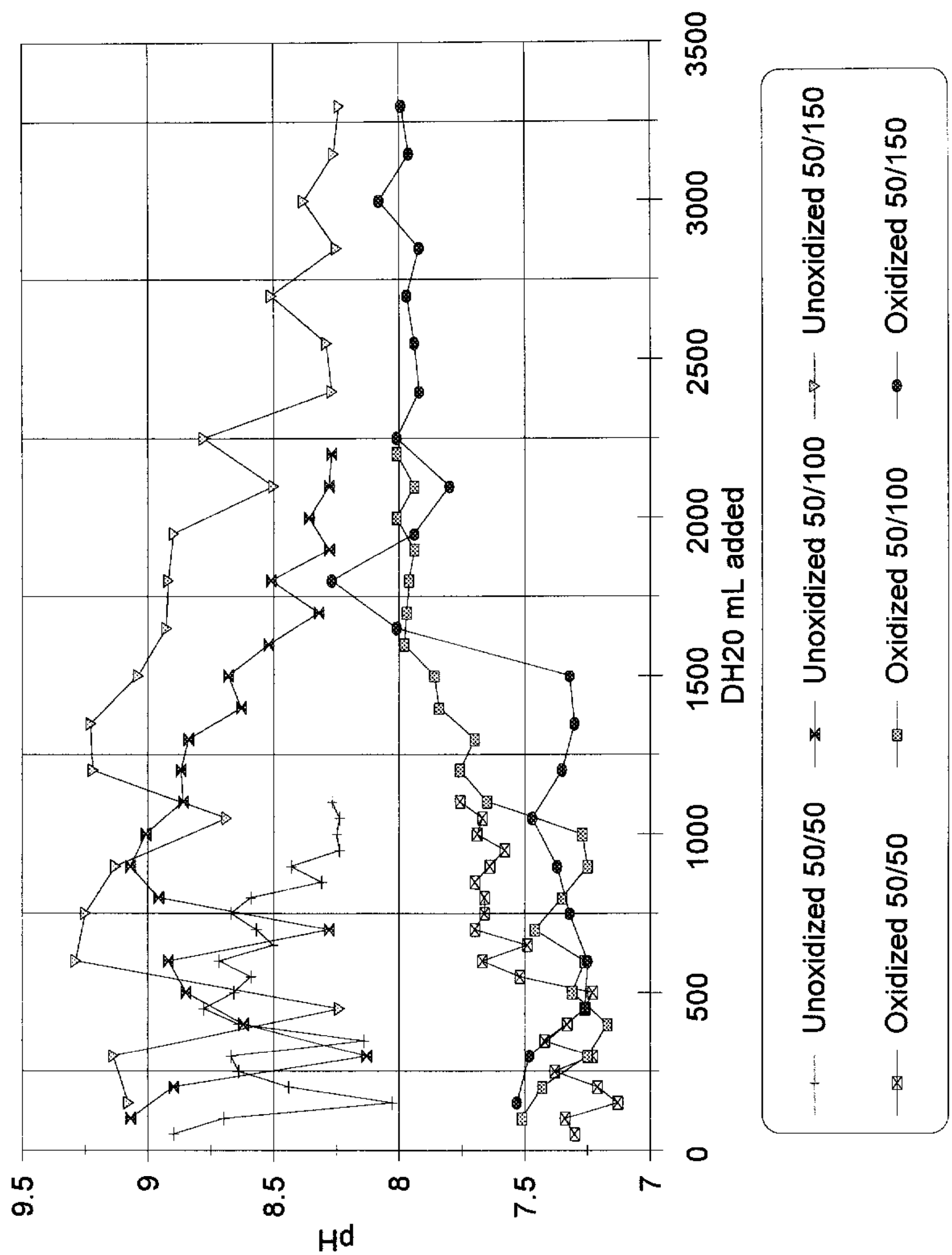
In Figure 3 the pH values of oxidized and unoxidized tailings slurries consisting of different quantities of water leach water are presented. The first run of the rinse cycle experiment (Table 12) may have not allowed for sufficient time between rinsing, and the solution may have been to saturated. Slurries of ratios of tailings to distilled water 1:1, 1: 2 and 1:3 tailings to distilled water were exchanged for a period of 17 days. Comparing all three runs,

Table 12: General Chemistry of Goldcorp Inc. Solid Slurry Samples. Solids Collected June 17-?, 1997.

Date	Samp. Locat.	Location Description	Sample Type	pH						Conductivity uS/cm					
			Day	set-up	1	2	3	4	5	set-up	1	2	3	4	5
17-Jun-97	S1	Primary Pond	fresh tailings	9.36	8.78	7.60	7.56	8.20	8.17	420	578	526	397	259	275
?	S2	Primary Pond @ Gabion Weir	brown layer below fresh		6.90	7.10	7.04	6.83	7.31		2,580	2,690	2,620	2,130	1,099
?	S3	Primary Pond	tailings below water		7.10	7.20	7.04	7.64	7.54		571	491	435	388	428
17-Jun-97	S4	Old Tailings after 2° Berm	tailings	7.18	6.95	7.20	6.90	6.81	7.18	3,490	3,740	3,080	2,690	2,420	2,380
17-Jun-97	S5	*Flotation Concentrate Pile*	*concentrate*	3.49	2.55	4.02	3.95	4.00	4.06	16,070	23,700	6,270	3,890	2,990	2,590
	S6	Abandoned Re-vagetated Tailings @ Hwy 125			7.00	7.00	6.98	7.06	6.80		832	606	576	508	601
	S7	Abandoned Re-vegetated Tailings behind the Delta Apts.			7.30	7.18	7.81	8.07	7.97		625	466	823	509	370

Date	Samp. Locat.	Location Description	Sample Type	Alkalinity mg/L						Acidity mg/L					
			Day	set-up	1	2	3	4	5	set-up	1	2	3	4	5
17-Jun-97	S1	Primary Pond	fresh tailings		65	80	75	70	65						
?	S2	Primary Pond @ Gabion Weir	brown layer below fresh		60	70	50	55	75						
?	S3	Primary Pond	tailings below water		130	130	135	130	135						
17-Jun-97	S4	Old Tailings after 2° Berm	tailings		50	50	45	50	45						
17-Jun-97	S5	Flotation Concentrate Pile	concentrate							8,620	8,700	2,150	950	800	500
	S6	Abandoned Re-vagetated Tailings @ Hwy 125			190	250	275	325	380						
	S7	Abandoned Re-vegetated Tailings behind the Delta Apts.			210	360	430	200	250						

Fig. 3: Dilution pH vs DH2O mL added
June 20 - July 7, 1997



bringing the total rinse volume up to 1 L, to 2 L and finally to 3.25 L indicates the following. The pH of the rinse water from the unoxidized tailings sample generally higher than the rinse water from the oxidized sample. Both samples, when diluted with 2.5 L of distilled water, approach pH 8. The pH of distilled water used was 6.74. It does suggest, in both material weathering products producing OH ions, or consuming H ions are present. These samples came form the surface of the OT-A area they would have been exposed to weathering since 1987, ie 10 years.

In Table 13 the results of chemical analysis are presented as the % of the elements in the tailings solids which have reported to the extracts i.e. distilled water in 0.45 µm filtered leach solution. In the 10 year exposure about 2% of the Ca is solubilized for the unoxidized material and 14% of in the oxidized material. As it is expected, iron at these high pH values does not mobilize. Na does leach very well and is likely a remnant of the milling reagent. The percentage S which is reporting to the distilled water is higher in the oxidized tailings material than in the unoxidized tailings material, suggesting that acid generation is taking place. Deriving a rate of generation of S products from these%, would indicate, that 36% of the S contained in the tailings has been oxidized and reports in the tailings as gypsum. If it is assumed that this oxidation rate will continue, then in about 30 years all of the S products which have reacted. If the processes, which have lead to the dissolution of the sulphate products remain the same as in the past 10 years, then it is reasonable to expect, given also the results of the sequential fractionation of the tailings, problems normally associated with acid generating waste materials are not expected for the Goldcorp tailings. In addition although the tailings are oxidizing in some areas of the tailings pond, the leach data from the short term leach cycles (Table 12) indicate, that a water cover produces higher alkalinity, then when the tailings are exposed. Thus clearly through effective tailings management which aims to maintain a water cover, acid generation rates will be further reduced and possibly totally eliminated.

Table 13: Percentage of Solids Tailings extracted with distilled water

	Unoxidized	Oxidized
Al	0.06%	0.01%
As	2.18%	0.18%
Cd	1.75%	0.32%
Ca	2.19%	13.94%
Cu	0.23%	0.22%
Fe	0.01%	0.01%
Mg	0.42%	8.29%
Mn	0.04%	2.07%
K	2.18%	5.04%
Na	29.73%	45.61%
Sr	3.13%	7.18%
S	2.34%	36.27%
Zn	0.77%	0.95%

3.4 The chemistry of the piezometer water

As a final check on the acid generation behaviour of the tailings, the water samples collected from the piezometers were assembled and analysed. The detailed data are presented in Appendix A. For the data interpretation, only those piezometers are selected which have data available covering a time period between 1994, up to 1997. Several piezometers were destroyed or have not been maintained. The piezometers were installed in fall, 1993.

In Figure 4, the pH values of the selected piezometers are plotted for the available time period. In the figure, the stratigraphic location where the piezometer has been installed is indicated. The piezometers installed in the oldest tailings, discharged during roaster operations in area OT-B2 (1965 - 1983) are DK93-2, and in OTB-1 (1948 - 1965), DK93-4A and DK 93-3B. DK-93-6A is installed in the newer tailings deposit areas OT-A, along with DK-93-6C, completed in the tailings or below the tailings. Similar to the results of the distilled water extractions, the piezometer waters can be separated into two groups, one with higher pH and one with lower pH. The lower pH in pore water is found in the oldest tailings and the higher pH values are reported for the piezometer water installed in the younger tailings deposit. This is, in principle, consistent with the previous findings from the leach tests. The pH values are, however, in the same range than reported for a background piezometer, installed in the vicinity of the Beaver Pond for which pH values of 7.5 - 7.9 are reported in DK-93-1 (Table 9a and 9 b, Appendix A).

In Figure 5, the conductivity data are presented and these indicate large differences between the piezometers and their location. Essentially, three piezometer waters suggest elevated electrical conductivities above a background value around 350 umhos/cm (DK-93-1), ranging from 1000 umhos/cm (DK-93-6A) to about 4500 umhos/cm for DK-93-6B. To obtain a better understanding of the chemistry of the elevated piezometer porewater, total iron concentrations are plotted in Figure 6. The pH and conductivity data were producing consistent values for the time period over which they have been measured, but

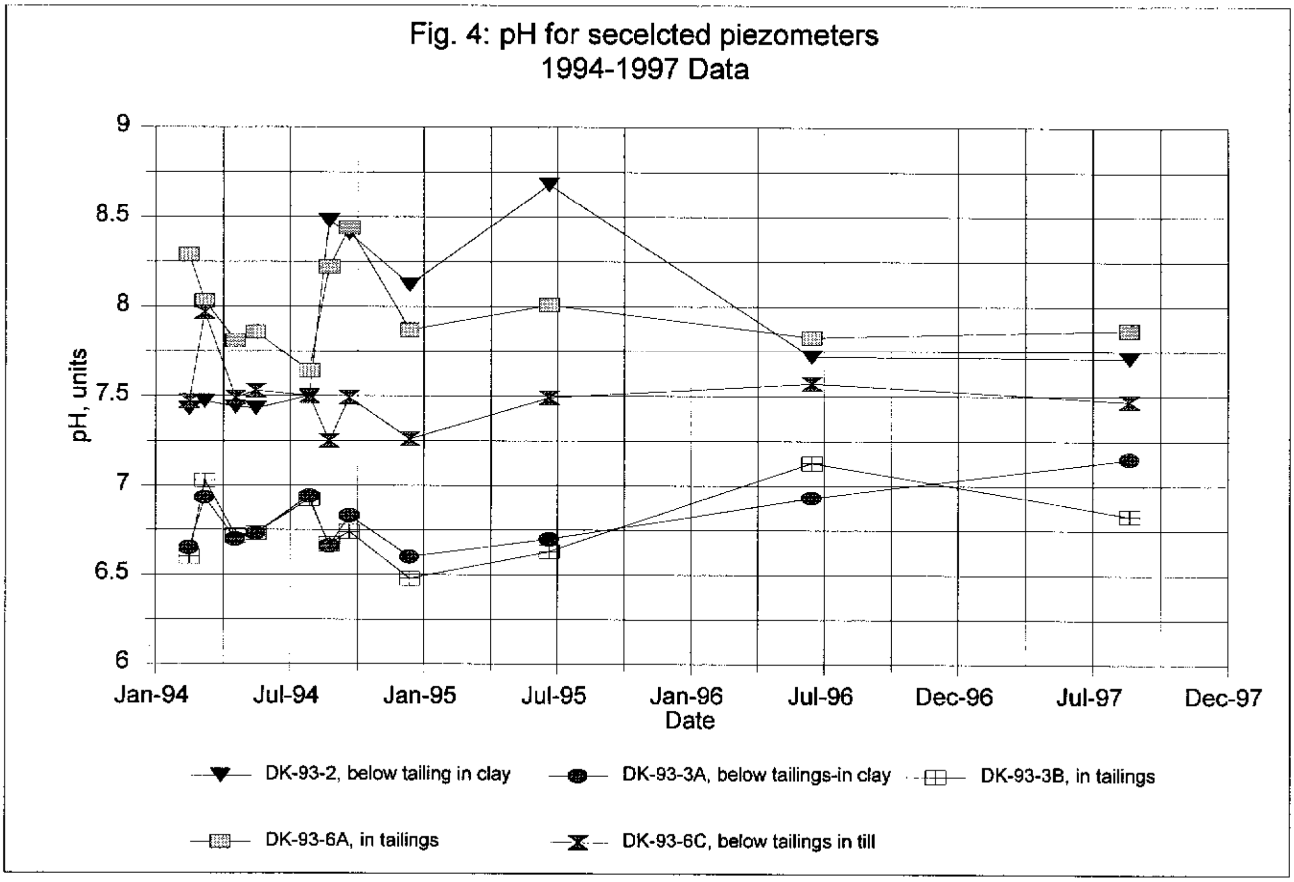


Fig. 4: pH for secelcted piezometers
1994-1997 Data
pH, units
9
8.5
8
7.5
7
6.5
6
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

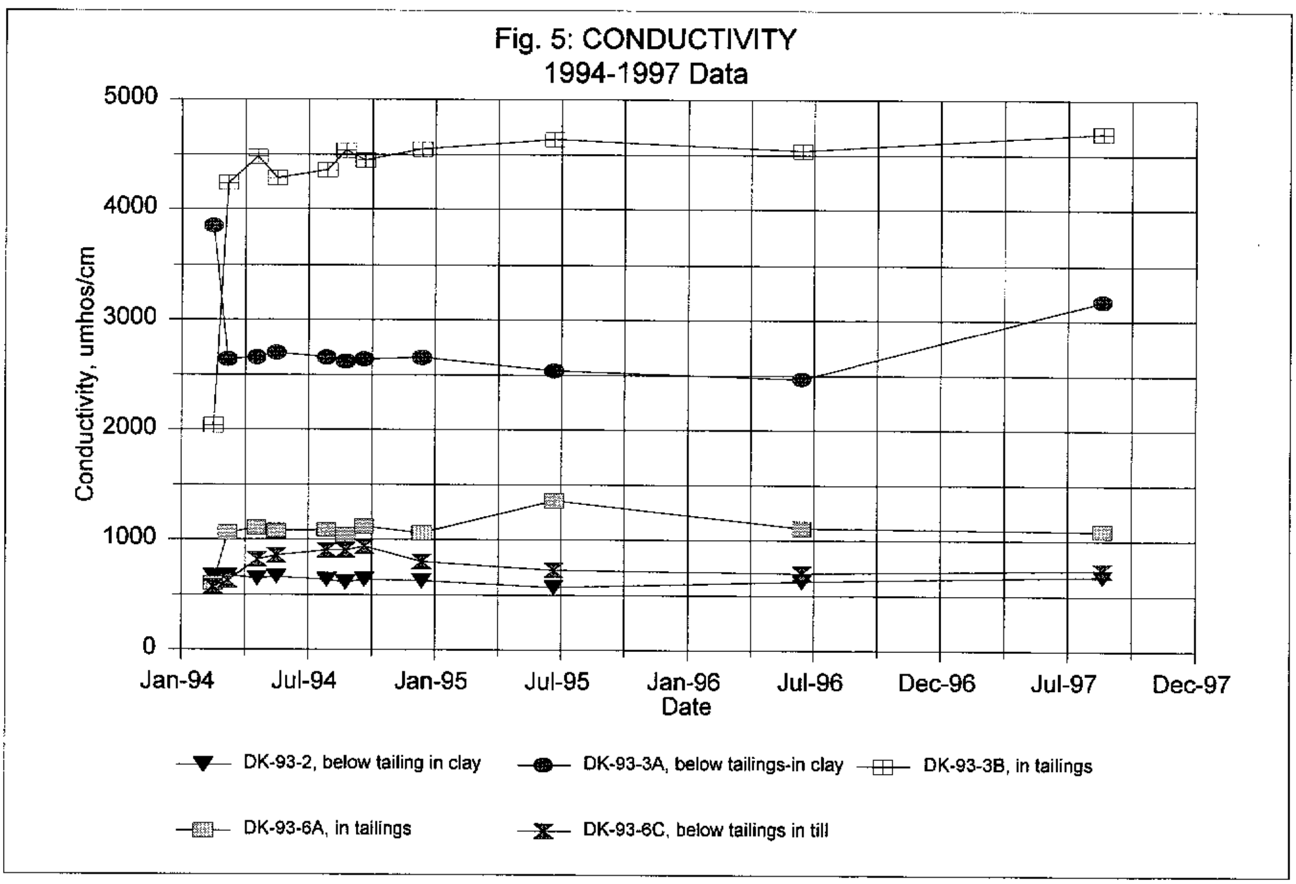


Fig. 5: CONDUCTIVITY
1994-1997 Data
Conductivity, umhos/cm
5000
4000
3000
2000
1000
0
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

this was not the case for the total iron concentrations. Large increases were noted in the samples collected in 1996 and 1997. Iron is an important element, since its mobility will result through hydrolyzation or oxidation in lower pH values. The iron in the tailings pore water is likely reduced, and if mobilized, can result in problems. The only data set which is presently available is that of the distilled water extracts (Table 4, Appendix A) to determine potential iron problems. The pH and the alkalinity was monitored covering a period of about 16 days, extensive iron dissolution and hydrolysis was not noted, as the pH values of the cumulated water did not change, nor did the measured alkalinity.

In Table 14, a comparison of the total iron concentrations to those reported as dissolved (0.45 µm filtered) are presented. The discrepancies between these values are evident and suggest that iron is precipitating after sampling. The sampling procedure was described as bailing of the piezometer for a standardized period of time, followed by sampling of water and filtration in the laboratory through the filter paper. The filtered sample is acidified in the laboratory and shipped with the sample for total determinations to the analytical laboratory. For the total metal determinations, generally the sample would be acid digested, and hence the randomly high totals are reported in comparison to the dissolved iron concentration. The large increases and/or discrepancies in the iron data are therefore associated with the sampling and bailing technique. In addition, such discrepancies are noted in background piezometer (Table 9a and 9 b Appendix A).

Although sampling methodology can explain the differences in relation to the increases in 1996 and 1997, it does not account for the large difference of the two elevated piezometers. In Figure 7, the sulphate concentrations are presented. Again, the piezometer in the older tailings (DK-93-3A,B), finished both below the tailings (A) and in the tailings (B) reports high sulphate concentrations. A very brief assessment was carried out of the flow direction, using the water levels from 1994 (Table 10 in Appendix A) presented in Map 4. From the flow directions, the arrows were derived between each piezometer, and it is conceivable that indeed a ground water plume has originated from the concentrate pile which is noted in piezometer DK-93-3A and DK-93-3B.

Table 14: Comparison of total and dissolved iron for selected piezometers

DK-93-2											
DATE	16-Feb-94	09-Mar-94	20-Apr-94	18-May-94	29-Jul-94	25-Aug-94	21-Sep-94	13-Dec-94	22-Jun-95	15-Jun-96	21-Aug-97
Fe-diss	0.183	0.203	0.308	0.325	0.276	0.172	0.118	0.390	0.934	0.33	0.48
Fe-tot	122	252	11.8	25.8	9.1	0.629	7.23			242	3310

DK-93-3A											
DATE	17-Feb-94	09-Mar-94	20-Apr-94	18-May-94	25-Jul-94	25-Aug-94	20-Sep-94	13-Dec-94	22-Jun-95	15-Jun-96	25-Aug-97
Fe-diss	0.289	0.283	0.298	0.224	0.201	0.537	0.464	1.310	0.888	0.03	4.94
Fe-tot	77.200	137.000	7.580	0.309	0.707	0.599	0.822			169	266

DK-93-3B											
DATE	17-Feb-94	09-Mar-94	20-Apr-94	18-May-94	25-Jul-94	25-Aug-94	20-Sep-94	13-Dec-94	22-Jun-95	15-Jun-96	25-Aug-97
Fe-diss	7.310	0.908	9.250	7.050	7.120	9.380	10.500	22.300	7.91	0.04	3.94
Fe-tot	11.300	15.000	14.000	12.400	17.300	12.800	14.200			100	44.2

DK-93-6A											
DATE	17-Feb-94	11-Mar-94	20-Apr-94	18-May-94	27-Jul-94	26-Aug-94	23-Sep-94	15-Dec-94	23-Jun-95	15-Jun-96	26-Aug-97
Fe-diss	0.109	0.109	0.039	1.340	0.030	0.030	0.030	0.030	0.046	0.03	0.12
Fe-tot	197.00	152.00	2.26	2.63	1.31	10.20	1.27			708	760

DK-93-6B											
DATE	17-Feb-94	11-Mar-94	20-Apr-94	18-May-94	27-Jul-94	26-Aug-94	23-Sep-94	15-Dec-94	23-Jun-95	15-Jun-96	26-Aug-97
Fe-diss	0.059	0.043	0.041	0.797	0.030	1.020	0.191	0.164	0.074	0.03	0.18
Fe-tot	52.20	73.70	0.60	2.15	0.96	3.56	3.91			188	157

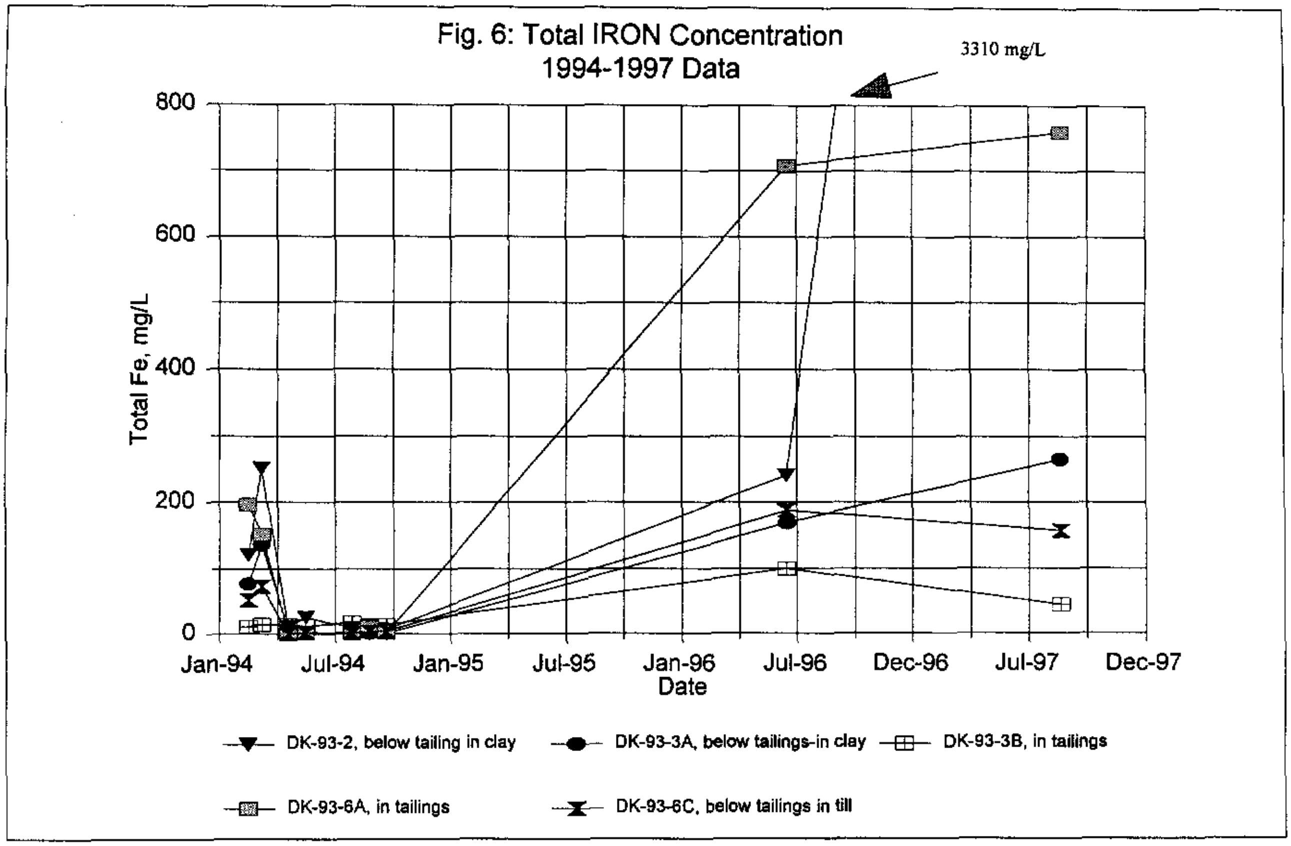


Fig. 6: Total IRON Concentration
1994-1997 Data
3310 mg/L
800
600
400
200
0
Total Fe, mg/L
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

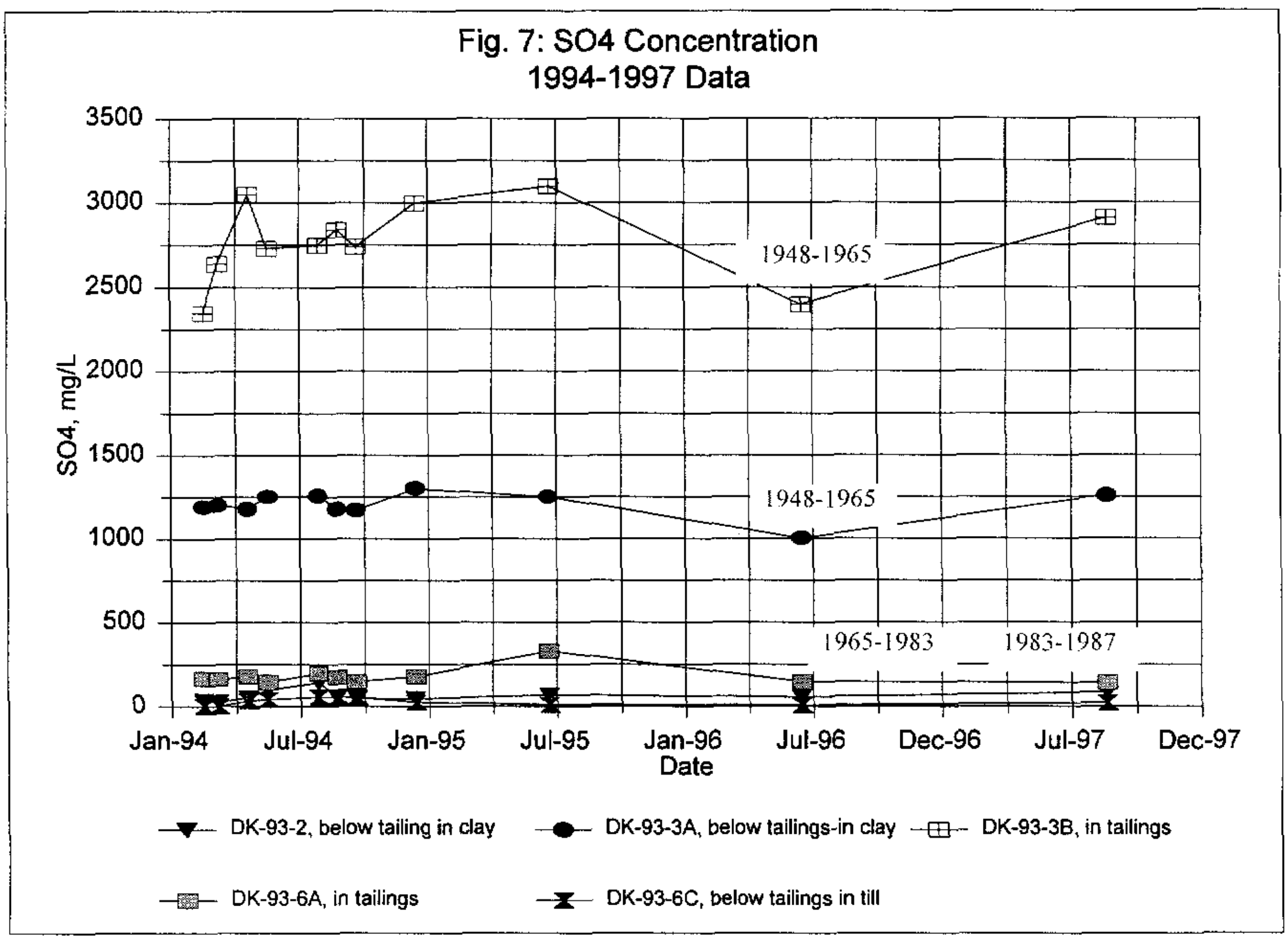
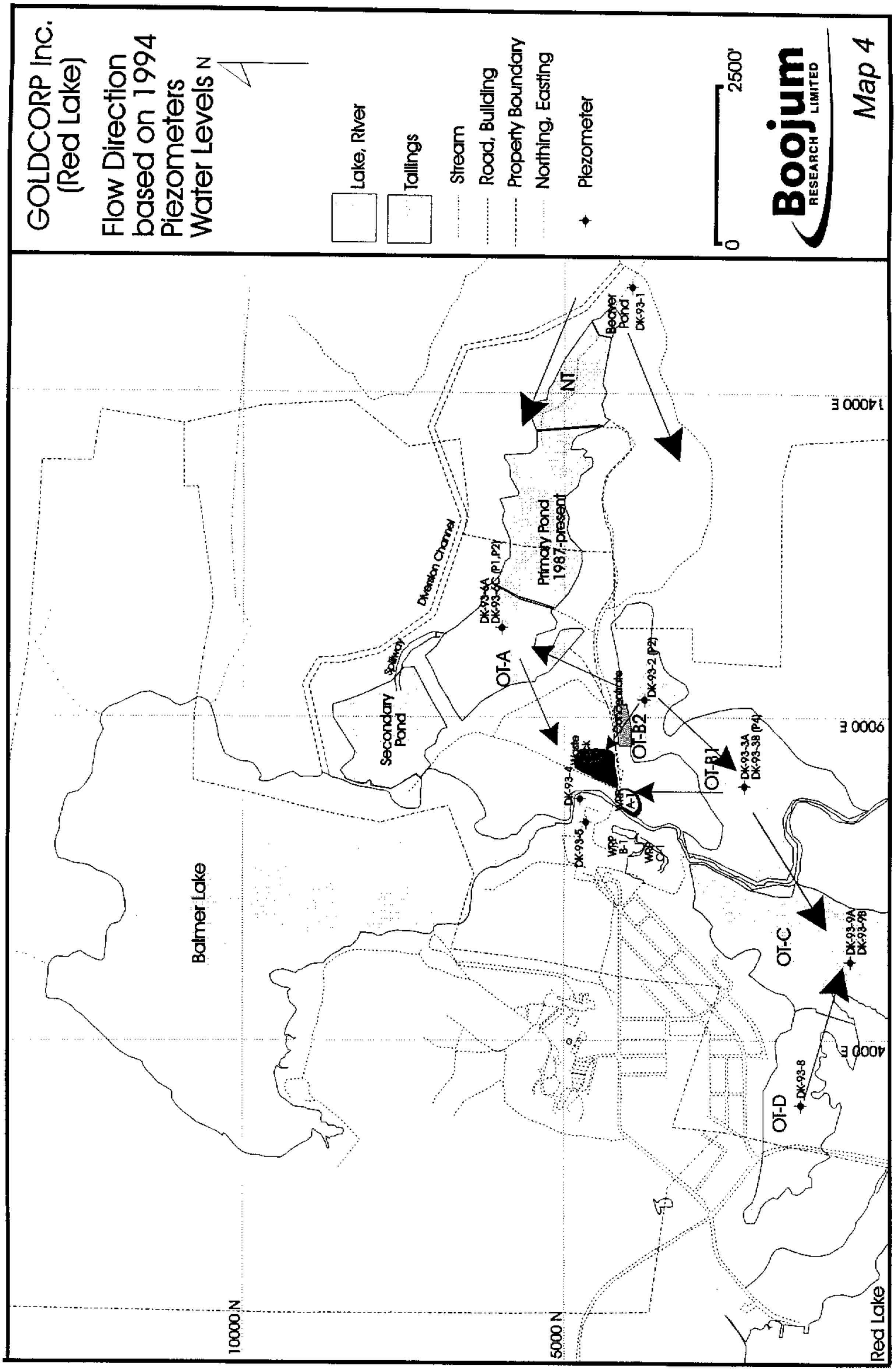


Fig. 7: SO4 Concentration
1994-1997 Data
3500
3000
2500
2000
1500
1000
500
0
SO4, mg/L
1948-1965
1948-1965
1965-1983
1983-1987
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till


GOLDCORP Inc.
(Red Lake)
Flow Direction
based on 1994
Piezometers
Water Levels
N
Lake, River
Tailings
Stream
Road, Building
Property Boundary
Northing, Easting
Piezometer
0
2500'
Map 4
Balmer Lake
Secondary Pond
Spillway
Diversion Channel
Primary Pond
1987-present
NT
Beaver Pond
DK-93-1
OT-A
OT-B1
OT-B2
OT-C
OT-D
DK-93-2 (P2)
DK-93-3A
DK-93-3B (P4)
DK-93-4
DK-93-5
DK-93-8
DK-93-9A
DK-93-9B
14000 E
9000 E
4000 E
10000 N
5000 N
Red Lake

To ascertain further what processes could be taking place leading to the elevated concentrations in piezometer DK-93-3 A and B, piezometer waters from all piezometers were further evaluated. In Appendix B, the dissolved and total concentrations for the elements Na (Figure 31- 35), K (16 - 20), Mg (21 to 25) and Mn (26 - 30) are plotted for the same piezometers where a long term record exists. These elements are generally present in very similar concentrations ranges up the 1996. The consistently large differences between dissolved and total concentrations in the later years are attributed to the sampling/bailing technique. If differences are evident, then they are present in water sampled early in the year. The differences are very large during spring sampling for iron (Figures 1 to 5, Appendix B). The processes which lead to these differences are not understood and may require some clarification.

When relating the piezometer water quality to that of the leach experiment on which a mass balance of oxidation products was carried out, the total concentrations were used. In Figure 8, the calcium concentrations are plotted. The same piezometer pair DK-93-3A and 3B are elevated in Ca, and in Mg (Figure 9), compared to the other piezometers. It should be noted that, for example, piezometer DK-93-6A does not show elevated concentrations. This piezometer was also completed in the tailings, although in the younger tailings deposit, but in pre-roaster tailings. Manganese (Figure 10) and potassium (Figure 11), two elements implicated in alkalinity generation when weathered, are both slightly higher than in the other piezometers. The concentrations of sodium (Figure 12) do not reflect the same pattern, since this element is interfered with by the cyanidation chemicals, and cannot be related to the weathering process.

The piezometer data relate to those of the distilled water leach tests, indicating that oxidation does take place and neutralisation products are solubilized in the tailings pore water, in specific locations, but not throughout the entire tailings deposit.

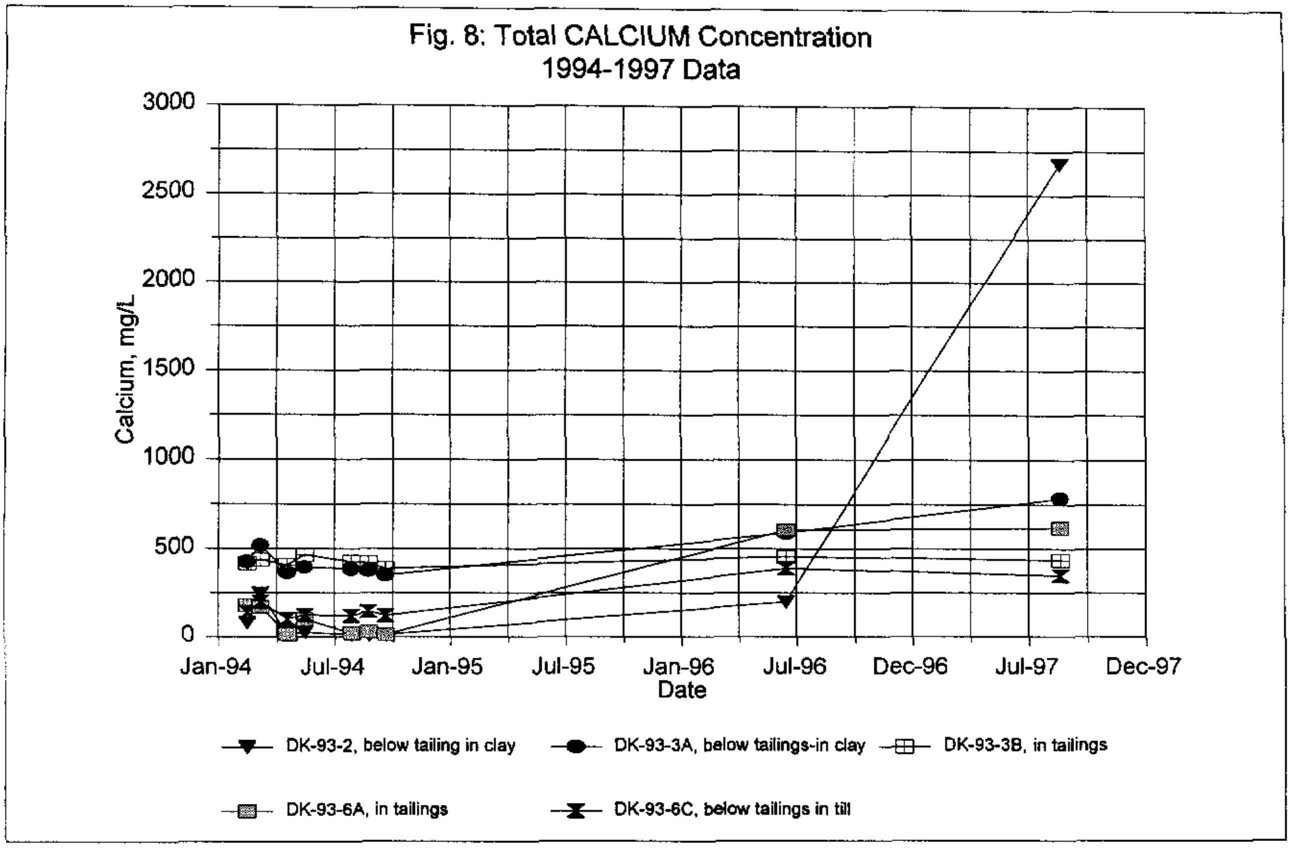


Fig. 8: Total CALCIUM Concentration
1994-1997 Data
3000
2500
2000
1500
1000
500
0
Calcium, mg/L
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

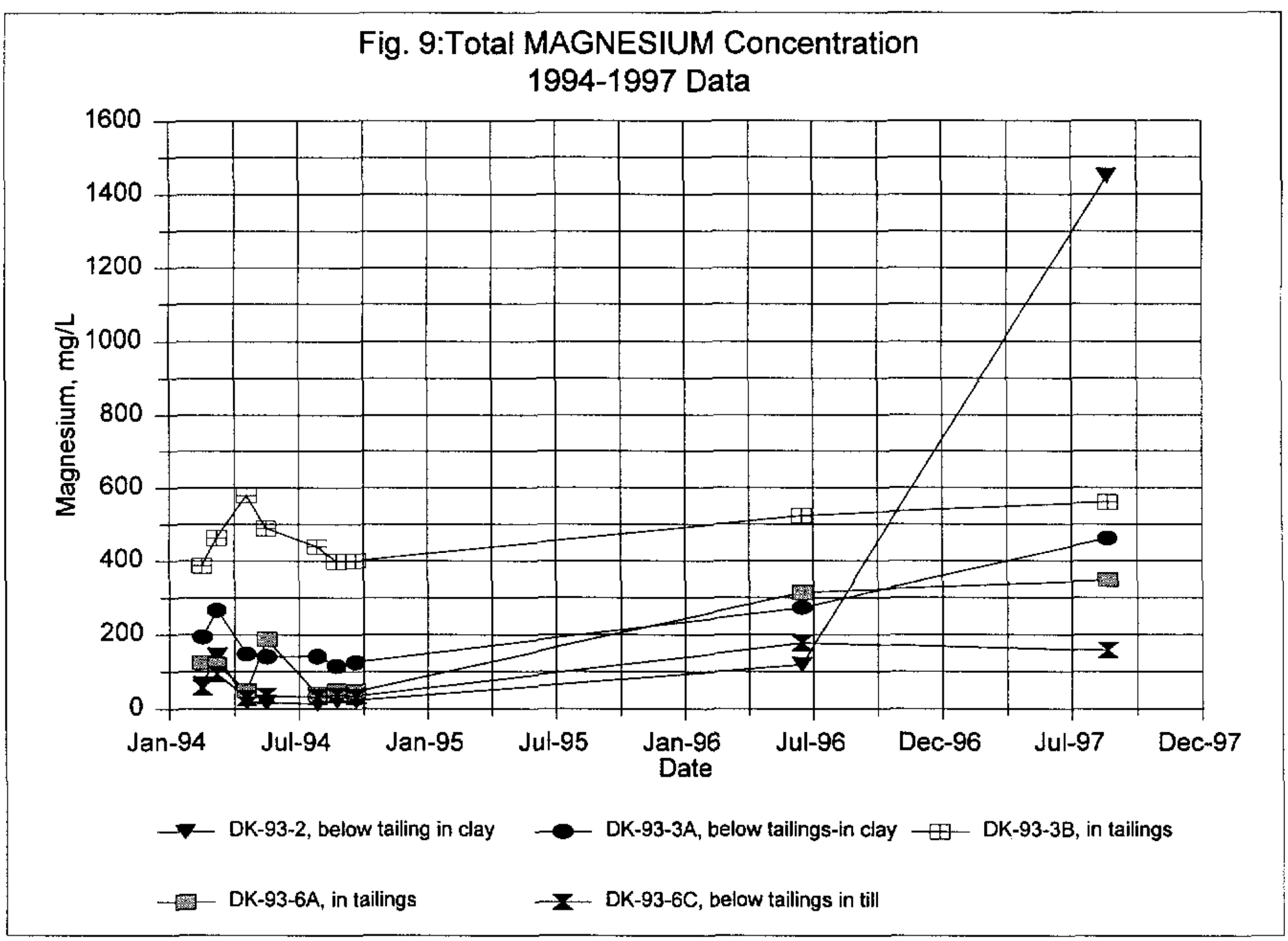


Fig. 9:Total MAGNESIUM Concentration
1994-1997 Data
1600
1400
1200
1000
800
600
400
200
0
Magnesium, mg/L
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

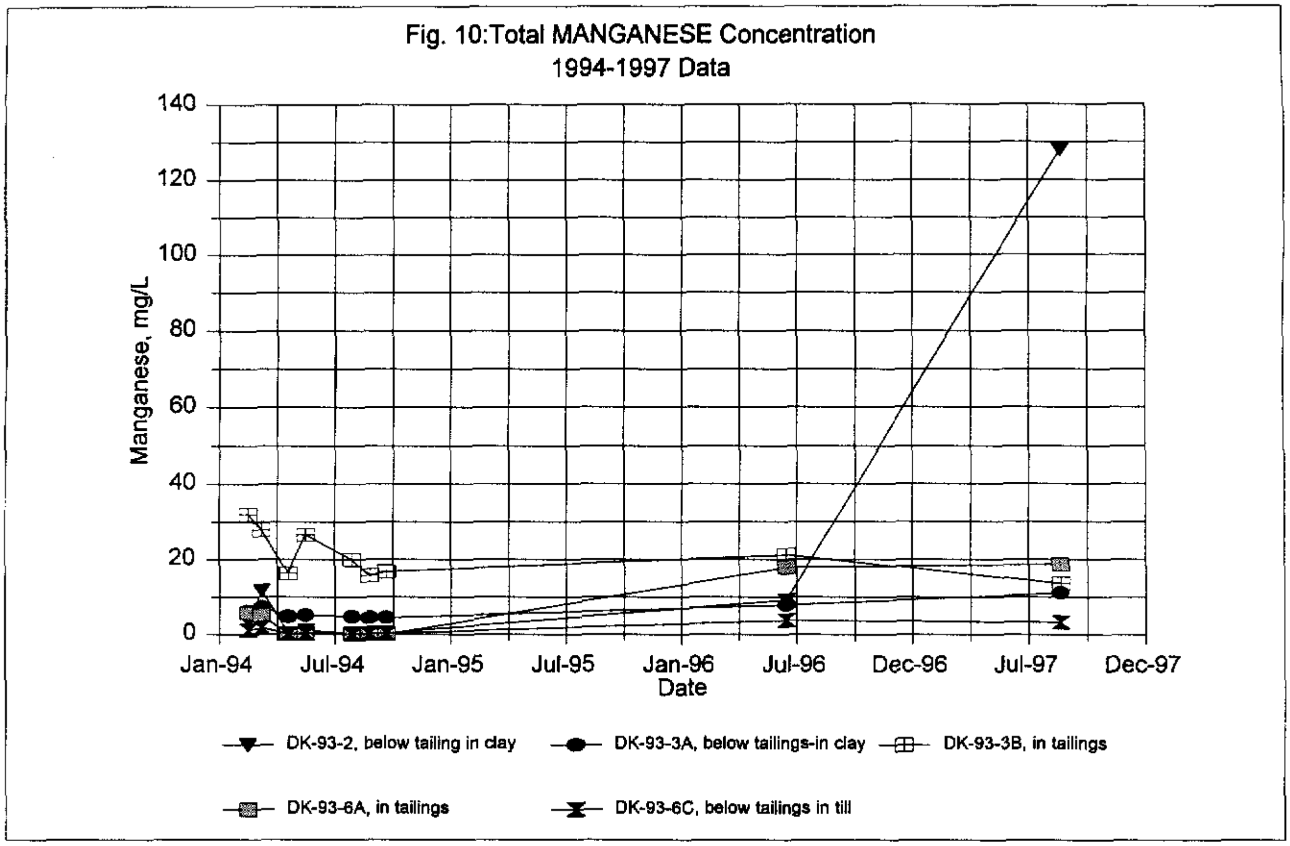


Fig. 10:Total MANGANESE Concentration
1994-1997 Data
Manganese, mg/L
140
120
100
80
60
40
20
0
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

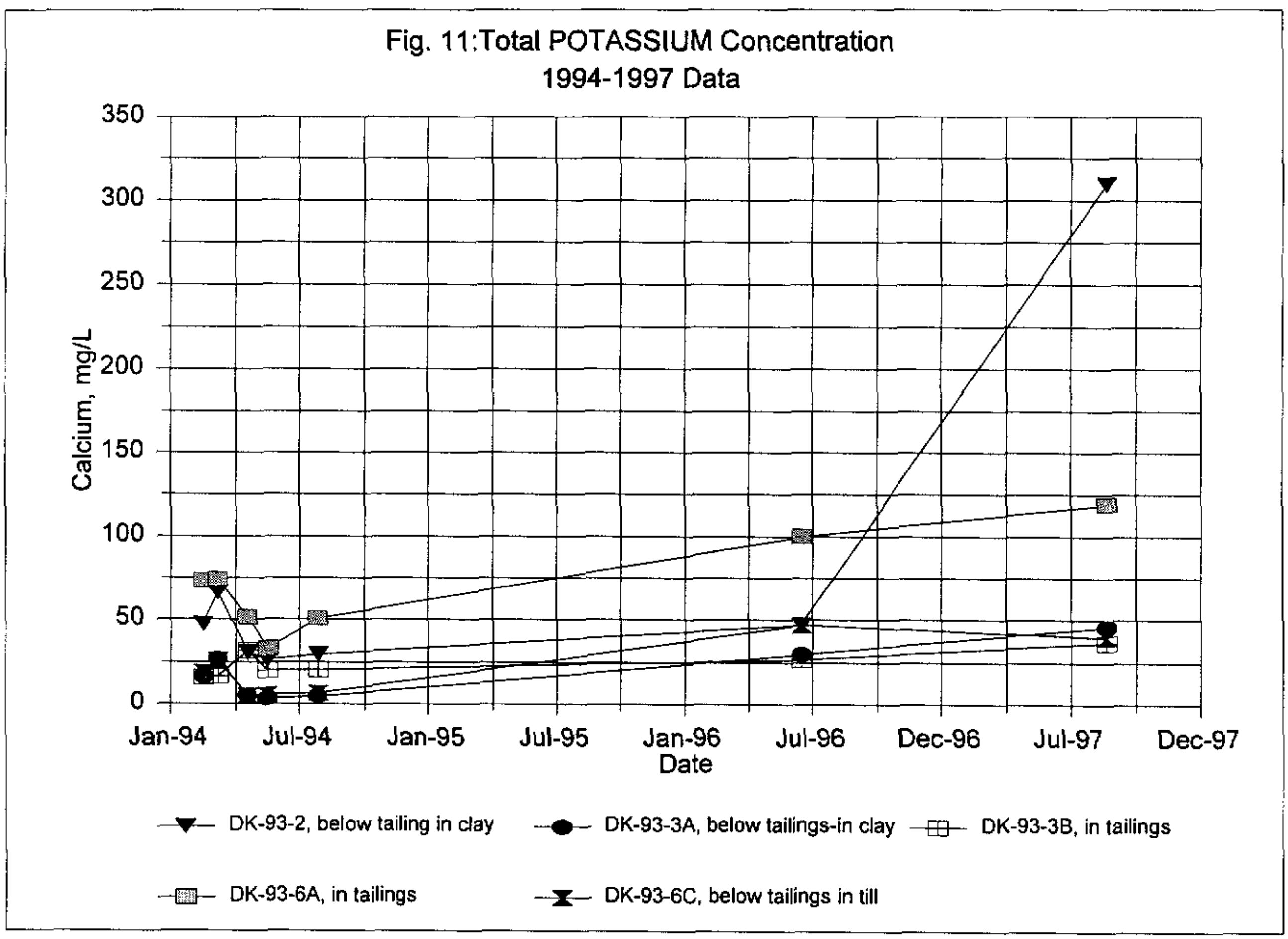


Fig. 11:Total POTASSIUM Concentration
1994-1997 Data
Calcium, mg/L
350
300
250
200
150
100
50
0
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

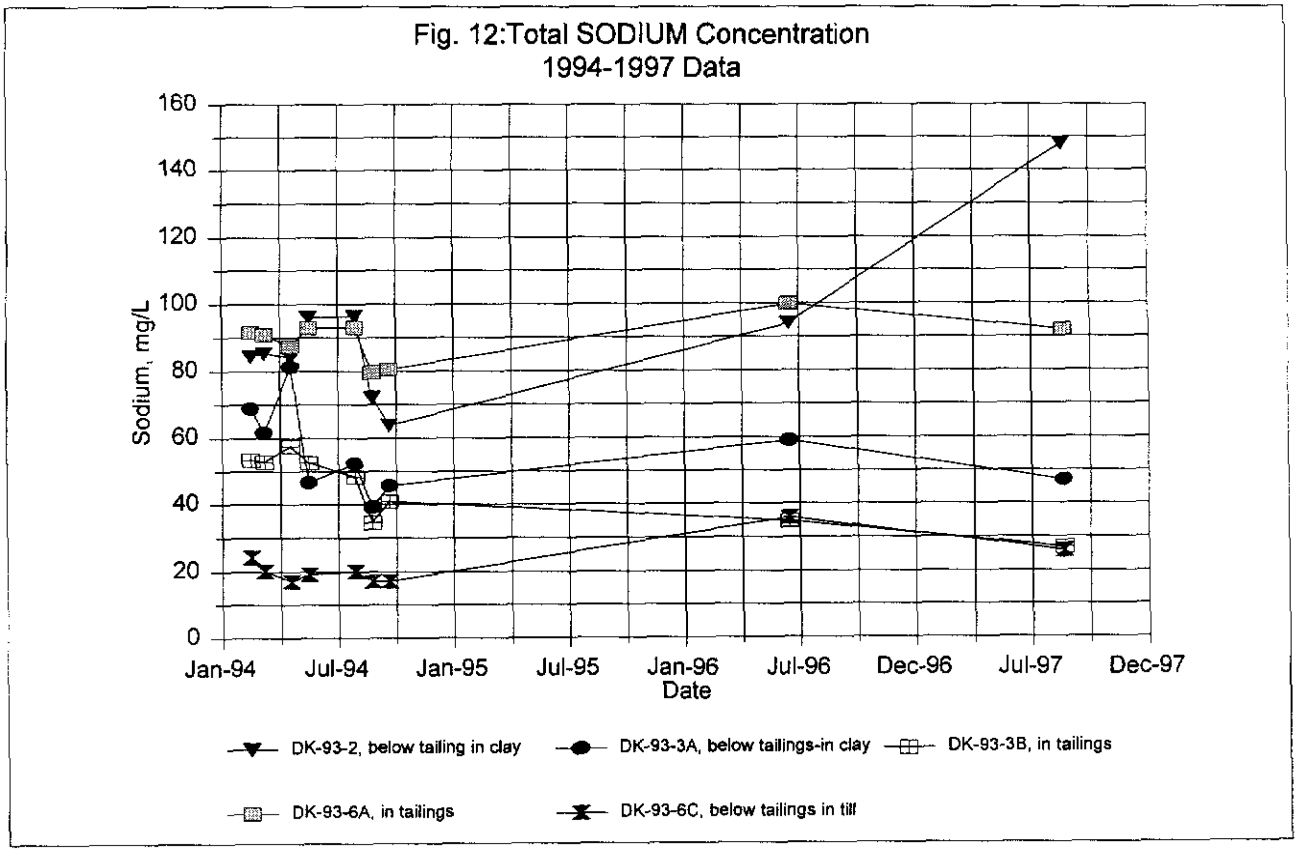


Fig. 12:Total SODIUM Concentration
1994-1997 Data
Sodium, mg/L
0
20
40
60
80
100
120
140
160
Jan-94
Jul-94
Jan-95
Jul-95
Jan-96
Jul-96
Dec-96
Jul-97
Dec-97
Date
DK-93-2, below tailing in clay
DK-93-3A, below tailings-in clay
DK-93-3B, in tailings
DK-93-6A, in tailings
DK-93-6C, below tailings in till

4.0 CONCLUSION

From the evaluation of the data generated during the field investigation and the experiments carried out both on site and in the laboratory, it can be concluded that acid generation and its associated problems resulting in acidic effluents with high metal concentrations is not a problem anticipated for the decommissioning of the Goldcorp tailings area.

Although the tailings undergo oxidation in certain locations, samples which reflect a weathering time span of about 10 years have not produced indication of depletion of neutralizing potential. Although attempts were made to determine the rate of oxidation and neutralisation provided by the alkalinity generating minerals present, it was not possible. It is evident that iron oxides are re-precipitating in the pore water of the tailings and high sulphate pore water, likely saturated gypsum solutions, are forming as evidenced in one of the piezometer's water quality data.

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APPENDIX A

LIST OF TABLES

Table 1a: Estimate of Tailings Tonnage, 1949-1978

Year	Tonnes Milled	Cumulative Tones Milled	Estimated Tonnes of Tailings @*0.99	Cumulative Tonnes of Tailings	Cleaner Concentrate @2.7%	Cumulative Cleaner Concentrate	COMMENT
1949	43,523	43,523	43,088	43,088			
1950	46,234	89,757	45,772	88,859			
1951	90,572	180,329	89,666	178,526			Roaster started
1952	102,442	282,771	101,418	279,943			
1953	125,606	408,377	124,350	404,293			
1954	131,554	539,931	130,238	534,532			
1955	144,261	684,192	142,818	677,350			
1956	156,980	841,172	155,410	832,760			
1957	163,793	1,004,965	162,155	994,915			
1958	162,659	1,167,624	161,032	1,155,948			
1959	85,613	1,253,237	84,757	1,240,705			Assume 6 months
1949-59		**1,253,237**		**1,240,705**			Backfill plant in operati
1959	85,613	85,613	47,087	47,087			45% to mine
1960	171,840	257,453	94,512	141,599			
1961	171,935	429,388	94,564	236,163			
1962	175,767	605,155	96,672	332,835			
1963	178,527	783,682	98,190	431,025			
1964	177,874	961,556	97,831	528,856			
1965	177,353	1,138,909	97,544	626,400			
1959-65		**1,138,909**		**626,400**			
1949-65		**2,392,146**		**1,867,105**			
1966	172,526	172,526	94,889	94,889			
1967	168,577	341,103	92,717	187,607			
1968	160,825	501,928	88,454	276,060			
1969	174,273	676,201	95,850	371,911			
1970	167,663	843,864	92,215	464,125			
1971	148,636	992,500	81,750	545,875			
1972	156,726	1,149,226	86,199	632,074			
1973	149,286	1,298,512	82,107	714,182			
1974	151,009	1,449,521	83,055	797,237			
1975	126,307	1,575,828	69,469	866,705			
1976	117,459	1,693,287	64,602	931,308			
1977	129,184	1,822,471	71,051	1,002,359			
1978	110,438	1,932,909	60,741	1,063,100			
1966-78		**1,932,909**		**1,063,100**			

Table 1b: Estimate of Tailings Tonnage, 1979-1996

Year	Tonnes Milled	Cumulative Tones Milled	Estimated Tonnes of Tailings @0.973*0.55	Cumulative Tonnes of Tailings	Cleaner Concentrate @2.7%	Cumulative Cleaner Concentrate	COMMENT
1979	117,716	117,716	62,996	62,996	3,178	3,178	Roaster shut down
1980	128,180	245,896	68,596	131,591	3,461	6,639	Assume January 1, 79
1981	189,494	435,390	101,408	232,999	5,116	11,756	
1982	197,265	632,655	105,566	338,565	5,326	17,082	
1979-82		**632,655**		**338,565**		**17,082**	
1983	205,768	205,768	110,117	110,117	5,556	5,556	
1984	206,873	412,641	110,708	220,825	5,586	11,141	
1985	215,140	627,781	115,132	335,957	5,809	16,950	
1986	239,736	867,517	128,295	464,252	6,473	23,423	
1987	242,140	1,109,657	129,581	593,833	6,538	29,961	
1983-87		**1,109,657**		**593,833**		**29,961**	
1988	259,211	259,211	138,717	138,717	6,999	6,999	
1989	276,784	535,995	148,121	286,838	7,473	14,472	
1990	284,506	820,501	152,253	439,091	7,682	22,154	
1991	315,017	1,135,518	168,581	607,672	8,505	30,659	
1992	322,646	1,458,164	172,664	780,336	8,711	39,370	
1993	334,736	1,792,900	179,134	959,470	9,038	48,408	
1994	318,122	2,111,022	170,243	1,129,713	8,589	56,998	
1995	227,490	2,338,512	121,741	1,251,455	6,142	63,140	
1996	93,882	2,432,394	50,241	1,301,696	2,535	65,675	
1988-96		**2,432,394**		**1,301,696**		**65,675**	

Table 2a: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids

* Values require verification (illegible on fax of data)
Hawley, J.R., 1979/1980. The Chemical Characteristics of Mineral Tailings in the Province of Ontario. MOE Report.

	UNITS	Old Tailings	New Tailings	Waste Rock	Red Lake Tailings (Hawley, 1979)
Ag	$ug.g^{-1}$	7000	9000	17000	
Al	$ug.g^{-1}$	16200	18400	15200	32300
As	$ug.g^{-1}$	1615	1293	2286	4176
B	$ug.g^{-1}$	1	1	1	99
Ba	$ug.g^{-1}$	57	118	108	17
Be	$ug.g^{-1}$	1.2	1.2	1.3	
Bi	$ug.g^{-1}$	11	13	11	<10
Ca	$ug.g^{-1}$	28600	27900	31100	3900
Cd	$ug.g^{-1}$	1000	1000	1000	18000
Co	$ug.g^{-1}$	25	22	28	22
Cu	$ug.g^{-1}$	114	115	109	59
Fe	$ug.g^{-1}$	74800	78500	67000	48000
K	$ug.g^{-1}$	4000	6200	5400	14000
Li	$ug.g^{-1}$	27	41	38	
Mg	$ug.g^{-1}$	18400	24400	23500	11000
Mn	$ug.g^{-1}$	1358	1510	1356	940
Mo	$ug.g^{-1}$	9	10	7	<1.1
Na	$ug.g^{-1}$	700	800	700	1700 - 21000
Ni	$ug.g^{-1}$	147	131	172	93
P	$ug.g^{-1}$	330	410	350	542
Pb	$ug.g^{-1}$	95	176	37	25
S, total	$ug.g^{-1}$	22500	14600	19300	12000
S as SO_4	$ug.g^{-1}$	1400	700	1700	
Sb	$ug.g^{-1}$	61	70	63	309
Si					
Sr	$ug.g^{-1}$	94	100	99	80
Th	$ug.g^{-1}$	1	1	1	
Tl					
Ti	$ug.g^{-1}$	700	1100	900	726
V	$ug.g^{-1}$	197	236	186	117
Zn	$ug.g^{-1}$	98	92	70	162
Zr					
Ga	$ug.g^{-1}$	3	3	4	
Sn	$ug.g^{-1}$	1	1	1	67
W	$ug.g^{-1}$	26	32	25	
Cr	$ug.g^{-1}$	215	243	229	71
Cyanide total					
TOC					
gravel (4.75-9.5mm)					
gravel (2.36-4.75mm)					
very coarse sand (1.18-2.36mm)					
coarse sand (0.60-1.18mm)					
medium sand (0.30-0.60mm)					
fine sand (0.15-0.30mm)					
very fine sand (0.075-0.15mm)					
silt and clay (<0.075mm)					

Table 2b: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids (continuation)

	UNITS	BCDMD 048107 94 94/10/28	BCUMC 048108 94 94/10/28	BCUCC 048109 94 94/10/29	STPT 048110 94 94/10/30	BCDCW 048111 94 94/10/29	SP 048112 94 94/10/30
Ag	$ug.g^{-1}$	5.3	5.4	0.6	<0.5	4.6	5
Al	$ug.g^{-1}$	34000	34000	15000	3400	31000	51000
As	$ug.g^{-1}$	3700	3400	90	310	5500	3200
B	$ug.g^{-1}$	<10	<10	<10	17	<10	<10
Ba	$ug.g^{-1}$	56	57	85	43	43	120
Be	$ug.g^{-1}$	<0.1	<0.1	0.3	<0.1	<0.1	0.1
Bi	$ug.g^{-1}$						
Ca	$ug.g^{-1}$	31000	32000	10000	28000	30000	42000
Cd	$ug.g^{-1}$	74	71	1.9	6.8	110	65
Co	$ug.g^{-1}$	75	91	13	18	75	36
Cu	$ug.g^{-1}$	1200	1200	28	250	420	2200
Fe	$ug.g^{-1}$	77000	81000	15000	8100	75000	59000
K	$ug.g^{-1}$	4400	4200	1500	270	3400	9100
Li	$ug.g^{-1}$						
Mg	$ug.g^{-1}$	24000	25000	5200	3800	20000	29000
Mn	$ug.g^{-1}$	1800	2000	300	200	1500	1600
Mo	$ug.g^{-1}$	6	3	2	2	3	7
Na	$ug.g^{-1}$	700	690	190	440	750	1100
Ni	$ug.g^{-1}$	410	410	22	150	300	160
P	$ug.g^{-1}$	630	620	690	580	260	420
Pb	$ug.g^{-1}$	140	140	<10	16	60	340
S, total	$ug.g^{-1}$	11000	12000	3700	5300	18000	7600
S as SO_4	$ug.g^{-1}$						
Sb	$ug.g^{-1}$						
Si	$ug.g^{-1}$	680	830	270	460	840	390
Sr	$ug.g^{-1}$	38	38	32	60	38	58
Th	$ug.g^{-1}$						
Tl	$ug.g^{-1}$	<20	<20	<20	<20	<20	<20
Ti	$ug.g^{-1}$	620	560	810	52	450	1100
V	$ug.g^{-1}$	170	170	31	13	160	190
Zn	$ug.g^{-1}$	610	610	97	160	430	2500
Zr	$ug.g^{-1}$	<5	<5	<5	<5	<5	6
Ga	$ug.g^{-1}$						
Sn	$ug.g^{-1}$	<5	<5	<5	<5	<5	<5
W	$ug.g^{-1}$						
Cr	$ug.g^{-1}$	140	110	32	12	83	170
Cyanide total	$ug.g^{-1}$	29	4.3	0.34	0.9	9.5	79
TOC	$ug.g^{-1}$	22000	21000	120000	420000	10000	16000
gravel (4.75-9.5mm)	$ug.g^{-1}$	<0.1	<0.1	3.3	3.2	<0.1	<0.1
gravel (2.36-4.75mm)	$ug.g^{-1}$	<0.1	<0.1	5.1	6.9	0.2	<0.1
very coarse sand (1.18-2.36mm)	$ug.g^{-1}$	<0.1	<0.1	4.8	10	0.3	<0.1
coarse sand (0.60-1.18mm)	$ug.g^{-1}$	0.2	<0.1	3.4	16	0.7	0.2
medium sand (0.30-0.60mm)	$ug.g^{-1}$	0.1	1.5	9.9	22	2	0.1
fine sand (0.15-0.30mm)	$ug.g^{-1}$	5	5.3	16	17	27	0.3
very fine sand (0.075-0.15mm)	$ug.g^{-1}$	5.3	4.9	10	7.6	32	0.2
silt and clay (<0.075mm)	$ug.g^{-1}$	89	88	48	17	38	99

Table 2c: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids (continuation)

	UNITS	BLN7E7 048113 94 94/10/31	BLN7E2 048114 94 94/10/31	BLN4E3 048115 94 94/10/31	CL 048116 94 94/11/01	SP 0-5 cm bulk 011742 95 95/03/01	SP 5-10 cm bulk 011748 95 95/03/01
Ag	$ug.g^{-1}$	10	15	8.9	1.1	13	17
Al	$ug.g^{-1}$	35000	44000	47000	16000	41000	36000
As	$ug.g^{-1}$	2700	3500	3600	40	3300	4000
B	$ug.g^{-1}$	<10	<10	<10	<10	<10	<10
Ba	$ug.g^{-1}$	62	88	88	130	96	61
Be	$ug.g^{-1}$	0.2	0.2	0.1	0.7	<0.1	<0.1
Bi	$ug.g^{-1}$						
Ca	$ug.g^{-1}$	35000	30000	38000	6000		
Cd	$ug.g^{-1}$	56	70	73	1.1	75	86
Co	$ug.g^{-1}$	28	56	44	12	47	70
Cu	$ug.g^{-1}$	780	2700	1200	67	1200	220
Fe	$ug.g^{-1}$	56000	59000	67000	19000	79000	110000
K	$ug.g^{-1}$	4500	5400	6400	1800		
Li	$ug.g^{-1}$						
Mg	$ug.g^{-1}$	29000	26000	30000	4100		
Mn	$ug.g^{-1}$	1200	1200	1400	670	1400	1200
Mo	$ug.g^{-1}$	10	5	6	1	3	2
Na	$ug.g^{-1}$	790	940	1100	130		
Ni	$ug.g^{-1}$	210	710	280	45	200	330
P	$ug.g^{-1}$	560	670	470	1300	270	110
Pb	$ug.g^{-1}$	110	180	150	34	190	79
S, total	$ug.g^{-1}$	3000	3800	5400	2400		
S as SO_4	$ug.g^{-1}$						
Sb	$ug.g^{-1}$						
Si	$ug.g^{-1}$	410	370	430	530		
Sr	$ug.g^{-1}$	53	56	58	27	56	52
Th	$ug.g^{-1}$						
Tl	$ug.g^{-1}$	<20	<20	<20	<20		
Ti	$ug.g^{-1}$	680	840	840	480	810	450
V	$ug.g^{-1}$	110	150	170	37	200	230
Zn	$ug.g^{-1}$	6000	3400	3000	200	1400	450
Zr	$ug.g^{-1}$	<5	<5	5	<5		
Ga	$ug.g^{-1}$						
Sn	$ug.g^{-1}$	<5	<5	<5	<5	<5	<5
W	$ug.g^{-1}$						
Cr	$ug.g^{-1}$	280	230	220	40	140	140
Cyanide total	$ug.g^{-1}$	260	110	110	0.67		
TOC	$ug.g^{-1}$	38000	32000	22000	110000		
gravel (4.75-9.5mm)	$ug.g^{-1}$	<0.1	<0.1	<0.1	1.1		
gravel (2.36-4.75mm)	$ug.g^{-1}$	<0.1	0.3	<0.1	9.8		
very coarse sand (1.18-2.36mm)	$ug.g^{-1}$	0.1	0.1	<0.1	15		
coarse sand (0.60-1.18mm)	$ug.g^{-1}$	0.1	0.1	<0.1	12		
medium sand (0.30-0.60mm)	$ug.g^{-1}$	11	3.3	3.6	16		
fine sand (0.15-0.30mm)	$ug.g^{-1}$	11	4.4	4.5	12		
very fine sand (0.075-0.15mm)	$ug.g^{-1}$	6	2.7	2.4	6.6		
silt and clay (<0.075mm)	$ug.g^{-1}$	72	89	90	28		

Table 2d: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids (continuation)

	UNITS	SP 10-15 cm bulk 011749 95 95/03/01	BCMD 0-5 cm bulk 011755 95 95/03/01	BCMD 5-10 cm bulk 011761 95 95/03/01	BCMD 10-15 cm bulk 011762 95 95/03/01	BCDCW 0-5 cm bulk 011768 95 95/03/01
Ag	$ug.g^{-1}$	18	13	12	15	14
Al	$ug.g^{-1}$	35000	33000	36000	34000	29000
As	$ug.g^{-1}$	3900	3600	3300	3000	5800
B	$ug.g^{-1}$	<10	<10	<10	<10	<10
Ba	$ug.g^{-1}$	58	61	60	55	43
Be	$ug.g^{-1}$	<0.1	<0.1	<0.1	<0.1	<0.1
Bi	$ug.g^{-1}$					
Ca	$ug.g^{-1}$					
Cd	$ug.g^{-1}$	83	77	70	63	120
Co	$ug.g^{-1}$	71	78	57	58	66
Cu	$ug.g^{-1}$	220	1300	810	540	400
Fe	$ug.g^{-1}$	110000	76000	74000	87000	88000
K	$ug.g^{-1}$					
Li	$ug.g^{-1}$					
Mg	$ug.g^{-1}$					
Mn	$ug.g^{-1}$	1200	1900	1800	2000	1600
Mo	$ug.g^{-1}$	3	2	2	1	1
Na	$ug.g^{-1}$					
Ni	$ug.g^{-1}$	340	460	380	340	330
P	$ug.g^{-1}$	92	670	420	320	170
Pb	$ug.g^{-1}$	78	140	130	150	70
S, total	$ug.g^{-1}$					
S as SO_4	$ug.g^{-1}$					
Sb	$ug.g^{-1}$					
Si	$ug.g^{-1}$					
Sr	$ug.g^{-1}$	51	42	44	40	39
Th	$ug.g^{-1}$					
Tl	$ug.g^{-1}$					
Ti	$ug.g^{-1}$	420	580	590	<5	390
V	$ug.g^{-1}$	240	150	160	170	160
Zn	$ug.g^{-1}$	440	670	580	560	650
Zr	$ug.g^{-1}$					
Ga	$ug.g^{-1}$					
Sn	$ug.g^{-1}$	<5	<5	<5	<5	<5
W	$ug.g^{-1}$					
Cr	$ug.g^{-1}$	140	110	110	110	86
Cyanide total	$ug.g^{-1}$					
TOC	$ug.g^{-1}$					
gravel (4.75-9.5mm)	$ug.g^{-1}$					
gravel (2.36-4.75mm)	$ug.g^{-1}$					
very coarse sand (1.18-2.36mm)	$ug.g^{-1}$					
coarse sand (0.60-1.18mm)	$ug.g^{-1}$					
medium sand (0.30-0.60mm)	$ug.g^{-1}$					
fine sand (0.15-0.30mm)	$ug.g^{-1}$					
very fine sand (0.075-0.15mm)	$ug.g^{-1}$					
silt and clay (<0.075mm)	$ug.g^{-1}$					

Table 2e: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids (continuation)

	UNITS	BCDCW 5-10 cm bulk 011774 95 95/03/01	BCDCW 10-15 cm bulk 011775 95 95/03/01	BCSTP 0-5 cm bulk 011781 95 95/03/01	BCSTP 5-10 cm bulk 011787 95 95/03/01	BCSTP 10-15 cm bulk 011788 95 95/03/01
Ag	$ug.g^{-1}$	15	15	13	15	14
Al	$ug.g^{-1}$	31000	30000	37000	34000	31000
As	$ug.g^{-1}$	6200	5500	6900	5900	5300
B	$ug.g^{-1}$	<10	<10	<10	<10	<10
Ba	$ug.g^{-1}$	48	40	72	50	43
Be	$ug.g^{-1}$	<0.1	<0.1	<0.1	<0.1	<0.1
Bi	$ug.g^{-1}$					
Ca	$ug.g^{-1}$					
Cd	$ug.g^{-1}$	130	110	140	130	110
Co	$ug.g^{-1}$	68	60	74	64	66
Cu	$ug.g^{-1}$	550	320	900	500	340
Fe	$ug.g^{-1}$	90000	93000	77000	87000	87000
K	$ug.g^{-1}$					
Li	$ug.g^{-1}$					
Mg	$ug.g^{-1}$					
Mn	$ug.g^{-1}$	1700	1600	1700	1800	1700
Mo	$ug.g^{-1}$	2	1	2	2	1
Na	$ug.g^{-1}$					
Ni	$ug.g^{-1}$	340	300	440	320	320
P	$ug.g^{-1}$	150	140	290	120	110
Pb	$ug.g^{-1}$	83	77	120	91	77
S, total	$ug.g^{-1}$					
S as SO_4	$ug.g^{-1}$					
Sb	$ug.g^{-1}$					
Si	$ug.g^{-1}$					
Sr	$ug.g^{-1}$	41	39	43	42	40
Th	$ug.g^{-1}$					
Tl	$ug.g^{-1}$					
Ti	$ug.g^{-1}$	420	360	630	450	380
V	$ug.g^{-1}$	170	180	170	170	170
Zn	$ug.g^{-1}$	710	440	670	740	790
Zr	$ug.g^{-1}$					
Ga	$ug.g^{-1}$					
Sn	$ug.g^{-1}$		<5	<5	<5	<5
W	$ug.g^{-1}$					
Cr	$ug.g^{-1}$		89	110	100	93
Cyanide total	$ug.g^{-1}$					
TOC	$ug.g^{-1}$					
gravel (4.75-9.5mm)	$ug.g^{-1}$					
gravel (2.36-4.75mm)	$ug.g^{-1}$					
very coarse sand (1.18-2.36mm)	$ug.g^{-1}$					
coarse sand (0.60-1.18mm)	$ug.g^{-1}$					
medium sand (0.30-0.60mm)	$ug.g^{-1}$					
fine sand (0.15-0.30mm)	$ug.g^{-1}$					
very fine sand (0.075-0.15mm)	$ug.g^{-1}$					
silt and clay (<0.075mm)	$ug.g^{-1}$					

Table 2f: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids (continuation)

	UNITS	BLN4E3 0-5 cm bulk 011794 95 95/03/01	BLN4E3 5-10 cm bulk 011800 95 95/03/01	BLN4E3 10-15 cm bulk 011806 95 95/03/01	BLN7E2 0-5 cm bulk 011812 95 95/03/01	BLN7E2 5-10 cm bulk 011813 95 95/03/01
Ag	$ug.g^{-1}$	14	7	3.8	12	4.2
Al	$ug.g^{-1}$	45000	29000	22000	35000	20000
As	$ug.g^{-1}$	2900	1500	150	2400	330
B	$ug.g^{-1}$	<10	10	10	<10	11
Ba	$ug.g^{-1}$	81	110	130	70	110
Be	$ug.g^{-1}$	0.1	0.4	0.5	0.2	0.4
Bi	$ug.g^{-1}$					
Ca	$ug.g^{-1}$					
Cd	$ug.g^{-1}$	61	33	4.2	51	7.8
Co	$ug.g^{-1}$	63	50	18	52	19
Cu	$ug.g^{-1}$	2400	880	70	1400	190
Fe	$ug.g^{-1}$	68000	42000	26000	59000	26000
K	$ug.g^{-1}$					
Li	$ug.g^{-1}$					
Mg	$ug.g^{-1}$					
Mn	$ug.g^{-1}$	1500	810	480	1300	470
Mo	$ug.g^{-1}$	3	3	<1	3	<1
Na	$ug.g^{-1}$					
Ni	$ug.g^{-1}$	640	560	89	560	160
P	$ug.g^{-1}$	510	720	850	480	720
Pb	$ug.g^{-1}$	150	62	12	96	18
S, total	$ug.g^{-1}$					
S as SO_4	$ug.g^{-1}$					
Sb	$ug.g^{-1}$					
Si	$ug.g^{-1}$					
Sr	$ug.g^{-1}$	56	51	46	51	39
Th	$ug.g^{-1}$					
Tl	$ug.g^{-1}$					
Ti	$ug.g^{-1}$	750	<5	910	650	810
V	$ug.g^{-1}$	170	87	45	130	45
Zn	$ug.g^{-1}$	2200	1100	190	1800	310
Zr	$ug.g^{-1}$					
Ga	$ug.g^{-1}$					
Sn	$ug.g^{-1}$	<5	<5	<5	<5	<5
W	$ug.g^{-1}$					
Cr	$ug.g^{-1}$	240	120	50	250	58
Cyanide total	$ug.g^{-1}$					
TOC	$ug.g^{-1}$					
gravel (4.75-9.5mm)	$ug.g^{-1}$					
gravel (2.36-4.75mm)	$ug.g^{-1}$					
very coarse sand (1.18-2.36mm)	$ug.g^{-1}$					
coarse sand (0.60-1.18mm)	$ug.g^{-1}$					
medium sand (0.30-0.60mm)	$ug.g^{-1}$					
fine sand (0.15-0.30mm)	$ug.g^{-1}$					
very fine sand (0.075-0.15mm)	$ug.g^{-1}$					
silt and clay (<0.075mm)	$ug.g^{-1}$					

Table 2g: GOLDCORP INC. , Red Lake Mine: ICP Analyses of Solids (continuation)

	UNITS	BLN7E2 10-15 cm bulk 011814 95 95/03/01	BLN7E7 0-5 cm bulk 011815 95 95/03/01	BLN7E7 5-10 cm bulk 011816 95 95/03/01	BLN7E7 10-15 cm bulk 011817 95 95/03/01
Ag	$ug.g^{-1}$	4.5	16	4.6	3.7
Al	$ug.g^{-1}$	23000	42000	21000	19000
As	$ug.g^{-1}$	230	2900	480	76
B	$ug.g^{-1}$	11	<10	11	<10
Ba	$ug.g^{-1}$	140	95	110	120
Be	$ug.g^{-1}$	0.6	0.3	0.5	0.5
Bi	$ug.g^{-1}$				
Ca	$ug.g^{-1}$				
Cd	$ug.g^{-1}$	5.3	57	10	3.4
Co	$ug.g^{-1}$	20	98	31	15
Cu	$ug.g^{-1}$	160	4600	540	55
Fe	$ug.g^{-1}$	29000	61000	28000	23000
K	$ug.g^{-1}$				
Li	$ug.g^{-1}$				
Mg	$ug.g^{-1}$				
Mn	$ug.g^{-1}$	530	1200	480	400
Mo	$ug.g^{-1}$	<1	7	2	<1
Na	$ug.g^{-1}$				
Ni	$ug.g^{-1}$	130	1400	310	56
P	$ug.g^{-1}$	840	650	860	810
Pb	$ug.g^{-1}$	16	170	27	15
S, total	$ug.g^{-1}$				
S as SO_4	$ug.g^{-1}$				
Sb	$ug.g^{-1}$				
Si	$ug.g^{-1}$				
Sr	$ug.g^{-1}$	42	56	42	37
Th	$ug.g^{-1}$				
Tl	$ug.g^{-1}$				
Ti	$ug.g^{-1}$	1000	880	920	880
V	$ug.g^{-1}$	51	150	49	40
Zn	$ug.g^{-1}$	280	2500	340	140
Zr	$ug.g^{-1}$				
Ga	$ug.g^{-1}$				
Sn	$ug.g^{-1}$	<5	<5	<5	<5
W	$ug.g^{-1}$				
Cr	$ug.g^{-1}$	62	200	55	45
Cyanide total	$ug.g^{-1}$				
TOC	$ug.g^{-1}$				
gravel (4.75-9.5mm)	$ug.g^{-1}$				
gravel (2.36-4.75mm)	$ug.g^{-1}$				
very coarse sand (1.18-2.36mm)	$ug.g^{-1}$				
coarse sand (0.60-1.18mm)	$ug.g^{-1}$				
medium sand (0.30-0.60mm)	$ug.g^{-1}$				
fine sand (0.15-0.30mm)	$ug.g^{-1}$				
very fine sand (0.075-0.15mm)	$ug.g^{-1}$				
silt and clay (<0.075mm)	$ug.g^{-1}$				

Table 3. The changes in pH over the decanted times and the DH_2O volume addition

June 20-July 7, 1997

Decanting Times	un-50/50		un-50/100		un-50/150		ox-50/50		ox-50/100		ox-50/150	
	pH	DH_2O added mL	pH	DH_2O added mL	pH	DH_2O added mL	pH	DH_2O added mL	pH	DH_2O added mL	pH	DH_2O added mL
1	8.90	50	9.07	100	9.08	150	7.30	50	7.51	100	7.53	150
2	8.70	100	8.90	200	9.14	300	7.34	100	7.43	200	7.48	300
3	8.03	150	8.13	300	8.24	450	7.13	150	7.25	300	7.26	450
4	8.44	200	8.62	400	9.29	600	7.21	200	7.17	400	7.25	600
5	8.64	250	8.85	500	9.25	750	7.38	250	7.31	500	7.32	750
6	8.67	300	8.92	600	9.13	900	7.23	300	7.26	600	7.37	900
7	8.14	350	8.28	700	8.69	1050	7.42	350	7.46	700	7.47	1050
8	8.64	400	8.96	800	9.22	1200	7.33	400	7.35	800	7.35	1200
9	8.78	450	9.07	900	9.23	1350	7.26	450	7.25	900	7.30	1350
10	8.66	500	9.01	1000	9.04	1500	7.23	500	7.27	1000	7.32	1500
11	8.59	550	8.86	1100	8.93	1650	7.52	550	7.65	1100	8.01	1650
12	8.72	600	8.87	1200	8.92	1800	7.67	600	7.76	1200	8.27	1800
13	8.50	650	8.84	1300	8.90	1950	7.49	650	7.70	1300	7.94	1950
14	8.57	700	8.63	1400	8.50	2100	7.70	700	7.84	1400	7.80	2100
15	8.67	750	8.68	1500	8.78	2250	7.66	750	7.86	1500	8.01	2250
16	8.59	800	8.52	1600	8.27	2400	7.66	800	7.98	1600	7.92	2400
17	8.31	850	8.32	1700	8.29	2550	7.70	850	7.97	1700	7.94	2550
18	8.43	900	8.51	1800	8.51	2700	7.64	900	7.96	1800	7.97	2700
19	8.24	950	8.28	1900	8.25	2850	7.58	950	7.94	1900	7.92	2850
20	8.25	1000	8.36	2000	8.38	3000	7.69	1000	8.01	2000	8.08	3000
21	8.24	1050	8.28	2100	8.26	3150	7.67	1050	7.94	2100	7.96	3150
22	8.27	1100	8.27	2200	8.24	3300	7.76	1100	8.01	2200	7.99	3300

Un-50/50 = Unoxidized 50mL volume of sample / 50 mL distilled water
Un-50/100 = Unoxidized 50mL volume of sample / 100 mL distilled water
Un-50/150 = Unoxidized 50mL volume of sample / 150mL distilled water
Ox-50/50 = Oxidized 50 mL volume of sample / 50 mL distilled water
Ox-50-100 = Oxidized 50 mL volume of sample / 100 mL distilled water
Ox-50/150 = Oxidized 50 mL volume of sample / 150 mL distilled water

Table 4: Cumulative Measurement, June 20 - July 7, 1997

Sampled Date	Treatment		Lab pH	Titrino pH	Cond. us/cm	Alkalinity CaCO3 ppm	Cumulative DH2O added mL
June 20-30,97 (220 hours)	Unoxidized	50/50	7.97	8.08	560	59.5	900
	Location:	50/100	7.98	8.02	360	49.4	1800
	S1	50/150	7.89	7.87	300	39.7	2700
	Oxidized	50/50	7.57	7.76	2450	36.1	900
	Location:	50/100	7.53	7.66	1970	34.7	1800
	S4	50/150	7.69	7.85	1140	36.8	2700
July 2-July3,97 (228 hours)	Unoxidized	50/50	8.14	8.08	230	62.9	1000
	Location:	50/100	7.95	7.92	190	52.2	2000
	S1	50/150	7.88	7.81	150	33.3	3000
	Oxidized	50/50	7.68	7.6	1210	34.5	1000
	Location:	50/100	7.85	7.89	190	45.2	2000
	S4	50/150	7.83	7.64	170	42.1	3000
July 4, 1997 (312 hours)	Unoxidized	50/50	8.24		280		1050
	Location:	50/100	8.28		200		2100
	S1	50/150	8.26		190		3150
	Oxidized	50/50	7.67		1400		1050
	Location:	50/100	7.94		220		2100
	S4	50/150	7.96		180		3150
July 7, 1997 (385 hours)	Unoxidized	50/50	8.27		400		1100
	Location:	50/100	8.27		300		2200
	S1	50/150	8.24		270		3300
	Oxidized	50/50	7.76		1210		1100
	Location:	50/100	8.01		290		2200
	S4	50/150	7.99		250		3300

50/50 = 50mL of wet sample / 50mL DH2O
50/100 = 50mL of wet sample / 100mL DH2O
50/150 = 50mL of wet sample / 150mL DH2O

Table 5: SEQUENTIAL TAILINGS DIGESTION		Dry Sample (g)	Carrier $D\text{-}H_2O$ (mL)	Glacial Acetic Acid (mL)	35% HCl Acid (mL)	73% HNO_3 Acid (mL)	Mixing Temp °C	Mixing Time (hrs)	Wash $D\text{-}H_2O$ (mL)	Volume Recov'd (mL)	pH	Acidity $mg.L^{-1}$	Dry Weight (g)
Part A: Acetic Acid Digestion: CALCITE													
Step A-1	Mix dry tails with $D\text{-}H_2O$	500	1500										
Step A-2	Add Acetic Acid			200									
Step A-3	measure pH, acidity												
Step A-4	Roll for 6 hours						20°	6					
Step A-5	measure pH, acidity												
Step A-6	Filter tailings												
Step A-7	Save filtrate												
Step A-8	Wash tailings with $D\text{-}H_2O$								1500				
Step A-9	Save wash solution												
Step A-10	Measure wash sol'n pH, acidity												
Step A-11	Dry tailings @ 110 °C												
Step A-12	Weigh tailings												
Part B: Hydrochloric Acid Digestion: DOLOMITE, IRON OXIDES													
Step B-1	Mix dry tails with $D\text{-}H_2O$		1500										
Step B-2	Add Hydrochloric Acid				200								
Step B-3	measure pH, acidity												
Step B-4	Roll for 6 hours						20°	6					
Step B-5	measure pH, acidity												
Step B-6	Filter tailings												
Step B-7	Save filtrate												
Step B-8	Wash tailings with $D\text{-}H_2O$								1500				
Step B-9	Save wash solution												
Step B-10	Measure wash sol'n pH, acidity												
Step B-11	Dry tailings @ 110 °C												
Step B-12	Weigh tailings												
Part C: Cold Nitric Acid Digestion: ARSENOPYRITE, SOME PYRITE, SECONDARY CU MINERALS													
Step C-1	Mix dry tails with $D\text{-}H_2O$		1500										
Step C-2	Add Nitric Acid					200							
Step C-3	measure pH, acidity												
Step C-4	Roll for 6 hours						20°	6					
Step C-5	measure pH, acidity												
Step C-6	Filter tailings												
Step C-7	Save filtrate												
Step C-8	Wash tailings with $D\text{-}H_2O$								1500				
Step C-9	Save wash solution												
Step C-10	Measure wash sol'n pH, acidity												
Step C-11	Dry tailings @ 110 °C												
Step C-12	Weigh tailings												
Part D: Hot Nitric Acid Digestion: PYRITE AND SULPHIDES													
Step D-1	Mix dry tails with $D\text{-}H_2O$		1500										
Step D-2	Add Nitric Acid					200							
Step D-3	measure pH, acidity												
Step D-4	Heat to 80° C and roll for 6 hours						80°	6					
Step D-5	measure pH, acidity												
Step D-6	Filter tailings												
Step D-7	Save filtrate												
Step D-8	Wash tailings with $D\text{-}H_2O$								1500				
Step D-9	Save wash solution												
Step D-10	Measure wash sol'n pH, acidity												
Step D-11	Dry tailings @ 110 °C												
Step D-12	Weigh tailings												

Table 6: GOLDCORP Extractions

	6393	6397	6396	6400	6402	6401		
	Unoxidized Tailings	**Oxidized Tailings**	**Unoxidized Tailings** 97/07/07	**Oxidized Tailings** 97/07/07	**Unoxidized Tailings** 97/07/09	**Oxidized Tailings** 97/07/09		
pH			9.1	7.51	9.42	7.61		
Conductivity (mS/cm)			180	920	70	270		
Alkalinity (mg/L)			33.6	32.1	27.4	20.3		
	mg/g			mg/L				
Al	38300	34900	0.219	0.032	0.308	0.025		
As	4120	3150	1.520	0.064	0.726	0.079		
Cd	98.5	75.4	0.030	0.003	0.013	0.003		
Ca	50200	44800	18.700	119.000	8.800	37.100		
Cu	106	107	0.003	0.003	0.003	0.003		
Fe	64600	70200	0.167	0.125	0.069	0.017		
Mg	18000	20500	1.130	38.400	0.780	4.080		
Mn	1020	1410	0.005	0.568	0.005	0.161		
K	6250	7300	2.400	7.300	1.000	1.900		
Na	1320	813	8.860	8.050	0.950	1.220		
Sr	70.2	54.6	0.039	0.076	0.016	0.022		
S	24600	19300	12.300	143.000	2.080	32.000		
Zn	78.3	93	0.010	0.016	0.005	0.006		
		grams	**Liters**					
	Slurry	**50 :**	**2**					
Miligrams of element in solid sample:			**Miligrams of element in water sample:**				**Total miligrams in**	
							Unoxid.	**Oxidized**
Al	1915	1745	0.438	0.064	0.05	0.616	1.054	0.114
As	206	157.5	3.04	0.128	0.158	1.452	4.492	0.286
Cd	4.925	3.77	0.06	0.006	0.006	0.026	0.086	0.012
Ca	2510	2240	37.4	238	74.2	17.6	55	312.2
Cu	5.3	5.35	0.006	0.006	0.006	0.006	0.012	0.012
Fe	3230	3510	0.334	0.25	0.034	0.138	0.472	0.284
Mg	900	1025	2.26	76.8	8.16	1.56	3.82	84.96
Mn	51	70.5	0.01	1.136	0.322	0.01	0.02	1.458
K	312.5	365	4.8	14.6	3.8	2	6.8	18.4
Na	66	40.65	17.72	16.1	2.44	1.9	19.62	18.54
Sr	3.51	2.73	0.078	0.152	0.044	0.032	0.11	0.196
S	1230	965	24.6	286	64	4.16	28.76	350
Zn	3.915	4.65	0.02	0.032	0.012	0.01	0.03	0.044

PERCENT OF MAXIMUM CONCENTRATION IN EXTRACTIONS		
	Unoxidized	**Oxidized**
Al	0.06	0.01
As	2.18	0.18
Cd	1.75	0.32
Ca	2.19	13.94
Cu	0.23	0.22
Fe	0.01	0.01
Mg	0.42	8.29
Mn	0.04	2.07
K	2.18	5.04
Na	29.73	45.61
Sr	3.13	7.18
S	2.34	36.27
Zn	0.77	0.95

Table 7: Assayers Results of Goldcorp Tailings Distilled Water Extractions

SAMPLE DATE		07-Jul-97	07-Jul-97	09-Jul-97	09-Jul-97
ASSAYERS CODE		6396	6400	6401	6402
SAMPLING LOCATION		Balmertown	Balmertown	Balmertown	Balmertown
		Unoxidized	Oxidized	Oxidized	Unoxidized
		Tailings	Tailings	Tailings	Tailings
		NT	OT-A	OT-A	NT
		S1	S4	S4	S1
Processing code		FX	FX	FX	FX
** L A B **					
Temp. (C)		25.1	25.2	22	21.7
pH		9.1	7.51	7.61	9.42
Cond. (umhos/cm)		180	920	270	70
Eh (mV)					
Acidity (mg/l)					
Alkalinity (mg/l)		33.6	32.1	20.3	27.4
ELEMENTS	Ag	<0.003	<0.003	<0.003	<0.003
	Al	0.219	0.032	<0.025	0.308
	As	1.52	0.064	0.079	0.726
	B	0.02	0.01	0.01	0.01
	Ba	<0.005	<0.005	<0.005	<0.005
	Be	<0.002	<0.002	<0.002	<0.002
	Bi	<0.05	<0.05	<0.05	<0.05
	Ca	18.7	119	37.1	8.8
	Cd	0.03	<0.003	<0.003	0.013
	Co	<0.005	<0.005	<0.005	<0.005
	Cr	<0.005	<0.005	<0.005	<0.005
	Cu	0.003	<0.003	<0.003	<0.003
	Fe	0.167	0.125	0.017	0.069
	K	2.4	7.3	1.9	<1
	Mg	1.13	38.4	4.08	0.78
	Mn	<0.005	0.568	0.161	<0.005
	Mo	0.02	<0.01	<0.01	<0.01
	Na	8.86	8.05	1.22	0.95
	Ni	<0.01	<0.01	<0.01	<0.01
	P	<0.06	<0.06	<0.06	<0.06
	Pb	<0.025	<0.025	<0.025	<0.025
	S	12.3	143	32	2.08
	Sn	<0.05	<0.05	<0.05	<0.05
	Sr	0.039	0.076	0.022	0.016
	Ti	<0.05	<0.05	<0.05	<0.05
	V	<0.01	<0.01	<0.01	<0.01
	Zn	0.01	0.016	0.006	<0.005

Table 8: Assayers Results of Goldcorp Tailings Solid Samples

SAMPLE DATE		17-Jun-97	17-Jun-97	17-Jun-97
ASSAYERS CODE		6393	6394	6397
SAMPLING LOCATION		Balmertown	Balmertown	Balmertown
		New	Concentrate	Oxidized
		Tailings	Tailings	Tailings
		NT	CONC PILE	OT-A
		S1	S5	S4
Processing code		SS	SS	SS
** L A B **				
Temp. (C)				
pH				
Cond. (umhos/cm)				
Eh (mV)				
Acidity (mg/l)				
Alkalinity (mg/l)				
ELEMENTS	Ag	<5	<5	<5
	Al	38300	10100	34900
	As	4120	47800	3150
	B	<5	<5	<5
	Ba	58.2	16.8	64.3
	Be	<5	<5	<5
	Bi	<25	<25	<25
	Ca	50200	12000	44800
	Cd	98.5	1800	75.4
	Co	38.5	590	34.8
	Cr	82.7	11.7	91.4
	Cu	106	950	107
	Fe	64600	298000	70200
	K	6250	631	7300
	Mg	18000	6040	20500
	Mn	1020	498	1410
	Mo	<5	<5	<5
	Na	1320	260	813
	Ni	189	1230	162
	P	115	<30	120
	Pb	126	138	122
	S	24600	383000	19300
	Sn	<25	<25	<25
	Sr	70.2	18.9	54.6
	Ti	405	178	510
	V	123	30.4	128
	Zn	78.3	2370	93
Moisture (%)		0.05	1.59	0.13

Table 9a: 1994 Pore Water Chemistry for Piezometer DK-93-1

DATE	16-Feb	08-Mar	21-Apr	18-May	29-Jul	26-Aug	22-Sep	14-Dec
PHYSICAL TESTS								
Conductivity	326	349	441	350	352	345	350	340
Hardness	156	166	156	182	185	195	168	181
pH	7.83	7.80	7.57	7.84	7.86	7.72	7.98	7.77
TSS	4690	2130	1290	800	1070	5260	1670	3280
DISSOLVED ANIONS								
Alkalinity	185	191	185	187	192	188	185	177
Cl	0.5	0.8	0.6	0.7	0.9	<0.5	<0.5	0.8
SO4	5.7	6.1	5.7	6.6	5.9	6.3	6.5	6.1
NUTRIENTS								
NH4	<0.005	<0.005	<0.019	0.029	0.021	0.011	0.015	<0.005
CYANIDES								
CN	0.007	<0.005	0.024	0.020	0.018	0.043	0.028	0.012
SCN	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.7
WAD-CN	<0.005	<0.005	0.021	0.016	0.007	0.042	0.028	<0.005
DISSOLVED METALS								
As	0.002	0.001	0.002	0.002	<0.02	0.005	0.008	0.043
Cu	0.009	0.001	0.002	0.003	<0.01	0.006	0.005	0.003
Fe	0.048	0.032	0.033	<0.03	<0.03	<0.03	<0.03	<0.03
Pb	<0.001	<0.001	<0.001	<0.001	<0.05	<0.001	0.003	<0.001
Ni	0.002	0.001	0.002	0.004	<0.02	0.009	0.001	0.001
Zn	0.014	0.018	0.013	0.010	0.007	0.005	<0.005	<0.005
Ca	45.4	48.7	46.5	54.3	55.2	58.8	50.0	53.6
K	3.6	2.8	2.9	2.8	3.5			
Mg	10.2	10.8	9.8	11.3	11.5	11.7	10.5	11.3
Mn	0.089	0.086	0.077	0.081	0.068	0.073	0.067	0.049
Na	4.70	3.80	3.70	4.00	5.00	3.26	3.83	5.43
TOTAL METALS								
As	0.005	0.003	0.028	0.003	<0.02	0.039	0.008	
Cu	0.018	0.002	0.002	0.003	<0.1	0.006	0.005	
Fe	12.400	6.460	0.054	1.350	2.320	0.240	0.878	
Pb	0.016	0.010	<0.001	0.004	<0.05	<0.001	0.003	
Ni	0.018	0.007	0.002	0.004	<0.02	0.009	0.003	
Zn	0.154	0.100	0.030	0.030	0.030	0.009	0.021	
Ca	80.1	72.8	46.5	56.3	57.7	59.6	58.5	
K	7.2	5.3	2.9	3.3	3.8			
Mg	23.2	20.9	9.8	12.2	19.3	12.3	13.0	
Mn	0.324	0.232	0.077	0.107	0.109	0.082	0.091	
Na	8.8	5.1	3.7	4.0	4.8	3.3	4.2	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9b: 1995-1996 Pore Water Chemistry for Piezometer DK-93-1

DATE	21-Jun-95	Jun-96	
PHYSICAL TESTS			
Conductivity	319	336	
Hardness	153	237	
pH	7.86	7.9	
TSS	295	858	
DISSOLVED ANIONS			
Alkalinity	174	182	
Cl	0.7	0.5	
SO4	7.3	7	
NUTRIENTS			
NH4	0.21	<0.005	
CYANIDES			
CN	<0.005	<0.005	
SCN	<0.5	<0.5	
WAD-CN	<0.005	<0.005	
METALS	DISSOL.	DISSOL.	TOTAL
Ag	<0.015	<0.01	<0.01
Al	<0.2	<0.2	8.7
As	<0.2	0.0089	0.0133
B	<0.1	<0.1	<0.1
Ba	0.022	0.04	0.1
Be	<0.005	<0.005	<0.005
Bi	<0.1	<0.1	<0.1
Ca	45.1	69.6	69
Cd	<0.01	<0.1	<0.01
Co	<0.015	<0.01	<0.01
Cr	<0.015	<0.01	<0.01
Cu	<0.01	<0.01	0.03
Fe	0.447	<0.03	5.5
Hg	<0.00005		
K	3.1	5	5
Li	<0.015	<0.01	<0.01
Mg	9.83	15.4	17.1
Mn	0.052	0.068	0.159
Mo	<0.03	<0.03	<0.03
Na	4.2	5	6
Ni	<0.02	<0.02	0.02
P	<0.3	<0.3	1.6
Pb	<0.050	<0.05	<0.05
Sb	<0.2	<0.2	<0.2
Se	<0.2	<0.2	<0.2
Si	5.15	6.99	18.2
Sn	<0.3	<0.03	<0.03
Sr	0.072	0.113	0.144
Ti	<0.01	<0.01	0.94
Tl	<0.1	<0.1	<0.1
V	<0.03	<0.03	<0.03
W	<0.1		
Zn	<0.005	0.012	0.066

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9c: 1994 Pore Water Chemistry for Piezometer DK-93-2

DATE	16-Feb	09-Mar	20-Apr	18-May	29-Jul	25-Aug	21-Sep	13-Dec
PHYSICAL TESTS								
Conductivity	670	676	649	665	638	612	640	631
Hardness	104	125	89.4	57.6	67.6	116	64.2	71.1
pH	7.43	7.47	7.44	7.43	7.50	8.48	8.41	8.12
TSS	51500	7980	13600	29300	15300	20700	84400	23500
DISSOLVED ANIONS								
Alkalinity	291	297	243	181	199	220	256	269
Cl	15.9	13.6	17.7	20.1	22.2	24.5	22.4	19.5
SO4	37.9	29.4	52.2	99.2	145.0	63.1	51.6	47.9
NUTRIENTS								
NH4	2.30	3.30	2.48	3.41	1.80	1.87	2.06	2.24
CYANIDES								
CN	0.649	0.416	0.590	0.927	1.550	2.490	1.820	0.714
SCN	8.4	8.1	0.8	<0.05	9.7	11.7	7.8	10.3
WAD-CN	0.020	0.019	0.058	0.012	0.036	0.210	0.153	0.021
DISSOLVED METALS								
As	34.1	36.3	32.5	30.4	33.0	44.1	23.9	35.1
Cu	0.012	<0.001	<0.001	<0.001	<0.01	<0.001	0.002	0.002
Fe	0.183	0.203	0.308	0.325	0.276	0.172	0.118	0.390
Pb	0.001	0.003	0.001	0.001	<0.05	<0.001	<0.001	0.002
Ni	0.009	0.006	0.008	0.006	<0.02	0.004	0.001	0.009
Zn	<0.005	0.013	0.006	<0.005	<0.005	<0.005	<0.005	<0.005
Ca	8.68	9.27	7.88	8.07	7.44	6.9	3.32	6.82
K	37.9	35.9	30.9	24.2	29.7			
Mg	20.1	24.8	16.9	9.1	11.9	23.9	13.6	13.1
Mn	0.022	0.048	0.033	0.01	0.011	0.012	0.012	0.012
Na	83.3	78.1	84.3	96	96.4	72.2	45.4	93.7
TOTAL METALS								
As	35.7	47.5	33.2	32.7	33	44.6	33	
Cu	0.247	0.408	0.024	0.019	<0.01	<0.001	0.012	
Fe	122	252	11.8	25.8	9.1	0.629	7.23	
Pb	0.465	0.847	0.092	0.393	0.117	0.01	0.09	
Ni	0.247	0.585	0.038	0.06	<0.02	0.005	0.019	
Zn	0.666	1.06	0.098	0.273	0.097	<0.005	0.069	
Ca	77.7	246	14.2	27.2	14.5	7.42	13.8	
K	47.4	66.3	30.9	26.5	29.7			
Mg	68.2	145	20.7	18.1	15	24	22.6	
Mn	3.84	11.7	0.397	1	0.365	0.035	0.441	
Na	84.5	85.4	84.3	96	96.4	72.2	63.8	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9d: 1995-1997 Pore Water Chemistry for Piezometer DK-93-2

DATE	22-Jun-95	Jun-96		21-Aug-97	
PHYSICAL TESTS					
Conductivity	572	628		675	
Hardness	50.3	69.2		44.4	
pH	8.68	7.72		7.71	
TSS	14400	1760		47200	
DISSOLVED ANIONS					
Alkalinity	171	211		205	
Cl	32	28.3		27.6	
SO4	69.4	60		86	
NUTRIENTS					
NH4	1.59	-0.005		3.88	
CYANIDES					
CN	0.913	0.962		3.5	
SCN	12.6	10		0.5	
WAD-CN	0.06	-0.083		0.088	
METALS	**DISSOL.**	**DISSOL.**	**TOTAL**	**DISSOL.**	**TOTAL**
Ag	<0.015	<0.01	<0.01	<0.01	<0.04
Al	<0.2	<0.2	120	<0.2	1620
As	31.9	35.9	42.9	35.5	133
B	0.31	0.3	0.3	0.3	0.5
Ba	<0.01	<0.01	0.21	<0.01	2.85
Be	<0.005	<0.005	<0.005	<0.005	<0.02
Bi	<0.1	<0.1	0.2	<0.1	0.9
Ca	5.2	5.05	199	5.48	2680
Cd	<0.01	<0.1	<0.01	<0.005	0.026
Co	0.022	0.02	0.14	0.03	1.6
Cr	<0.015	<0.01	0.33	<0.01	3.98
Cu	<0.01	<0.01	0.39	<0.01	5.57
Fe	0.934	0.33	242	0.48	3310
Hg	<0.00005				
K	25.5	27	47	25	310
Li	0.026	0.03	0.15	0.02	1.64
Mg	9.05	13.7	120	7.45	1450
Mn	0.009	0.011	9.21	0.008	128
Mo	0.12	0.11	0.09	0.12	<0.1
Na	90.7	94	94	88	148
Ni	<0.02	<0.02	0.51	<0.02	6.82
P	<0.3	<0.3	1	<0.3	12
Pb	<0.05	<0.05	0.54	<0.05	6.8
Sb	<0.2	<0.2	<0.2	<0.2	<0.2
Se	<0.2	<0.2	<0.2	<0.2	<0.8
Si	1.36	1.35	118	1.57	872
Sn	<0.3	<0.03	<0.03	<0.03	0.1
Sr	0.032	0.045	0.239	0.039	2.71
Ti	<0.01	<0.01	2.14	<0.01	20.2
Tl	<0.1	<0.1	<0.1	<0.1	<0.4
V	<0.03	<0.03	0.46	<0.03	5.1
W	0.17				
Zn	<0.005	<0.005	0.911	<0.005	11.2

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9e: 1994 Pore Water Chemistry for Piezometer DK-93-3A

DATE	17-Feb	09-Mar	20-Apr	18-May	25-Jul	25-Aug	20-Sep	13-Dec
PHYSICAL TESTS								
Conductivity	3850	2640	2660	2700	2660	2620	2640	2660
Hardness	1580	1510	1530	1570	1540	1420	1280	1690
pH	6.65	6.93	6.70	6.73	6.94	6.66	6.83	6.60
TSS	39500	30500	4310	7640	7160	5100	3740	7010
DISSOLVED ANIONS								
Alkalinity	571	582	626	493	547	585	558	643
Cl	21.9	21.8	21.2	21.3	22.0	22.4	22.4	22.2
SO4	1190	1200	1180	1250	1260	1180	1170	1300
NUTRIENTS								
NH4	0.140	0.325	0.150	0.326	0.305	0.300	0.361	0.480
CYANIDES								
CN	0.037	0.032	0.053	0.026	0.024	0.019	0.027	0.012
SCN	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
WAD-CN	<0.005	0.006	0.011	0.016	0.014	0.009	0.014	<0.005
DISSOLVED METALS								
As	0.638	0.452	0.916	0.268	0.330	0.105	0.373	0.252
Cu	0.002	0.001	0.002	0.001	<0.01	<0.001	0.002	<0.001
Fe	0.289	0.283	0.298	0.224	0.201	0.537	0.464	1.310
Pb	0.002	<0.001	<0.001	<0.001	<0.05	<0.001	<0.001	<0.001
Ni	0.052	0.072	0.056	0.065	0.062	0.052	0.060	0.055
Zn	0.005	0.016	0.007	<0.005	0.005	<0.005	<0.005	<0.005
Ca	383	363	367	395	388	376	329	419
K	4.8	4.1	4.2	3.6	4.4			
Mg	152	146	148	141	138	116	111	156
Mn	5.08	4.87	4.82	5.10	4.78	4.48	4.23	5.54
Na	57.9	56.8	61.3	46.0	50.9	39.3	41.7	50.5
TOTAL METALS								
As	2.140	2.370	0.916	0.294	0.350	0.338	0.373	
Cu	0.142	0.284	0.007	0.001	<0.01	<0.001	0.002	
Fe	77.200	137	7.580	0.309	0.707	0.599	0.822	
Pb	0.025	0.044	0.001	<0.001	<0.05	<0.001	<0.001	
Ni	0.307	0.609	0.057	0.065	0.063	0.053	0.060	
Zn	0.240	0.486	0.036	<0.005	0.007	<0.005	0.006	
Ca	427	520	367	395	388	376	350	
K	16.7	26.9	5.4	3.6	4.9			
Mg	194	267	148	141	143	116	126	
Mn	6.20	7.32	4.83	5.10	4.78	4.49	4.44	
Na	69.0	61.7	81.4	46.7	52.4	39.3	45.8	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9f: 1995-1997 Pore Water Chemistry for Piezometer DK-93-3A

DATE	22-Jun-95	Jun-96		25-Aug-97	
PHYSICAL TESTS					
Conductivity	2540	2470		3180	
Hardness	1420	1810		2230	
pH	6.7	6.93		7.15	
TSS	6050	10100		14300	
DISSOLVED ANIONS					
Alkalinity	542	492		573	
Cl	22.6	22.9		30.4	
SO4	1250	1000		1260	
NUTRIENTS					
NH4	0.51	0.54		1.68	
CYANIDES					
CN	0.012	0.012		0.078	
SCN	<0.5	<0.5		<0.5	
WAD-CN	0.005	0.006		0.014	
METALS	**DISSOL.**	**DISSOL.**	**TOTAL**	**DISSOL.**	**TOTAL**
Ag	<0.015	<0.01	<0.01	<0.01	<0.01
Al	<0.2	<0.2	118	<0.2	171
As	<0.2	0.149	1.32	12.6	8.2
B	0.13	0.2	0.2	0.4	0.3
Ba	0.034	0.04	0.97	0.05	1.57
Be	<0.005	<0.005	<0.005	<0.005	0.006
Bi	0.14	<0.1	0.2	<0.1	0.2
Ca	348	448	592	420	788
Cd	<0.01	<0.1	<0.01	<0.005	<0.005
Co	0.018	0.03	0.12	0.02	0.21
Cr	<0.015	<0.01	0.36	<0.01	0.52
Cu	0.074	<0.01	0.24	<0.01	0.53
Fe	0.888	<0.03	169	4.94	266
Hg	<0.00005				
K	4.1	5	30	11	46
Li	0.039	0.05	0.23	0.06	0.37
Mg	133	168	273	286	462
Mn	4.38	6.12	7.8	6.95	11.1
Mo	<0.03	<0.03	<0.03	<0.03	<0.03
Na	39.2	55	59	37	47
Ni	0.045	0.09	0.5	0.04	0.98
P	<0.3	<0.3	12.5	<0.3	27.2
Pb	<0.05	<0.05	0.07	<0.05	<0.05
Sb	<0.2	<0.2	<0.2	<0.2	<0.2
Se	<0.2	<0.2	<0.2	<0.2	<0.2
Si	8.86	11.2	107	10.5	97.2
Sn	<0.3	<0.03	<0.03	<0.03	<0.03
Sr	0.481	0.621	0.962	0.776	1.21
Ti	<0.01	<0.01	15	<0.01	16.7
Tl	<0.1	<0.1	<0.1	<0.1	0.2
V	<0.03	<0.03	0.35	<0.03	0.53
W	<0.1				
Zn	0.013	0.006	0.429	0.008	0.815

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9g: 1994 Pore Water Chemistry for Piezometer DK-93-3B

DATE	17-Feb	09-Mar	20-Apr	18-May	25-Jul	25-Aug	20-Sep	13-Dec
PHYSICAL TESTS								
Conductivity	2040	4240	4480	4290	4360	4540	4450	4550
Hardness	2680	2970	3400	3020	2870	2690	2560	3490
pH	6.60	7.03	6.72	6.73	6.92	6.67	6.74	6.48
TSS	238	150	179	394	396	97	265	715
DISSOLVED ANIONS								
Alkalinity	617	604	400	637	??	??	??	??
Cl	16.5	17.2	22.3	18.8	22.2	25.0	26.2	27.7
SO4	2350	2640	3050	2730	2750	2840	2740	3000
NUTRIENTS								
NH4	5.100	5.860	6.700	6.540	5.220	6.400	4.390	4.350
CYANIDES								
CN	0.032	0.096	0.077	0.122	0.140	0.110	0.124	0.109
SCN	5.4	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
WAD-CN	0.006	0.067	0.063	0.019	0.022	0.034	0.054	0.015
DISSOLVED METALS								
As	2.200	0.032	8.950	1.210	0.640	6.460	3.230	7.530
Cu	0.002	0.004	0.003	<0.001	<0.01	<0.001	0.002	0.050
Fe	7.310	0.908	9.250	7.050	7.120	9.380	10.500	22.300
Pb	<0.001	<0.001	<0.001	<0.001	<0.05	<0.001	<0.001	<0.001
Ni	0.042	0.047	0.023	0.023	<0.02	0.012	0.016	0.016
Zn	0.036	0.043	0.019	0.022	0.006	0.006	0.008	0.014
Ca	410	425	407	444	427	416	386	493
K	15.4	16.3	32.0	20.0	20.6			
Mg	368	484	580	464	438	400	386	549
Mn	32.00	27.00	15.40	25.40	20.00	15.90	16.80	20.60
Na	53.5	52.0	57.7	50.1	48.4	34.7	39.0	48.8
TOTAL METALS								
As	2.210	1.320	10.900	2.230	5.340	7.240	4.490	
Cu	0.042	0.004	0.003	0.001	<0.01	0.002	0.003	
Fe	11.300	15.000	14.000	12.400	17.300	12.800	14.200	
Pb	0.008	0.007	<0.001	0.001	<0.05	0.001	0.002	
Ni	0.042	0.054	0.029	0.024	0.024	0.013	0.020	
Zn	0.118	0.125	0.039	0.025	0.017	0.009	0.012	
Ca	417	437	407	464	427	418	386	
K	16.3	16.9	32.0	20.3	20.6			
Mg	388	464	580	490	440	400	401	
Mn	32.10	27.90	16.40	26.60	20.00	15.90	16.80	
Na	53.5	52.9	57.7	52.7	48.4	34.7	41.0	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9h: 1995-1997 Pore Water Chemistry for Piezometer DK-93-3B

DATE	22-Jun-95	Jun-96		25-Aug-97	
PHYSICAL TESTS					
Conductivity	4640	4540		4700	
Hardness	3790	3090		3390	
pH	6.63	7.13		6.83	
TSS	120	4370		2750	
DISSOLVED ANIONS					
Alkalinity	541	602		583	
Cl	43.2	49.6		51.5	
SO4	3100	2400		2910	
NUTRIENTS					
NH4	8.02	6.51		19.1	
CYANIDES					
CN	0.056	0.111		0.306	
SCN	0.5	<0.5		0.5	
WAD-CN	0.033	0.066		0.033	
METALS	**DISSOL.**	**DISSOL.**	**TOTAL**	**DISSOL.**	**TOTAL**
Ag	<0.015	<0.01	<0.01	<0.01	<0.01
Al	<0.2	<0.2	41.5	<0.2	11.2
As	12.9	2.63	7.94	9.5	14.6
B	0.85	0.6	0.6	0.8	0.8
Ba	0.082	0.08	0.26	0.06	0.16
Be	<0.005	<0.005	<0.005	<0.005	<0.005
Bi	0.18	0.2	0.3	<0.1	0.1
Ca	469	446	455	431	437
Cd	<0.01	<0.1	<0.01	<0.005	<0.005
Co	<0.015	0.01	0.04	<0.01	0.02
Cr	<0.015	<0.01	0.07	<0.01	0.02
Cu	<0.01	<0.01	0.16	<0.01	0.05
Fe	7.91	0.04	100	3.94	44.2
Hg	<0.00005				
K	36.2	19	27	35	37
Li	0.141	0.08	0.15	0.12	0.14
Mg	637	481	524	562	562
Mn	14.6	20.5	21.2	12.5	13.7
Mo	<0.03	<0.03	<0.03	<0.03	<0.03
Na	38.1	38	35	28	27
Ni	0.023	0.04	0.16	<0.02	0.07
P	<0.3	<0.3	0.5	<0.3	0.7
Pb	<0.05	<0.05	0.11	<0.05	0.06
Sb	<0.2	<0.2	<0.2	<0.2	<0.2
Se	<0.2	<0.2	<0.2	<0.2	<0.2
Si	8.23	7.26	47.3	8.59	20.5
Sn	<0.3	<0.03	<0.03	<0.03	<0.03
Sr	1.31	1.17	1.16	1.2	1.19
Ti	<0.01	<0.01	0.46	<0.01	0.19
Tl	<0.1	0.2	0.2	<0.1	0.1
V	<0.03	<0.03	0.17	<0.03	0.05
W	<0.1				
Zn	0.007	0.012	0.541	0.011	0.262

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9i: 1994 Pore Water Chemistry for Piezometer DK-93-4

DATE	17-Feb	10-Mar	20-Apr	18-May	25-Jul	24-Aug	20-Sep	12-Dec
PHYSICAL TESTS								
Conductivity	1030	2080	1990	1890	1840	1680	1720	1710
Hardness	1170	1160	1180	1100	999	974	886	1020
pH	6.82	7.08	7.02	6.95	7.14	6.85	7.10	6.80
TSS	3770	3380	1040	823	7100	313	201	550
DISSOLVED ANIONS								
Alkalinity	428	409	422	431	437	391	425	408
Cl	16.0	14..5	13.9	18.5	32.6	42.6	51.2	35.5
SO4	874	915	879	682	664	580	514	310
NUTRIENTS								
NH4	0.045	0.054	0.036	0.018	0.011	0.016	0.019	0.007
CYANIDES								
CN	0.018	0.044	0.029	0.028	0.052	0.089	0.081	0.025
SCN	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.6
WAD-CN	<0.005	0.020	0.010	0.017	0.036	0.081	0.042	0.008
DISSOLVED METALS								
As	0.009	0.008	0.011	0.007	<0.2	0.016	0.011	0.009
Cu	0.004	0.006	0.005	0.011	0.016	0.004	0.011	0.006
Fe	0.213	0.212	<0.03	0.033	0.087	0.037	0.046	0.116
Pb	<0.001	<0.001	<0.001	<0.011	<0.05	<0.001	<0.001	<0.001
Ni	0.113	0.064	0.086	0.088	0.071	0.054	0.080	0.025
Zn	0.014	0.025	0.008	0.008	0.006	<0.005	<0.005	<0.005
Ca	289	284	287	271	250	222	219	241
K	8.6	6.8	6.6	5.9	5.8			
Mg	106	108	112	103	90.8	77.8	82.3	102
Mn	1.09	0.65	0.57	0.55	0.45	0.29	0.44	0.13
Na	29.4	24.9	25.8	247.0	29.3	24.9	31.2	43.6
TOTAL METALS								
As	0.126	0.406	0.016	0.024	<0.2	0.035	0.015	
Cu	0.209	0.406	0.008	0.011	0.034	0.005	0.011	
Fe	48.700	75.400	1.290	2.570	4.390	0.236	0.515	
Pb	0.060	0.112	0.001	0.007	<0.05	0.001	0.001	
Ni	0.209	0.305	0.086	0.088	0.094	0.061	0.083	
Zn	0.273	0.446	0.031	0.020	0.029	0.006	0.006	
Ca	343	379	287	271	255	226	230	
K	16.8	21.0	7.1	6.1	6.5			
Mg	139	174	112	103	95.6	81.2	87.2	
Mn	1.99	2.20	0.57	0.56	0.53	0.29	0.46	
Na	36.6	30.0	25.8	24.7	29.3	28.5	31.2	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9j: 1995-1996 Pore Water Chemistry for Piezometer DK-93-4

DATE	20-Jun-95	Jun-96	
PHYSICAL TESTS			
Conductivity	1060	1200	
Hardness	606	596	
pH	7.16	7.31	
TSS	1600	127	
DISSOLVED ANIONS			
Alkalinity	353	361	
Cl	32.7	41.5	
SO4	214	243	
NUTRIENTS			
NH4	0.09	0.007	
CYANIDES			
CN	0.056	0.06	
SCN	0.8	<0.5	
WAD-CN	0.014	0.012	
METALS	DISSOL.	DISSOL.	TOTAL
Ag	<0.015	<0.01	<0.01
Al	<0.2	<0.2	0.4
As	<0.2	0.0093	0.0121
B	1.09	2.3	2.2
Ba	0.043	0.05	0.05
Be	<0.005	<0.005	<0.005
Bi	<0.1	<0.1	<0.1
Ca	151	160	150
Cd	<0.01	<0.1	<0.01
Co	0.037	0.03	0.03
Cr	<0.015	<0.01	<0.01
Cu	<0.01	<0.01	<0.01
Fe	0.477	0.06	0.73
Hg	<0.00005		
K	3.3	4	4
Li	0.056	0.04	0.04
Mg	55.8	48	45.7
Mn	0.158	0.114	0.121
Mo	<0.03	<0.03	<0.03
Na	48.6	44	41
Ni	0.032	0.03	0.03
P	<0.3	<0.3	<0.03
Pb	<0.05	<0.05	<0.05
Sb	<0.2	<0.2	<0.2
Se	<0.2	<0.2	<0.2
Si	8.81	7.93	8.09
Sn	<0.3	<0.03	<0.03
Sr	0.327	0.347	0.325
Ti	<0.01	<0.01	0.07
Tl	<0.1	<0.1	<0.1
V	<0.03	<0.03	<0.03
W	<0.1		
Zn	<0.005	<0.005	0.007

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9k: 1995 Pore Water Chemistry for Piezometer DK-93-5

DATE	20-Jun-95
PHYSICAL TESTS	
Conductivity	807
Hardness	434
pH	7.36
TSS	232
DISSOLVED ANIONS	
Alkalinity	258
Cl	35.2
SO4	138
NUTRIENTS	
NH4	0.24
CYANIDES	
CN	0.737
SCN	1
WAD-CN	0.066
METALS	**DISSOL.**
Ag	<0.015
Al	<0.2
As	<0.2
B	0.57
Ba	0.059
Be	<0.005
Bi	<0.1
Ca	115
Cd	<0.01
Co	0.027
Cr	<0.015
Cu	0.014
Fe	0.641
Hg	<0.00005
K	6.5
Li	0.026
Mg	35.5
Mn	0.388
Mo	<0.03
Na	29.5
Ni	<0.02
P	<0.3
Pb	<0.05
Sb	<0.2
Se	<0.2
Si	5.77
Sn	<0.3
Sr	0.226
Ti	<0.01
Tl	<0.1
V	<0.03
W	<0.1
Zn	<0.005

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9I: 1994 Pore Water Chemistry for Piezometer DK-93-6A

DATE	17-Feb	11-Mar	20-Apr	18-May	27-Jul	26-Aug	23-Sep	15-Dec
PHYSICAL TESTS								
Conductivity	604	1060	1110	1080	1090	1040	1120	1060
Hardness	223	225	244	319	224	224	225	223
pH	8.29	8.03	7.81	7.86	7.64	8.22	8.44	7.87
TSS	8440	4370	10300	2170	5000	1740	3320	3390
DISSOLVED ANIONS								
Alkalinity	146	175	145	212	130	147	184	168
Cl	142.0	127.0	129.0	123.0	137.0	137.0	127.0	138.0
SO4	169	163	180	148	200	173	150	179
NUTRIENTS								
NH4	6.100	6.990	7.170	6.400	6.750	6.350	5.460	6.780
CYANIDES								
CN	0.123	0.371	0.479	0.266	0.230	0.270	0.086	0.301
SCN	41.2	31.6	28.3	29.1	34.8	17.0	35.3	38.1
WAD-CN	0.033	0.033	0.078	0.032	0.034	0.058	0.047	0.034
DISSOLVED METALS								
As	8.020	7.630	8.870	5.050	7.680	7.410	6.800	9.260
Cu	0.001	<0.001	0.003	0.005	<0.01	<0.001	0.002	<0.001
Fe	0.109	0.109	0.039	1.340	<0.03	<0.03	<0.03	<0.03
Pb	0.001	<0.001	<0.001	0.009	<0.05	<0.001	<0.001	<0.001
Ni	0.009	0.012	0.019	0.007	<0.02	0.007	0.010	0.010
Zn	<0.005	0.007	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Ca	16.1	17.3	16.6	47.6	22.4	16	12.6	29.5
K	51.6	51.7	61.4	33.0	50.6			
Mg	44.3	44.1	49.2	48.6	40.7	44.6	47.1	36.2
Mn	0.02	0.02	0.03	0.11	0.01	0.01	0.01	0.01
Na	91.0	90.5	87.4	99.2	93.2	76.3	80.5	110.0
TOTAL METALS								
As	14.200	13.700	7.090	418.0	7.580	8.060	6.800	
Cu	0.384	0.324	0.005	0.012	<0.01	0.032	0.005	
Fe	197.00	152.00	2.26	2.63	1.31	10.20	1.27	
Pb	0.465	0.452	0.008	0.031	<0.05	0.090	0.012	
Ni	0.475	0.349	0.022	0.020	<0.02	0.031	0.018	
Zn	0.761	0.517	0.028	0.010	0.013	0.064	0.009	
Ca	181	171	17.3	102	22.8	29.7	13.1	
K	73.4	74.2	51.4	33.1	50.6			
Mg	125	120	49.2	188	40.7	50	47.1	
Mn	5.75	5.11	0.07	0.17	0.05	0.40	0.04	
Na	91.7	91.0	87.4	92.9	93.2	79.7	80.5	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9m: 1995-1997 Pore Water Chemistry for Piezometer DK-93-6A

DATE	23-Jun-95	Jun-96		26-Aug-97	
PHYSICAL TESTS					
Conductivity	1360	1110		1090	
Hardness	409	222		228	
pH	8.01	7.83		7.87	
TSS	30900	4400		14000	
DISSOLVED ANIONS					
Alkalinity	140	182		199	
Cl	161	134		125	
SO4	332	150		145	
NUTRIENTS					
NH4	8.81	7.23		8.05	
CYANIDES					
CN	0.46	0.228		0.565	
SCN	40.4	30.3		24.9	
WAD-CN	0.042	0.035		0.029	
METALS	**DISSOL.**	**DISSOL.**	**TOTAL**	**DISSOL.**	**TOTAL**
Ag	<0.015	<0.01	0.01	<0.01	<0.01
Al	<0.2	<0.2	346	<0.2	457
As	6.73	6.43	23.7	7.1	27.7
B	0.32	0.4	0.3	0.4	0.3
Ba	0.03	0.01	0.59	0.01	0.81
Be	<0.005	<0.005	<0.005	<0.005	<0.005
Bi	<0.1	<0.1	0.2	<0.1	0.3
Ca	68.9	22.7	606	18.4	620
Cd	<0.01	<0.01	<0.01	<0.005	0.005
Co	0.02	0.01	0.38	0.01	0.43
Cr	<0.015	<0.01	0.89	<0.01	1.17
Cu	0.017	<0.01	1.29	<0.01	1.51
Fe	0.046	<0.03	708	0.12	760
Hg	<0.00005				
K	61.2	47	100	47	119
Li	0.098	0.08	0.45	0.1	0.59
Mg	57.5	40.1	314	44.3	348
Mn	0.047	0.031	18	0.016	18.8
Mo	0.047	0.04	<0.03	0.03	0.03
Na	119	97	100	83	92
Ni	0.023	0.02	1.87	<0.02	2.01
P	<0.3	<0.3	2.4	<0.3	3
Pb	<0.05	<0.05	0.9	<0.05	0.98
Sb	<0.2	<0.2	<0.2	<0.2	<0.2
Se	<0.2	<0.2	<0.2	<0.2	<0.2
Si	1.41	1.28	224	1.45	263
Sn	<0.3	<0.03	<0.03	<0.03	<0.03
Sr	0.573	0.294	0.924	0.294	1.03
Ti	<0.01	<0.01	5.8	<0.01	7.71
Tl	<0.1	<0.1	<0.5	<0.1	0.2
V	<0.03	<0.03	1.17	<0.03	1.55
W	<0.1				
Zn	<0.005	0.007	2.1	<0.005	2.43

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9n: 1994 Pore Water Chemistry for Piezometer DK-93-6C

DATE	17-Feb	11-Mar	20-Apr	18-May	27-Jul	26-Aug	23-Sep	15-Dec
PHYSICAL TESTS								
Conductivity	573	630	821	855	905	902	940	800
Hardness	283	286	380	454	438	443	403	417
pH	7.47	7.97	7.49	7.53	7.50	7.25	7.49	7.26
TSS	23600	14400	2990	9760	8370	6010	3340	5700
DISSOLVED ANIONS								
Alkalinity	318	330	391	402	46	418	430	373
Cl	19.1	19.7	25.1	27.7	30.4	29.8	31.3	27.2
SO4	3.8	4.6	34.2	42.5	59.2	59.5	57.4	27.1
NUTRIENTS								
NH4	0.940	1.290	0.925	1.010	0.900	0.950	0.818	1.170
CYANIDES								
CN	0.010	0.006	0.012	0.013	<0.005	0.007	0.023	0.006
SCN	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.5
WAD-CN	<0.005	<0.005	<0.005	<0.005	<0.005	0.006	<0.005	<0.005
DISSOLVED METALS								
As	0.025	0.016	0.004	0.014	<0.2	0.029	0.071	176
Cu	0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Fe	0.059	0.043	0.041	0.797	<0.03	1.020	0.191	0.164
Pb	<0.001	<0.001	<0.001	<0.001	<0.05	<0.001	<0.001	<0.001
Ni	0.002	0.001	0.001	0.001	<0.02	0.001	0.001	<0.001
Zn	<0.005	0.010	<0.005	0.006	<0.005<	<0.005	<0.005	<0.005
Ca	76	76.7	102	123	121	125	112	113
K	6.2	5.4	5.4	5.9	6.5			
Mg	22.6	22.9	30.7	35.8	32.8	31.9	29.9	32.8
Mn	0.14	0.16	0.18	0.24	0.21	0.23	0.22	0.15
Na	17.3	17.2	17.3	19.3	19.6	17.0	17.2	20.8
TOTAL METALS								
As	0.033	0.081	0.010	0.227	<0.2	0.032	0.090	
Cu	0.067	0.113	<0.001	<0.001	<0.01	0.004	0.010	
Fe	52.20	73.70	0.60	2.15	0.96	3.56	3.91	
Pb	0.024	0.028	<0.001	0.001	<0.05	0.002	0.003	
Ni	0.058	0.107	0.006	0.002	<0.02	0.006	0.011	
Zn	0.179	0.266	0.019	0.007	<0.005	0.008	0.014	
Ca	140	200	102	123	121	144	122	
K	19.2	26.2	5.4	5.9	6.6			
Mg	58.9	94.4	30.7	35.8	33.4	37.9	34.9	
Mn	1.13	1.83	0.18	0.26	0.21	0.31	0.33	
Na	24.4	20.3	17.3	19.3	20.1	17.3	17.2	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9o: 1995-1997 Pore Water Chemistry for Piezometer DK-93-6C

DATE	23-Jun-95	Jun-96		26-Aug-97	
PHYSICAL TESTS					
Conductivity	726	705		736	
Hardness	365	154		353	
pH	7.49	7.57		7.47	
TSS	21700	2790		16700	
DISSOLVED ANIONS					
Alkalinity	367	352		369	
Cl	27.9	25.4		23.6	
SO4	13.7	9		22	
NUTRIENTS					
NH4	1.21	0.82		1.21	
CYANIDES					
CN	<0.005	<0.005		0.014	
SCN	0.8	<0.5		<0.5	
WAD-CN	<0.005	<0.005		<0.005	
METALS	**DISSOL.**	**DISSOL.**	**TOTAL**	**DISSOL.**	**TOTAL**
Ag	<0.015	<0.01	<0.01	<0.01	<0.01
Al	<0.2	<0.2	152	<0.2	112
As	<0.2	0.0053	0.0565	0.06	0.13
B	<0.1	<0.1	0.1	<0.1	<0.1
Ba	0.177	0.04	1.22	0.14	1
Be	<0.005	<0.005	<0.005	<0.005	<0.005
Bi	<0.1	<0.1	0.2	<0.1	0.1
Ca	97.4	25.4	389	96.5	349
Cd	<0.01	<0.01	<0.01	<0.01	<0.01
Co	<0.015	<0.01	0.09	<0.01	0.07
Cr	<0.015	<0.01	0.3	<0.01	0.22
Cu	<0.01	<0.01	0.2	<0.01	0.22
Fe	0.074	<0.03	188	0.18	157
Hg	<0.00005				
K	7.5	4	47	5	40
Li	0.017	0.01	0.27	0.01	0.22
Mg	29.6	22.1	178	27.3	159
Mn	0.07	0.084	3.82	0.225	3.3
Mo	<0.03	<0.03	<0.03	<0.03	<0.03
Na	21.3	18	36	18	26
Ni	<0.02	<0.02	0.22	<0.02	0.19
P	<0.3	<0.3	14.8	<0.3	13.3
Pb	<0.05	<0.05	0.09	<0.05	<0.05
Sb	<0.2	<0.2	<0.2	<0.2	<0.2
Se	<0.2	<0.2	<0.2	<0.2	<0.2
Si	11.5	1.64	93.2	11.6	65.2
Sn	<0.3	<0.03	<0.03	<0.03	<0.03
Sr	0.333	0.111	0.921	0.316	0.736
Ti	<0.01	<0.01	15.6	<0.01	12.1
Tl	<0.1	<0.1	<0.1	<0.1	0.1
V	<0.03	<0.03	0.38	<0.03	0.31
W	<0.1				
Zn	<0.005	0.022	0.517	<0.005	0.436

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9p: 1994-1995 Pore Water Chemistry for Piezometer DK-93-8

DATE	17-Feb	11-Mar	20-Apr	18-May	27-Jul	26-Aug	23-Sep	15-Dec
PHYSICAL TESTS								
Conductivity	1650	1710	1630	1670	1720	1730	1700	1790
Hardness	977	939	933	952	911	812	727	1080
pH	7.35	7.30	7.24	7.16	7.22	7.38	7.64	7.13
TSS	1780	1300	5620	1850	1620	161	2720	1340
DISSOLVED ANIONS								
Alkalinity	594	595	635	632	562	640	814	624
Cl	48.2	50.5	47.4	51.1	59.6	61.6	61.1	72.6
SO4	382	374	323	335	377	374	305	397
NUTRIENTS								
NH4	2.700	2.970	2.870	3.420	2.350	2.320	2.740	2.350
CYANIDES								
CN	0.152	0.079	0.182	0.137	0.100	0.130	0.091	0.182
SCN	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
WAD-CN	0.016	0.009	0.046	0.024	0.023	0.084	0.060	0.010
DISSOLVED METALS								
As	161	166	176	183	166	178	183	176
Cu	0.001	<0.001	0.004	<0.001	<0.01	0.001	0.002	0.001
Fe	0.562	0.722	2.320	0.258	0.265	0.567	0.545	0.181
Pb	0.001	<0.001	0.002	<0.001	<0.05	<0.001	<0.001	<0.001
Ni	0.007	0.013	0.011	0.005	<0.02	0.003	0.002	0.005
Zn	0.008	0.023	0.017	<0.005	0.007	<0.005	<0.005	<0.005
Ca	157	149	145	140	156	142	117	179
K	16.5	14.7	14.8	14.1	14.7			
Mg	142	138	139	142	126	111	106	153
Mn	0.14	0.11	0.15	0.08	0.07	0.06	0.06	0.07
Na	28.3	29.3	28.6	29.5	30.9	25.6	28.9	38.3
TOTAL METALS								
As	170	174	179	196	166	183	183	
Cu	0.095	0.081	0.048	0.001	0.019	0.003	0.007	
Fe	84.90	81.90	33.30	0.90	17.90	5.56	5.35	
Pb	0.063	0.059	0.039	0.002	<0.05	0.006	0.005	
Ni	0.228	0.220	0.079	0.005	0.042	0.010	0.013	
Zn	0.601	0.580	0.290	0.011	0.133	0.039	0.042	
Ca	197	211	163	141	172	165	130	
K	18.0	17.9	16.2	14.2	15.5			
Mg	173	181	153	144	138	135	127	
Mn	3.13	3.54	1.10	0.09	0.73	0.19	0.18	
Na	28.8	29.4	29.0	29.9	30.9	27.1	31.1	

pH in units, conductivity in umhos/cm, other parameters in ppm

DATE	20-Jun-95
PHYSICAL TESTS	
Conductivity	1930
Hardness	1220
pH	7.25
TSS	1370
DISSOLVED ANIONS	
Alkalinity	585
Cl	87.5
SO4	476
NUTRIENTS	
NH4	2.39
CYANIDES	
CN	0.12
SCN	0.5
WAD-CN	0.017
METALS	**DISSOL.**
Ag	<0.015
Al	<0.2
As	182
B	0.16
Ba	0.024
Be	<0.005
Bi	0.2
Ca	212
Cd	<0.01
Co	<0.015
Cr	<0.015
Cu	<0.01
Fe	4.11
Hg	<0.00005
K	19.1
Li	0.076
Mg	168
Mn	0.094
Mo	<0.03
Na	45.3
Ni	<0.02
P	<0.3
Pb	<0.05
Sb	<0.2
Se	<0.2
Si	8.48
Sn	<0.3
Sr	0.84
Ti	<0.01
Tl	<0.1
V	<0.03
W	<0.1
Zn	0.012

Table 9q: 1994 Pore Water Chemistry for Piezometer DK-93-9A

DATE	17-Feb	10-Mar	21-Apr	18-May	27-Jul	26-Aug	23-Sep	15-Dec
PHYSICAL TESTS					Standpipe has been damaged			
Conductivity	806	844	850	824				
Hardness	415	407	366	422				
pH	7.30	7.40	7.26	7.43				
TSS	13500	6400	840	471				
DISSOLVED ANIONS								
Alkalinity	479	468	468	461				
Cl	1.0	1.8	1.4	1.3				
SO4	6.2	6.1	2.4	<1				
NUTRIENTS								
NH4	1.150	1.450	1.080	0.996				
CYANIDES								
CN	0.026	0.015	0.023	0.016				
SCN	<0.5	<0.5	<0.5	<0.5				
WAD-CN	<0.005	<0.005	0.013	0.010				
DISSOLVED METALS								
As	3.88	3.53	0.468	0.125				
Cu	0.002	0.002	<0.001	0.002				
Fe	0.159	0.072	0.707	0.235				
Pb	0.001	<0.001	<0.001	<0.001				
Ni	0.003	0.002	<0.001	0.002				
Zn	<0.005	0.011	<0.005	<0.005				
Ca	105	104	96.8	108				
K	6.9	5.5	<2	5.9				
Mg	37.4	36.9	34.9	36.7				
Mn	0.17	0.20	0.16	0.14				
Na	18.8	19.7	18.1	19.1				
TOTAL METALS								
As	5.64	7.58	0.531	0.45				
Cu	0.080	0.040	<0.001	0.004				
Fe	59.80	33.00	2.39	3.02				
Pb	0.031	0.014	<0.001	0.001				
Ni	0.080	0.058	0.002	0.002				
Zn	0.193	0.123	0.013	0.006				
Ca	196	147	96.8	111				
K	19.5	12.2	5.2	5.9				
Mg	81.8	63.7	34.9	38				
Mn	1.29	0.73	0.17	0.15				
Na	27.8	23.5	18.1	19.1				

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 9r: 1994 Pore Water Chemistry for Piezometer DK-93-9B

DATE	17-Feb	10-Mar	21-Apr	18-May	27-Jul	26-Aug	23-Sep	15-Dec
PHYSICAL TESTS								Standpipe
Conductiv	1690	1860	1770	1760	3310	2640	2240	has
Hardness	1070	1170	1090	287	2070	1610	958	been
pH	7.32	7.32	7.13	7.19	7.46	7.50	7.97	damaged
TSS	7810	5460	2260	1880	1920	457	1840	
DISSOLVED ANIONS								
Alkalinity	898	947	981	958	692	814	795	
Cl	7.7	8.1	7.9	7.5	11.6	21.5	14.8	
SO4	96.9	113	79.5	43.7	1610	945	542	
NUTRIENTS								
NH4	7.100	8.230	7.500	7.960	2.770	3.680	3.340	
CYANIDES								
CN	0.142	0.288	0.082	0.056	0.170	0.130	0.744	
SCN	4.3	4.4	4.7	4.4	5.5	5.4	5.7	<0.5
WAD-CN	0.009	<0.009	0.030		0.014	0.050	0.028	<0.005
DISSOLVED METALS								
As	438	455	422	410	235	330	314	
Cu	0.003	0.001	0.002	<0.001	<0.01	<0.001	<0.001	
Fe	2.170	1.960	2.430	5.080	4.370	2.170	0.586	
Pb	<0.001	<0.001	<0.001	0.007	<0.05	<0.001	<0.001	
Ni	0.012	0.012	0.002	0.003	<0.02	0.007	0.012	
Zn	0.006	0.030	<0.005	0.029	<0.005	<0.005	<0.005	
Ca	81.5	107	102	19	118	86.7	60.4	
K	36.8	32.0	29.9	32.0	54.7			
Mg	210	219	202	58.2	430	314	195	
Mn	0.28	0.26	0.21	0.17	0.67	0.28	0.21	
Na	25.1	24.0	21.0	22.2	98.4	65.1	52.8	
TOTAL METALS								
As	448	455	431	433	235	334	314	
Cu	0.003	0.264	0.003	0.019	0.023	0.010	0.044	
Fe	92.50	107.00	4.05	14.50	21.70	11.30	32.40	
Pb	<0.001	0.129	0.003	0.020	<0.05	0.022	0.080	
Ni	0.183	0.206	0.016	0.034	0.043	0.019	0.062	
Zn	0.870	1.020	0.030	0.120	0.162	0.109	0.340	
Ca	125	173	102	109	131	100	94.6	
K	37.0	36.6	29.9	33.3	54.7			
Mg	234	250	202	191	430	326	229	
Mn	2.83	4.43	0.24	0.51	1.56	0.68	1.81	
Na	25.2	24.0	21.0	23.4	98.4	65.1	57.2	

pH in units, conductivity in umhos/cm, other parameters in ppm

Table 10: GOLDCORP Piezometer Water Levels.

PIEZO	Coordinates		Top of Standpipe	Stick-up	1994 WATER LEVELS, m								
	Northing	Easting	Elev., m	m	16-Feb	8-Mar	21-Apr	18-May	29-Jul	26-Aug	22-Sep	14-Dec	Average
DK-93-6C	6727.346	10027.870	364.497	0.762	361.747	361.577	361.697	362.267	362.297	362.377	362.397	362.557	362.115
DK93-6A	5706.485	10026.940	365.434	1.600	361.684	361.827	361.907	362.237	362.687	362.567	362.307	362.607	362.228
DK93-2	3893.737	9279.578	371.820	1.684	368.120	367.940	368.400	369.550	369.380	369.490	369.400	369.090	368.921
DK-93-3A	3068.236	8400.963	364.857	1.143	361.707	361.567	361.757	362.817	362.257	362.067	361.827	362.237	362.030
DK-93-3B	3048.758	8398.045	364.737	1.143	361.947	361.827	361.907	362.236	362.687	362.567	362.307	362.607	362.261
DK-93-9A	703.876	5356.116	359.980	1.397	356.510	356.470	356.440	356.640					356.515
DK-93-9B	694.318	5350.660	359.462	0.940	356.162	356.002	355.852	356.262	356.642	356.682	356.812		356.345
DK-93-8	1714.319	3135.488	360.729	1.321	360.609	357.369	357.589	358.559	358.259	358.379	358.479	358.599	358.480
DK-93-1 (utm)			380.180	1.180	377.460	377.450	377.530	377.210	377.320	377.450	377.250	377.070	377.343
DK-93-4	4824.780	7383.370	364.450	1.390	359.200	359.020	359.030	359.260	360.440	360.200	360.300	359.970	359.678
DK-93-5	4609.470	7115.250	368.840	1.460	dry	dry	dry	dry	dry	dry	dry	dry	dry

Table 11: GOLDCORP Sequential Extraction Data

			S1	S2	S3	S4	S6	S7
			New Tailings 1987-present	Primary Pond Mouth of Gabion Weir	Primary Pond Below Water Sampling Stn	Old Tailings 1983-1987	Abandoned Tailings @ Hwy 125	Old Tailings 1965-83
Set-Up								
	pH		8.19	8.5	8.42	7.46	8.42	7.74
	Alkalinity	$mg.L^{-1}$	70	65	135	55	180	70
	Cond.	mS/cm	7.9	8.5	5.1	>999	4.9	10.6
	Temp.	°C	25.5	25.6	25.1	25.3	25.2	25.3
A: Acetic Acid								
Initial	pH		3.04	3.17	3.36	2.86	3.32	3.11
	Acidity	$mg.L^{-1}$	91,000	85,000	71,000	90,000	84,000	83,000
6 h	pH		3.52	3.49	3.86	3.49	3.83	3.44
	Acidity	$mg.L^{-1}$	92,000	91,000	69,000	91,000	75,000	83,000
Wash	pH		4.11	4.2	4.34	4.03	4.29	3.98
	Acidity	$mg.L^{-1}$	9,500	10,500	11,000	10,000	9,000	11,000
	Initial wt	g	125	125	125	125	125	125
	Final wt	g	117.5	115.32	115.49	114.16	114.1	119.14
	Wt Loss	%	6.0%	7.7%	7.6%	8.7%	8.7%	4.7%
B: HCL (35%)								
Initial	pH		0	0	0	0	0	0
	Acidity	$mg.L^{-1}$	62,000	52,500	51,500	61,500	42,500	58,000
6 h	pH		0.1	0.2	0.4	0.1	0.2	0.1
	Acidity	$mg.L^{-1}$	52,500	46,500	48,500	54,000	34,500	47,000
Wash	pH		1	1.3	1.4	1.1	1.5	1.4
	Acidity	$mg.L^{-1}$	7,000	6,000	7,000	5,500	3,000	4,500
	Initial wt	g	117.5	115.43	115.49	114.16	114.1	119.14
	Final wt	g	96.59	93.1	92.55	93.07	87.58	99.91
	Wt Loss	%	16.7%	17.9%	18.4%	16.9%	21.2%	15.4%
C: Cold HNO3 (73%)								
Initial	pH		0.15	0.1	0.1	0.1	0.1	0.2
	Acidity	$mg.L^{-1}$	94,000	88,500	87,500	94,000	96,500	95,500
6 h	pH		0.49	0.45	0.49	0.43	0.3	0.6
	Acidity	$mg.L^{-1}$	85,000	80,000	79,000	87,000	88,000	84,000
Wash	pH		0.4	0.9	0.88	0.91	0.85	0.55
	Acidity	$mg.L^{-1}$	7,000	6,500	9,000	8,500	8,500	7,500
	Initial wt	g	96.59	93.1	92.55	93.07	87.58	99.91
	Final wt	g	85.54	78.32	77.71	80.28	79	87.94
	Wt Loss	%	8.8%	11.8%	11.9%	10.2%	6.9%	9.6%
D: Hot HNO3 (73%)								
Initial	pH		0.15	0.1	0.15	0.1	0.1	0.25
	Acidity	$mg.L^{-1}$	83,000	91,500	84,000	81,000	100,000	87,000
6 h	pH		0.49	0.45	0.49	0.43	0.3	0.6
	Acidity	$mg.L^{-1}$	112,000	124,000	143,000	107,500	101,000	102,000
Wash	pH		0.7	0.5	0.6	0.7	0.55	0.9
	Acidity	$mg.L^{-1}$	15,000	21,500	15,500	11,000	20,000	8,500
	Initial wt	g	85.54	78.32	77.71	80.28	79	87.94
	Final wt	g	75.3	69.19	69.53	71.49	68.5	76.36
	Wt Loss	%	8.2%	7.3%	6.5%	7.0%	8.4%	9.3%

Table 12a

The geochemistry of gold-bearing chemical sediments, Dickenson Mine, Red Lake, Ontario: a reconnaissance study

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Auriferous sedimentary rocks at the Dickenson Mine, Red Lake, are characterized by large enrichments of rare elements (Au, Ag, Pd, As, Sb, B) and metals conventionally considered to be relatively immobile (Ni, W), with negligible concentrations of the abundant and mobile base metals. Based on a reconnaissance survey involving 12 rocks analysed for 42 elements, these sediments can be represented in terms of a mixture of two components—mafic volcaniclastic material and hydrothermal precipitates. The volcaniclastic component has a composition closely comparable to mafic volcanic rocks that conformably envelope the sediments, and contributes all of the Al, Ti, V, Sc, Zr, and F. The hydrothermal component donates Si, Fe, Mn, Mg, Ca, K, Si, and C, together with Au, Ag, Pd, As, Sb, B, W, and Ni. Chromium and nickel are contributed from both sources. Alteration of the mafic volcanic rocks that envelope the auriferous sediments involved fixation of Si, K, and CO_2, in contrast to the typical patterns of metasomatism in footwall rocks to base metal massive sulphide deposits, which are characterized by Mg, Fe (Si, S) addition.

Quartz isolated from chert within the auriferous sediments and volcanic rocks has a maximum $\delta^{18}O$ of 19‰, which is within the range of δ values for marine cherts of ~2.8 Ga, and signifies precipitation in equilibrium with ambient marine water of ~−1‰ at ~70–90°C. Mafic volcanic wall rocks to the sediments have whole-rock $\delta^{18}O$ values of 16–17‰ and $\Delta_{quartz-chlorite} \leq 2‰$. The anomalously heavy whole-rock isotopic composition and small quartz–chlorite fractionation may result from the growth of chlorite from precursors such as zeolites and smectite. Alternatively, chlorite or its precursors may have become enriched in ^{18}O by isotope exchange with metamorphic hydrothermal fluids that become ponded when discharge to the hydrosphere was capped by overlying basalts.

Rare earth element (REE) distributions in sediments and mafic volcanic wall rocks are characterized by relatively flat normalized patterns up to Sm–Gd, followed by an abrupt continuous decline in abundance. This implies modification of the primary tholeiitic abundances by hydrothermal solutions capable of mobilizing heavy REE's relative to light REE's. The environment that would satisfy the geological observations and chemical data is one of metamorphic hydrothermal fluids emmanating onto the sea floor during a period of relatively quiescent mafic submarine volcanism, with subsequent capping of the discharge during emplacement of the overlying pillow basalts.

Table 12b Abundances of major and selected trace elements in gold-bearing sediments and host rocks, Dickenson Mine, together with element abundances in primary basalts (major element oxides in weight percent, trace elements in parts per million)

					Fe	HOST	HOST		Fe		Primary[a]
	967	968	969	970	KMI	971	972	973	974	974I	basalt
SiO_2	64.31	50.24	63.71	56.90	49.94	60.72	60.52	52.89	38.91	57.92	49.8
TiO_2	0.31	0.85	0.15	0.62	0.33	0.93	0.87	0.92	0.28	0.55	2.3
Al_2O_3	5.18	13.23	3.31	9.24	5.63	14.62	13.39	14.82	1.29	8.78	14.7
Fe_2O_3	13.51	13.29	13.33	11.08	24.24	10.69	10.34	11.01	28.63	17.03	12.4
MnO	0.08	0.27	0.16	0.27	0.14	0.23	0.23	0.20	0.28	0.22	0.2
MgO	1.48	3.86	2.65	4.12	1.85	3.30	4.92	5.01	5.18	2.93	7.6
CaO	2.19	8.21	4.73	6.85	2.61	4.38	6.66	8.90	10.36	4.86	10.6
K_2O	0.72	2.07	0.45	1.30	0.43	1.90	1.88	0.68	0.07	1.19	1.0
Na_2O	0.19	0.41	0.11	0.43	0.08	0.26	0.43	0.56	0.10	0.15	2.4
P_2O_5	—	0.02	—	—	—	0.04	0.01	0.06	0.02	0.02	0.2
LOI[b]	10.87	7.83	10.19	7.84	17.94	3.61	1.31	6.15	17.96	7.27	
Total	98.84	100.28	98.79	87.65	103.19	100.68	100.56	101.20	103.08	100.02	
Au	15	0.61	26	49	70	0.12	0.15	1.4	2.45	5.0	0.002[c]
Ag	4.1	5.4	3.1	3.1	6.2	3.1	3.0	6.2	5.2	2.1	0.11[d]
Pd	0.045	0.284	0.160	0.217	0.240			0.180	0.369		0.008
Li	32	36	18		36	92	98	51	10	35	17
Be	0.1	0.3	0.1	0.3	—	0.4	0.4	0.1	—	—	1
B	150	100	200		35	80	45	15	70	90	5
F	120	428	68	220	76	372	348	148	20	124	400
Cl	150	—	200	200	200	100	—	100	100	100	60
S (%)	5.4	4.0	9.3	7.6	5.70	0.7	0.20	0.20	9.90	5.40	0.03
As	16 000	280	370	145	8400	210	1	1	380	105	2
Sb	200	320	20	20	280	36	18	20	20	20	0.2
Sc	16	42	14		15	45	42	42	34	23	30
V	94	240	57	198	123	270	264	270	98	146	250
Cr	116	209	81	152	149	222	207	231	131	181	170
Co	99	30	43	28	241	44	15	106	44	88	48
Ni	192	127	138	177	456	84	89	103	914	624	130
Cu	47	123	11	93	28	109	107	109	27	201	87
Zn	1390	72	79	4720	76	66	71	81	5076	87	105
Mo	—	—	—	—	—	—	—	—	—	—	1.5
Sn	—	—	—	—	—	—	—	—	3	3	1.5

Table 12c

(Concluded)

	967	968	969	970	KMI	971	972	973	974	974I	Primary[a] basalt
W	—	20	—	—	60	90	20	70	—	40	0.4[e]
Hg	1.2	0.092	0.36	6.76	0.36	0.39	0.25	0.24	0.22	0.25	0.09
Pb	—	—	—	—	—	—	—	—	—	—	6
Th	—	—	—	—	—	—	—	—	—	—	4
U	0.2	—	—	—	—	0.2	—	—	—	—	1
Rb	24	51	19	36	13	50	47	29	10	32	30
Sr	16	81	35	58	16	74	78	75	11	34	465
Y	6	14	1	11	2	13	7	6	25	2	21
Zr	23	50	14	35	12	61	58	61	24	30	140
Nb	—	—	—	—	0.3	—	—	—	—	—	19
Cd	—	—	—	—	—	—	—	—	—	—	0.22
Ba	52	106	26	178	31	224	233	105	7	182	330
sg[f]	2.78	2.84	2.95	2.87	3.64	2.90	2.83	2.93	3.35	2.98	
Al/Ti	14.7	13.7	19.5	13.2	15.1	13.9	13.6	14.2	4.1	14.1	
Al/V	290	290	310	250	240	290	270	290	70	320	
Al/Sc	1710	1670	1250		1990	1720	1690	1870	200	2020	
Al/Y	4570	5000	17 510	4440	14 890	5950	10 120	13 070	270	23 220	
Al/Zr	1190	1400	1250	1400	2480	1270	1220	1280	280	1550	
Fe^{2+}[g]						7.58	7.48				
$Fe^{2+}/\Sigma Fe$						0.71	0.72				

[a]Data from Turekian and Wedepohl (1961).
[b]LOI = weight percent loss on ignition at 1100°C.
[c]Data from Parthe and Crocket (1972) and Kwong and Crocket (1978).
[d]Data from Frueh and Vincent (1972).
[e]Data from Helsen *et al.* (1978).
[f]sg = specific gravity ($2\sigma = \pm 0.01$).
[g]Fe^{2+} expressed as Fe_2O_3.

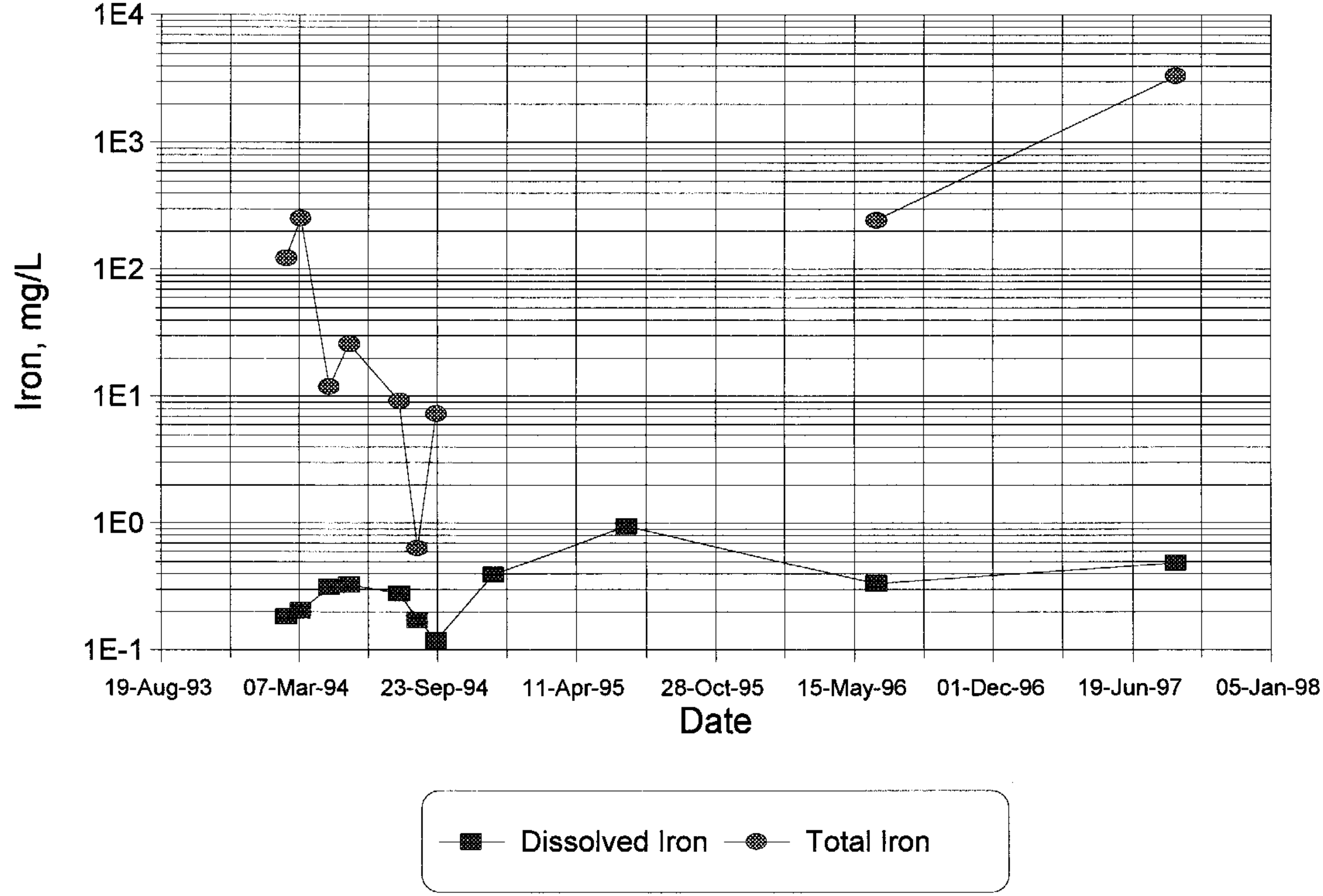
Table 13: Ratios on LOI (from R. Kerrick, 1981)

Sample #	CaO	MgO	Type	CaO:MgO	LOI 1100°	Fe_2O_3 %	S %	Calculated LOI:Fe_2O_3	LOI:S	Calculated FeS_2 from Fe_2O_3	FeS_2 from S
967	2.19	1.48	Au Bearing	1.48	10.87	13.51	5.4	0.80	2.01	20.3	10.2
968	8.21	3.86	Sediments	2.13	7.83	13.29	4.0	0.59	1.96	19.9	7.4
969	4.73	2.65	Sediments	1.78	10.19	13.33	9.3	0.73	1.10	20.0	17.3
970	6.85	4.12	Sediments	1.66	7.84	11.08	7.6	0.71	1.03	16.6	14.1
973	8.90	5.01	Sediments	1.78	6.15	11.01	? 0.20	0.56	30.8	16.5	? 0.4
974I	4.86	2.93	Sediments	1.66	7.27	17.03	5.40	0.43	1.35	25.5	10.0
KM1	2.61	1.85	Fe Form	1.41	17.94	24.24	5.70	0.74	3.15	36.4	10.6
974	10.36	5.18	Fe Form	2.00	17.96	28.63	9.90	0.63	1.81	42.9	18.4
971	4.38	3.30	HOST	1.33	3.61	10.69	0.7	0.34	5.16	16.0	1.3
972	6.66	4.92	HOST	1.35	1.31	10.34	0.20	0.13	6.55	15.51	0.4

APPENDIX B

LIST OF FIGURES

Fig 1: Piezometer DK-93-2
Total and Dissolved Iron



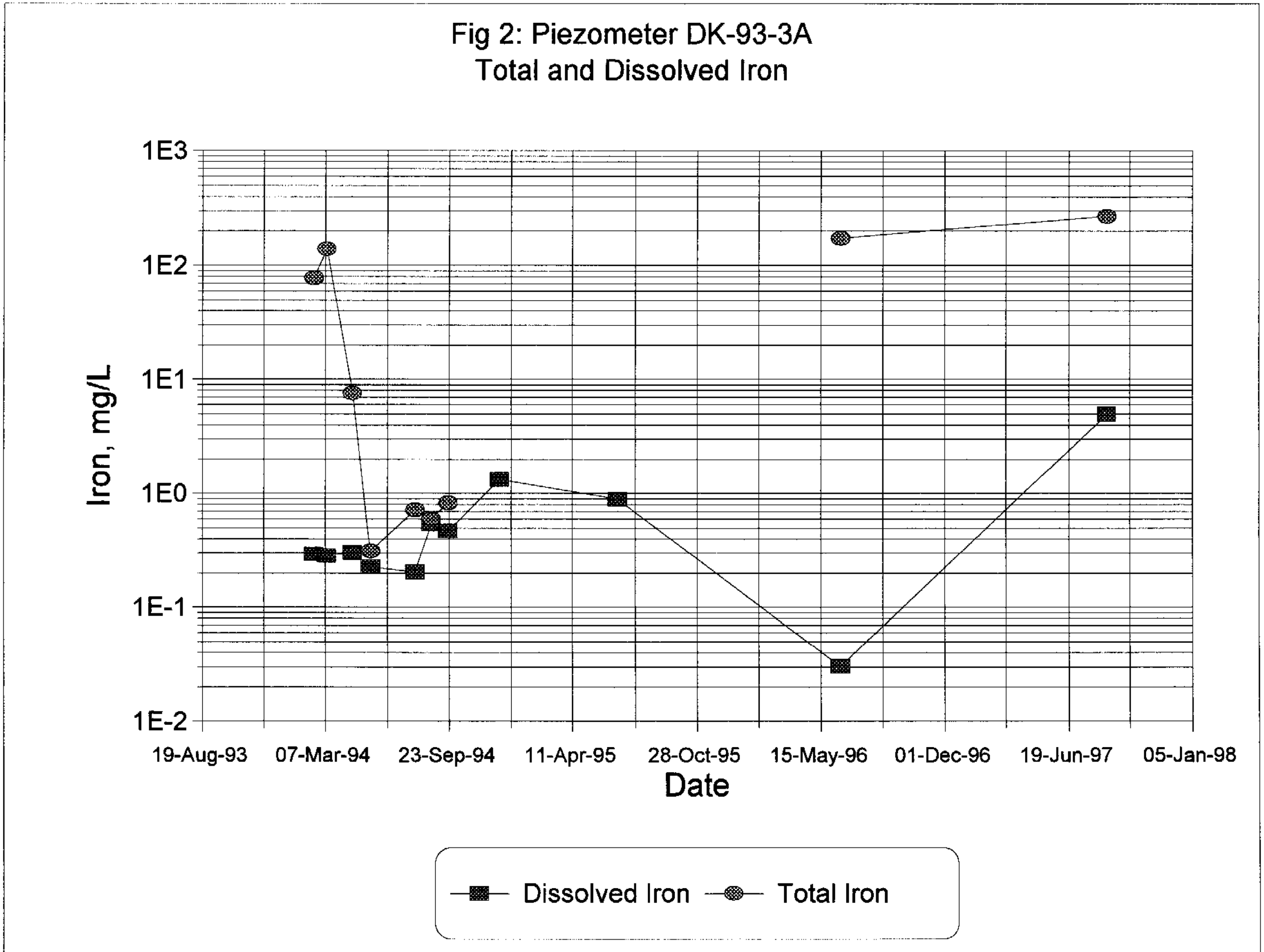


Fig 2: Piezometer DK-93-3A
Total and Dissolved Iron
1E3
1E2
1E1
1E0
1E-1
1E-2
Iron, mg/L
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Iron
Total Iron

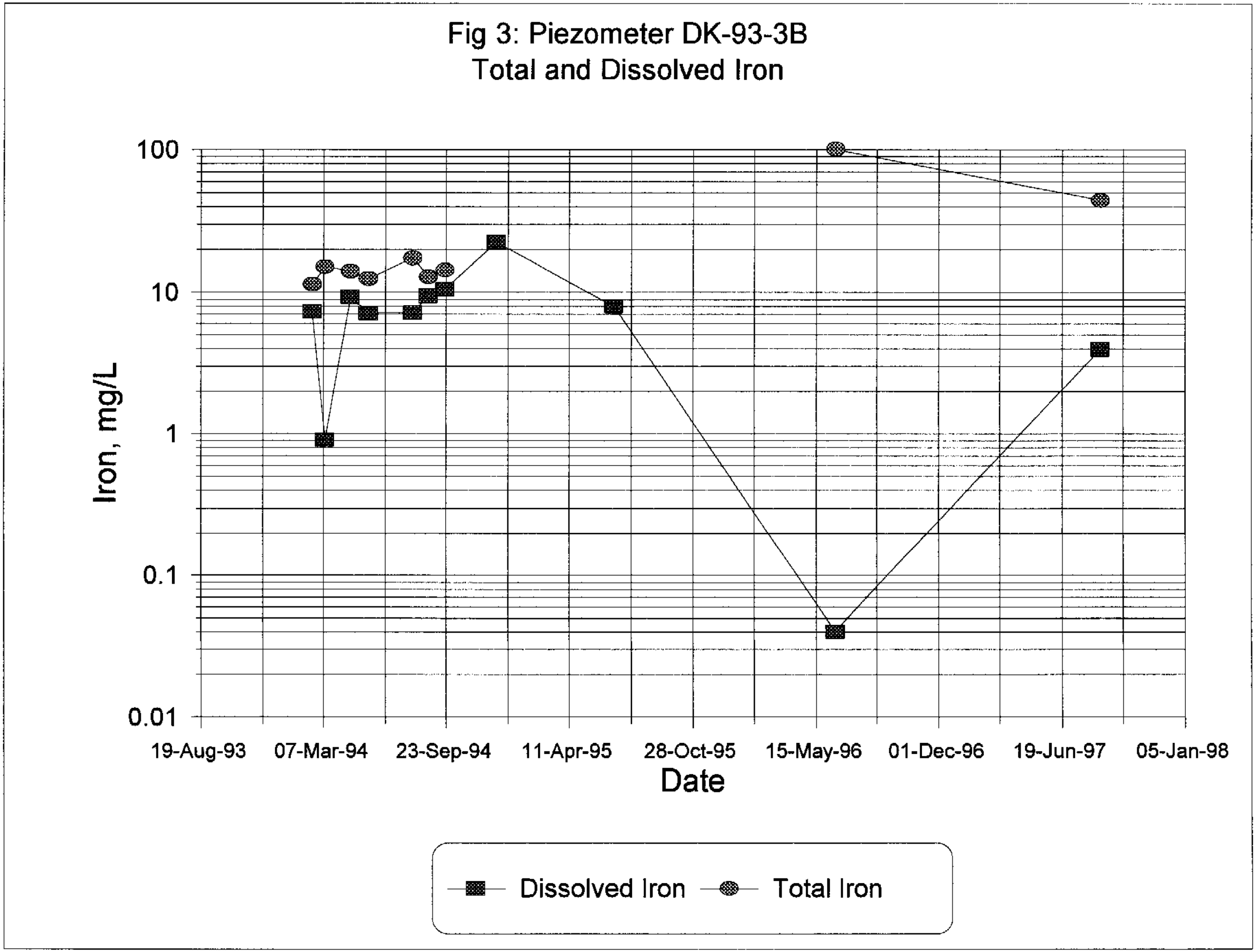


Fig 3: Piezometer DK-93-3B
Total and Dissolved Iron
100
10
1
0.1
0.01
Iron, mg/L
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Iron
Total Iron

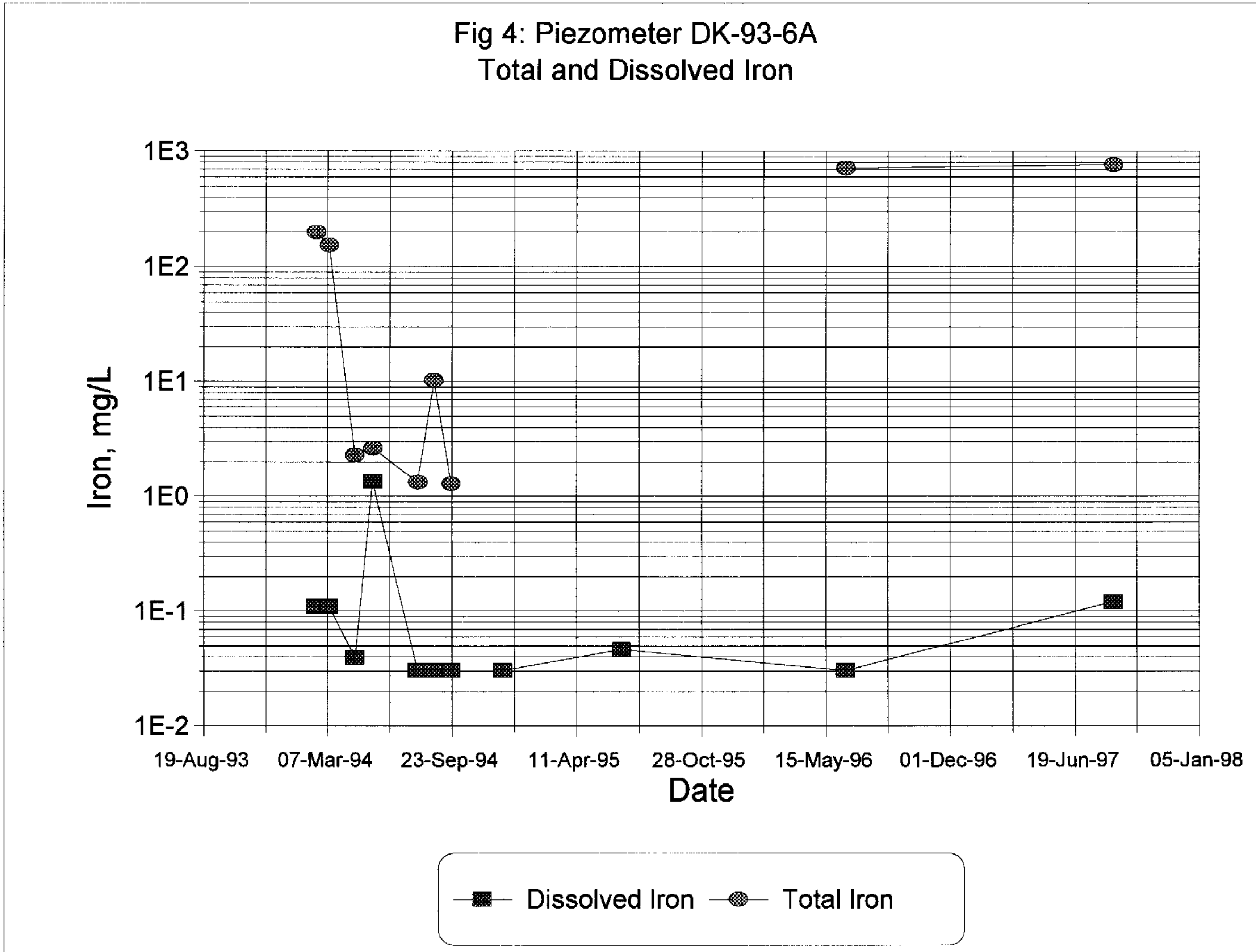
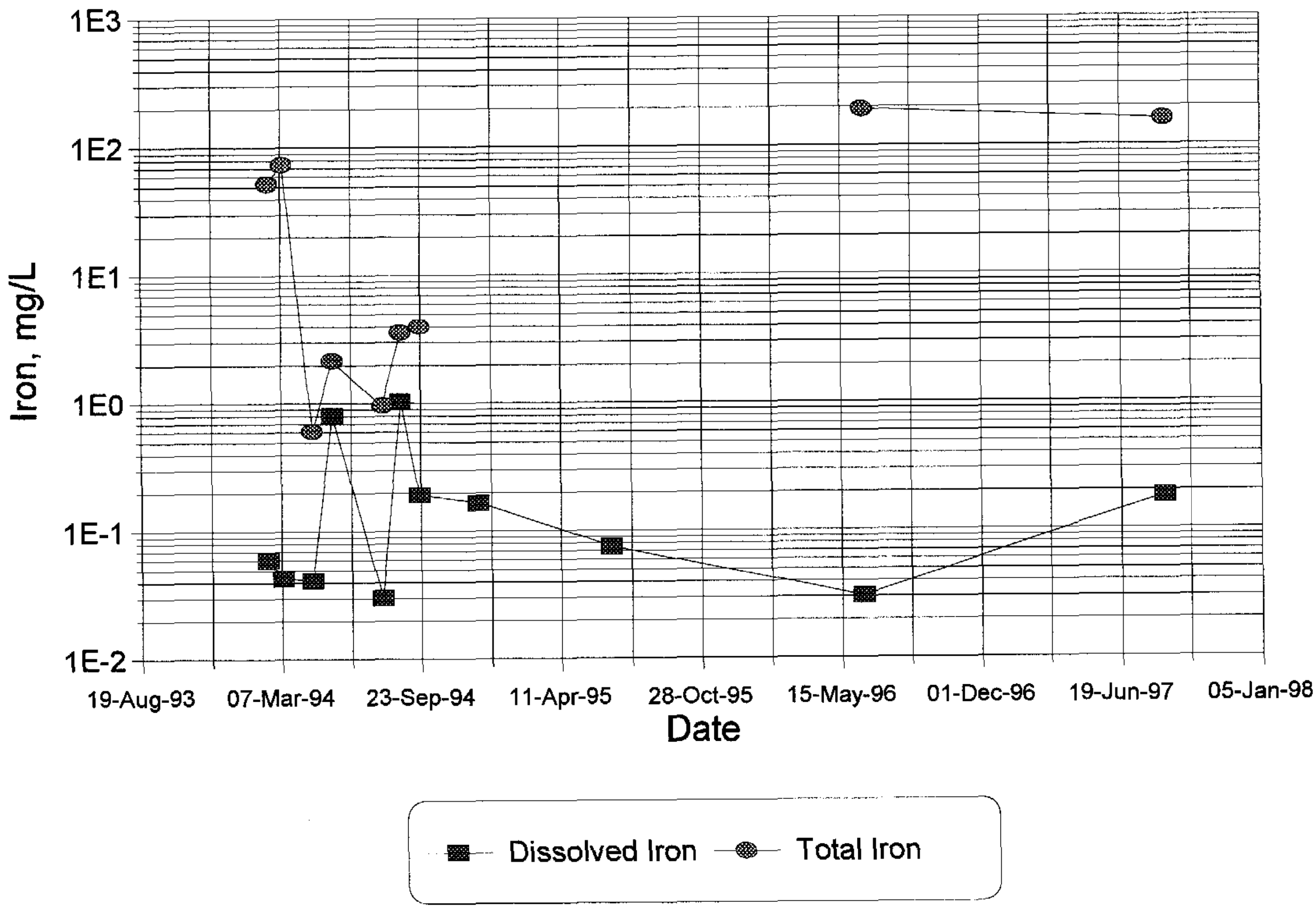


Fig 4: Piezometer DK-93-6A
Total and Dissolved Iron
Iron, mg/L
1E3
1E2
1E1
1E0
1E-1
1E-2
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Iron
Total Iron

Fig 5: Piezometer DK-93-6C
Total and Dissolved Iron



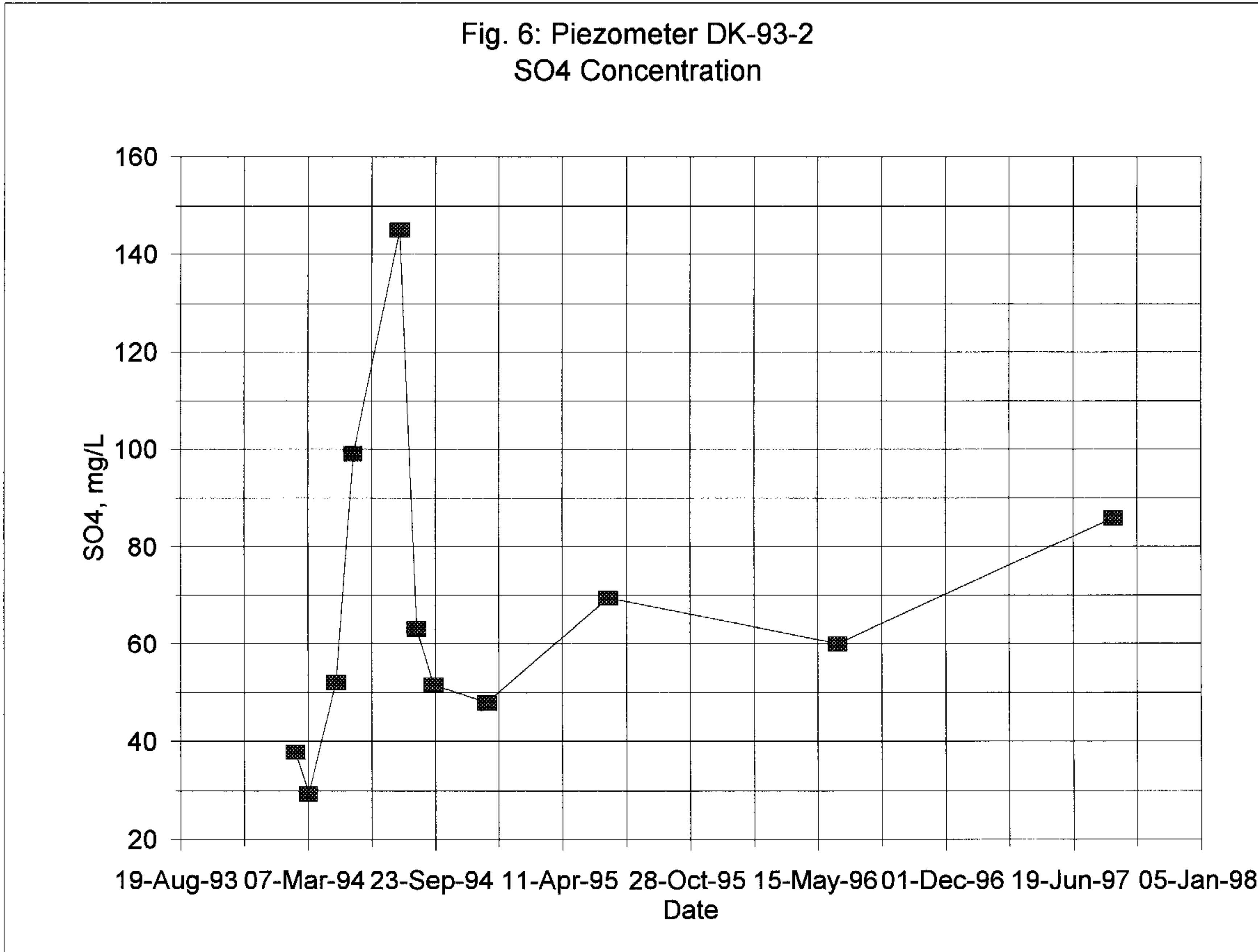


Fig. 6: Piezometer DK-93-2
SO4 Concentration
160
140
120
100
80
60
40
20
SO4, mg/L
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date

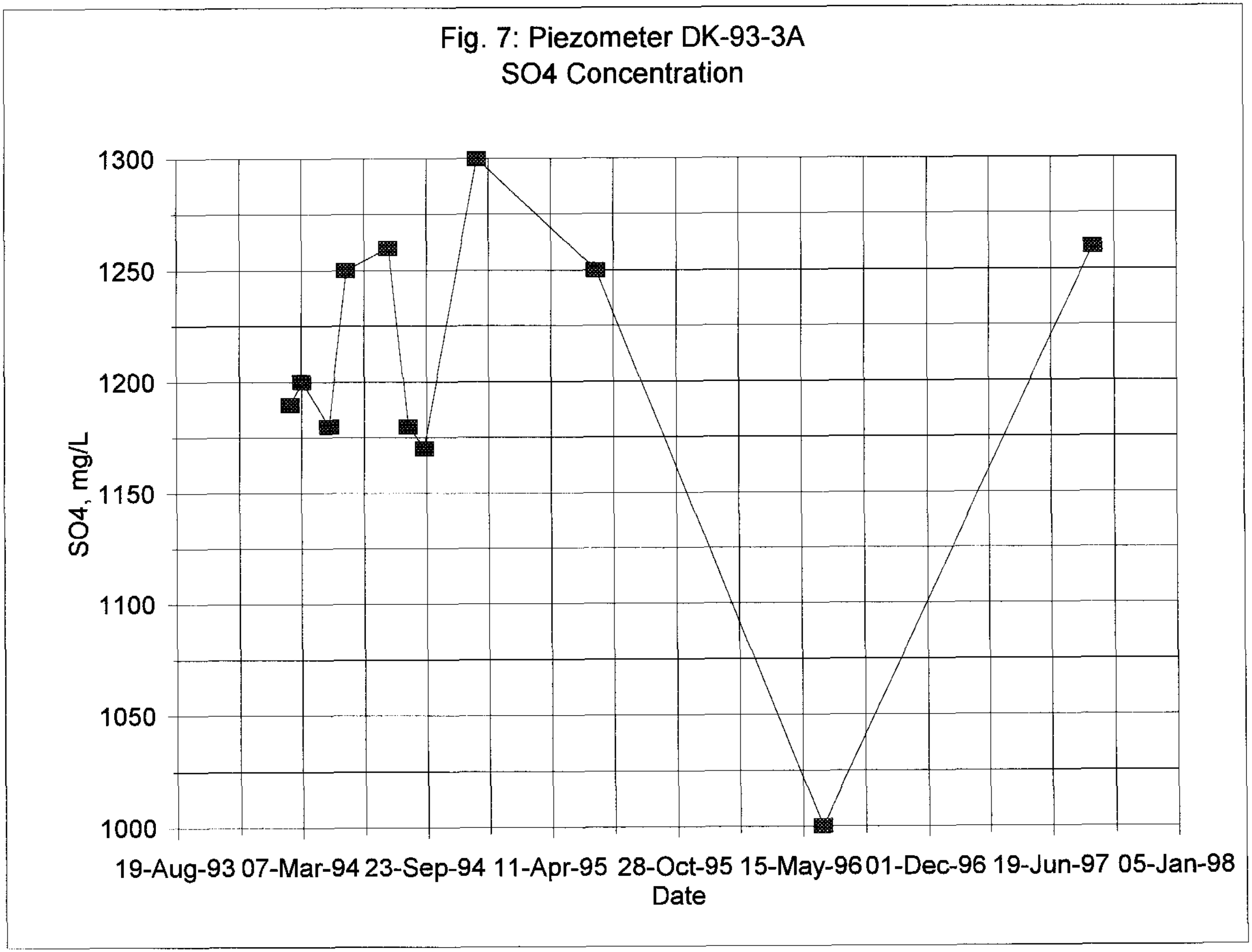
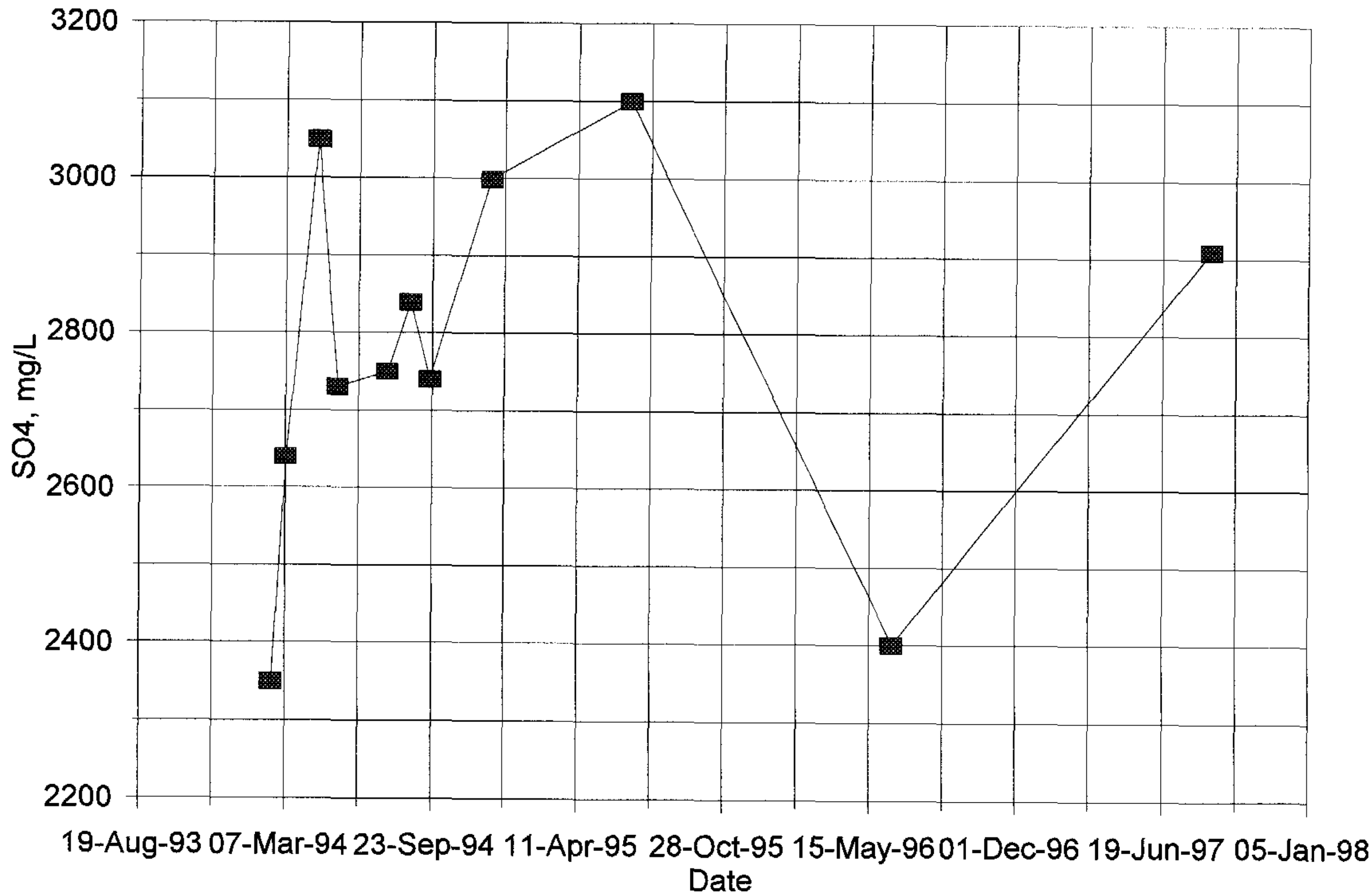


Fig. 7: Piezometer DK-93-3A
SO4 Concentration
SO4, mg/L
1300
1250
1200
1150
1100
1050
1000
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date

Fig. 8: Piezometer DK-93-3B
SO4 Concentration



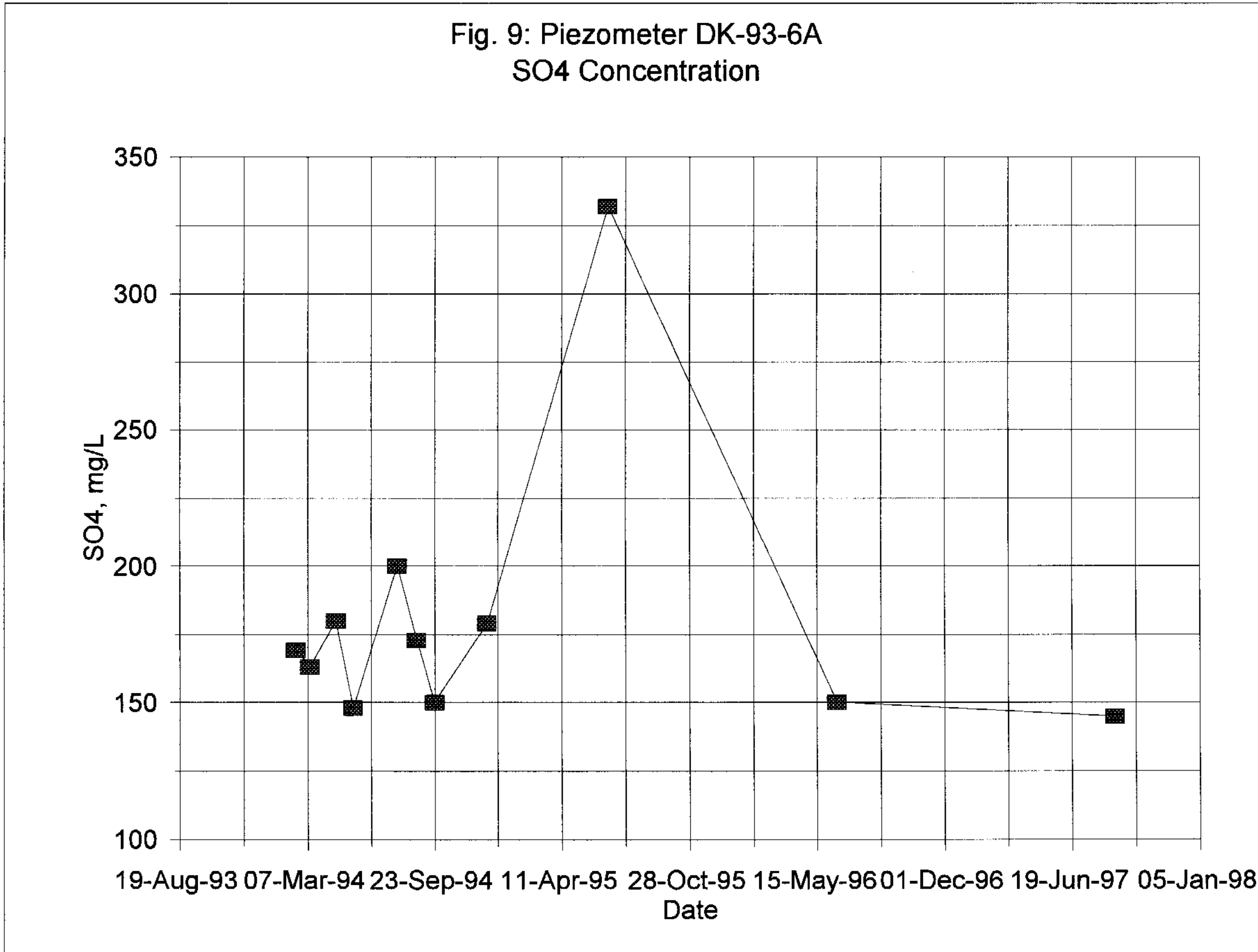


Fig. 9: Piezometer DK-93-6A
SO4 Concentration
350
300
250
200
150
100
SO4, mg/L
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date

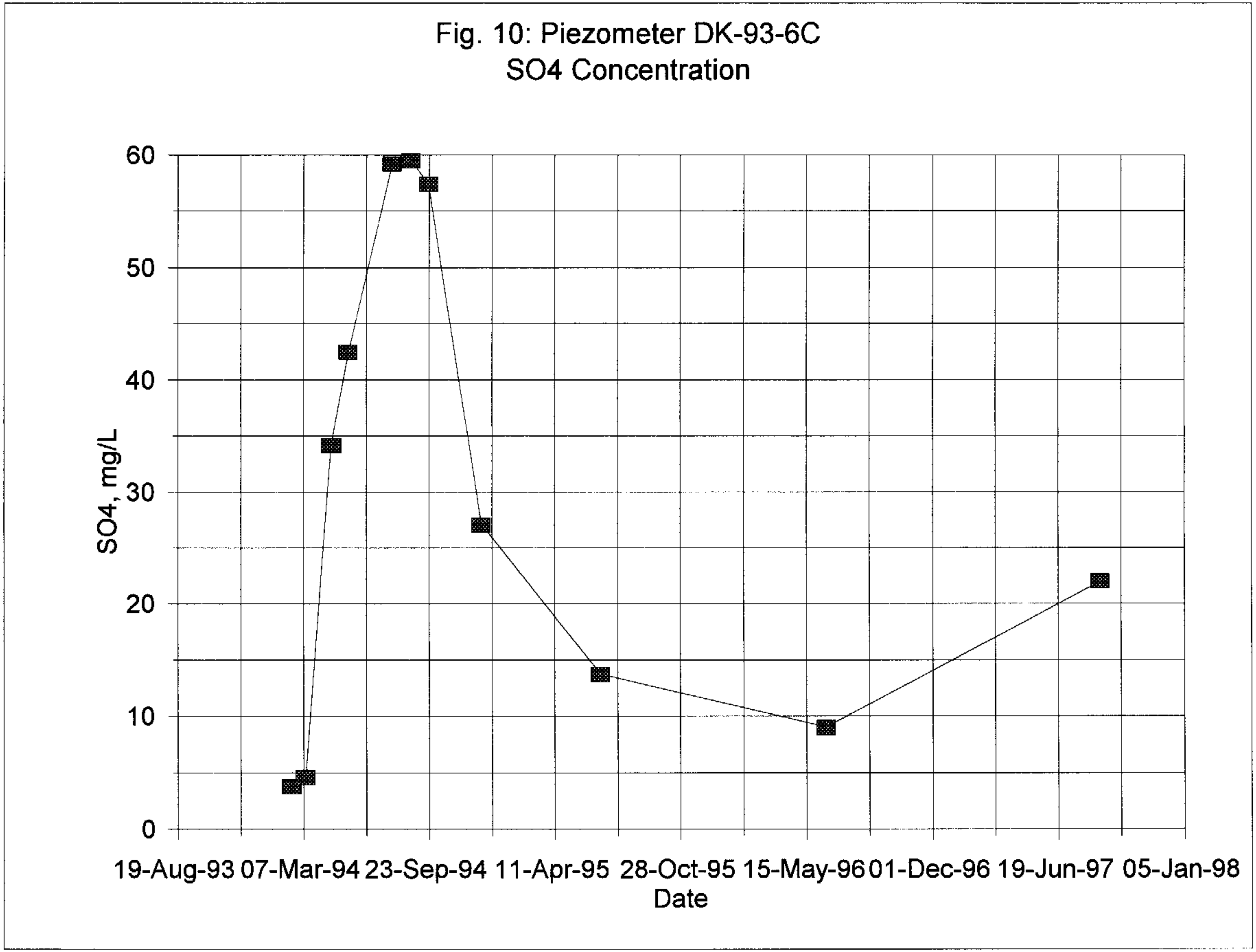


Fig. 10: Piezometer DK-93-6C
SO4 Concentration
SO4, mg/L
60
50
40
30
20
10
0
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date

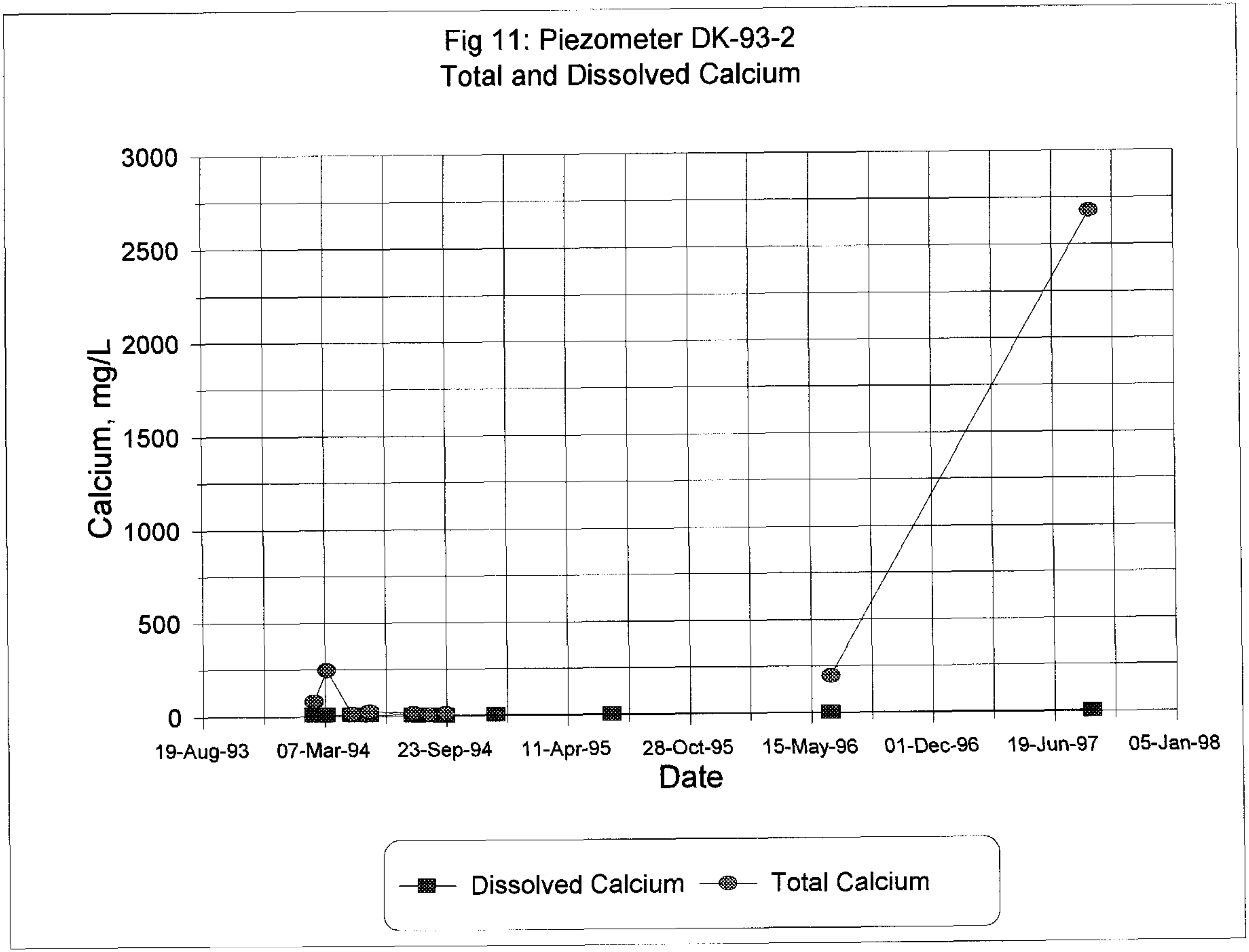


Fig 11: Piezometer DK-93-2
Total and Dissolved Calcium
Calcium, mg/L
3000
2500
2000
1500
1000
500
0
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Calcium
Total Calcium

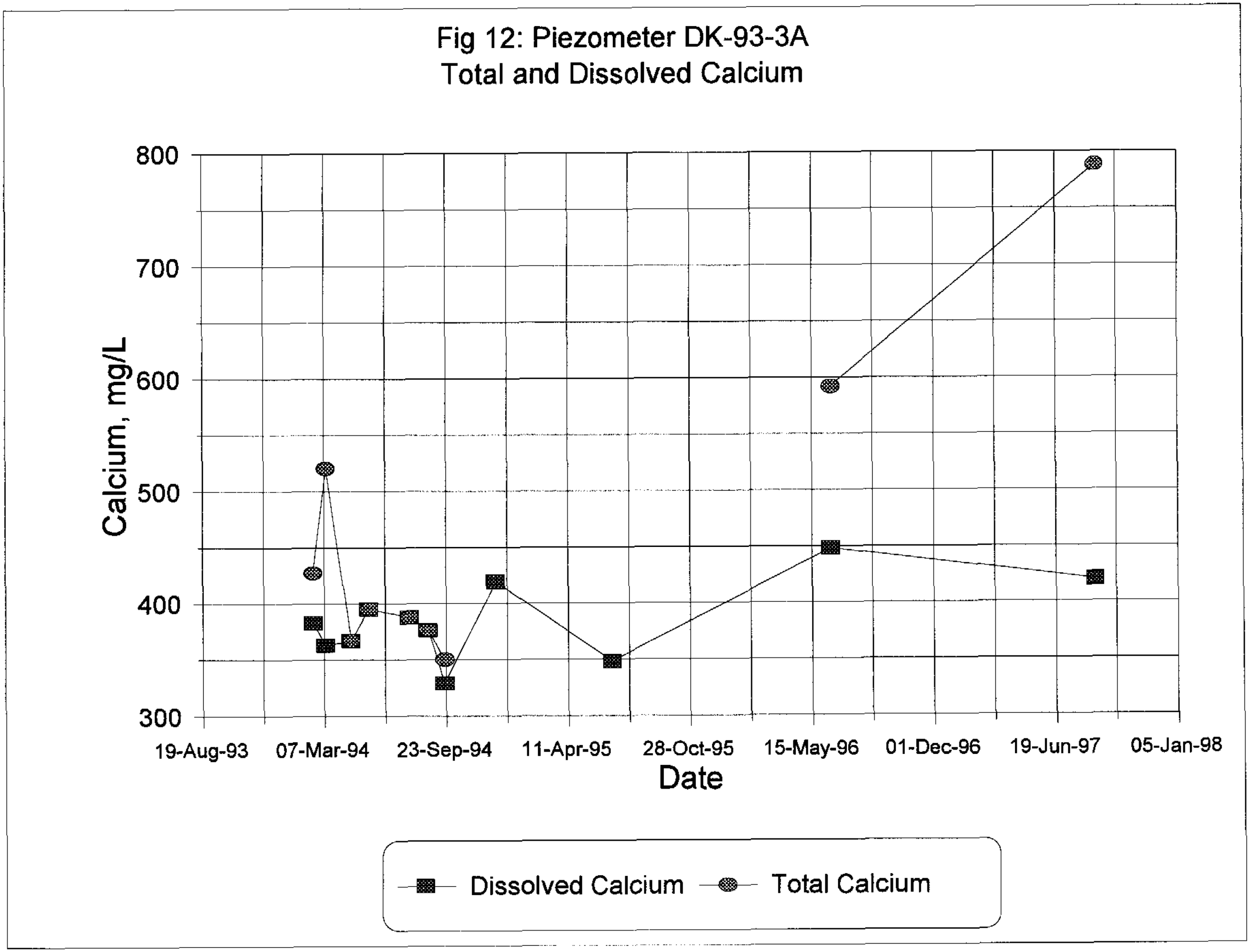


Fig 12: Piezometer DK-93-3A
Total and Dissolved Calcium
Calcium, mg/L
800
700
600
500
400
300
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Calcium
Total Calcium

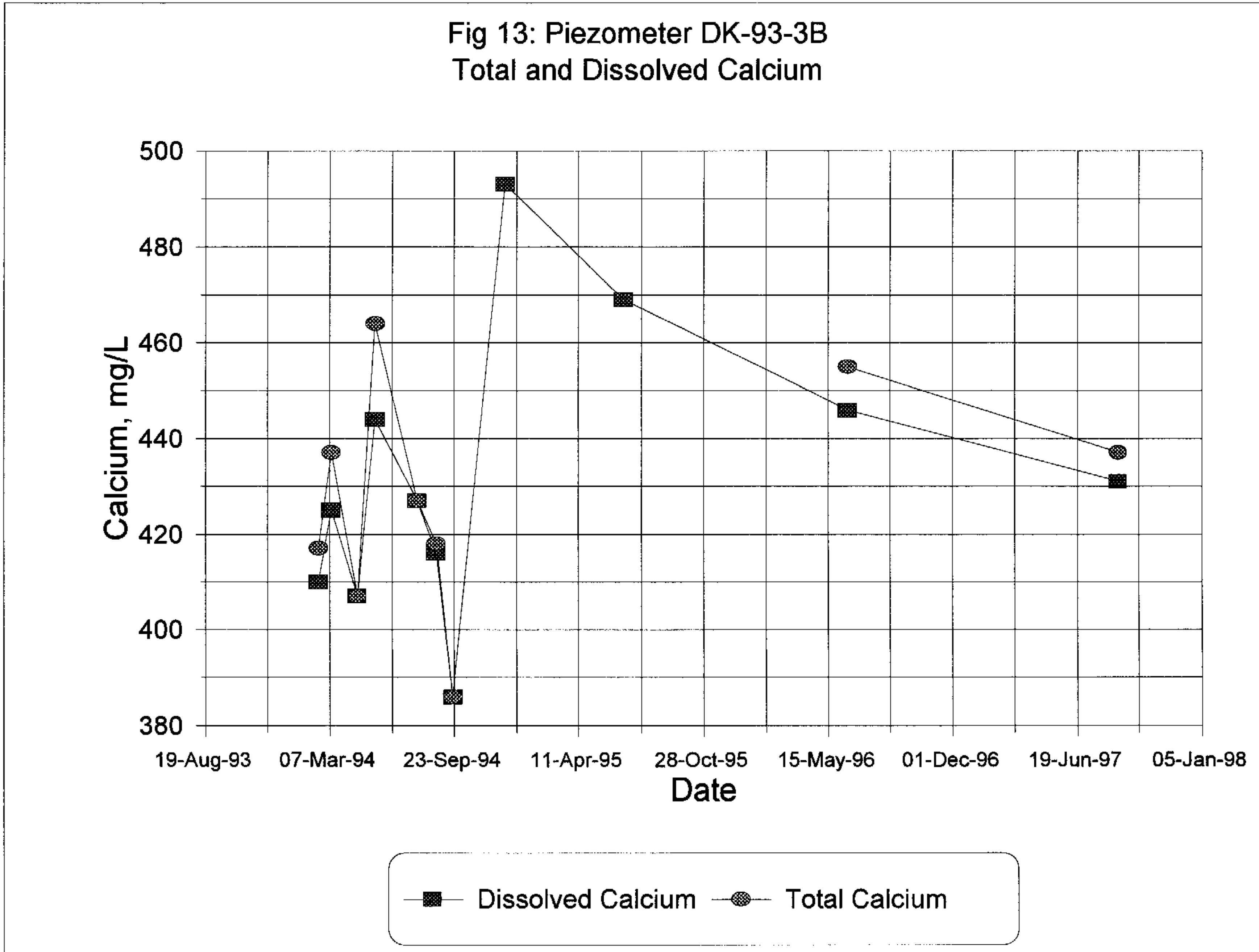


Fig 13: Piezometer DK-93-3B
Total and Dissolved Calcium
Calcium, mg/L
500
480
460
440
420
400
380
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Calcium
Total Calcium

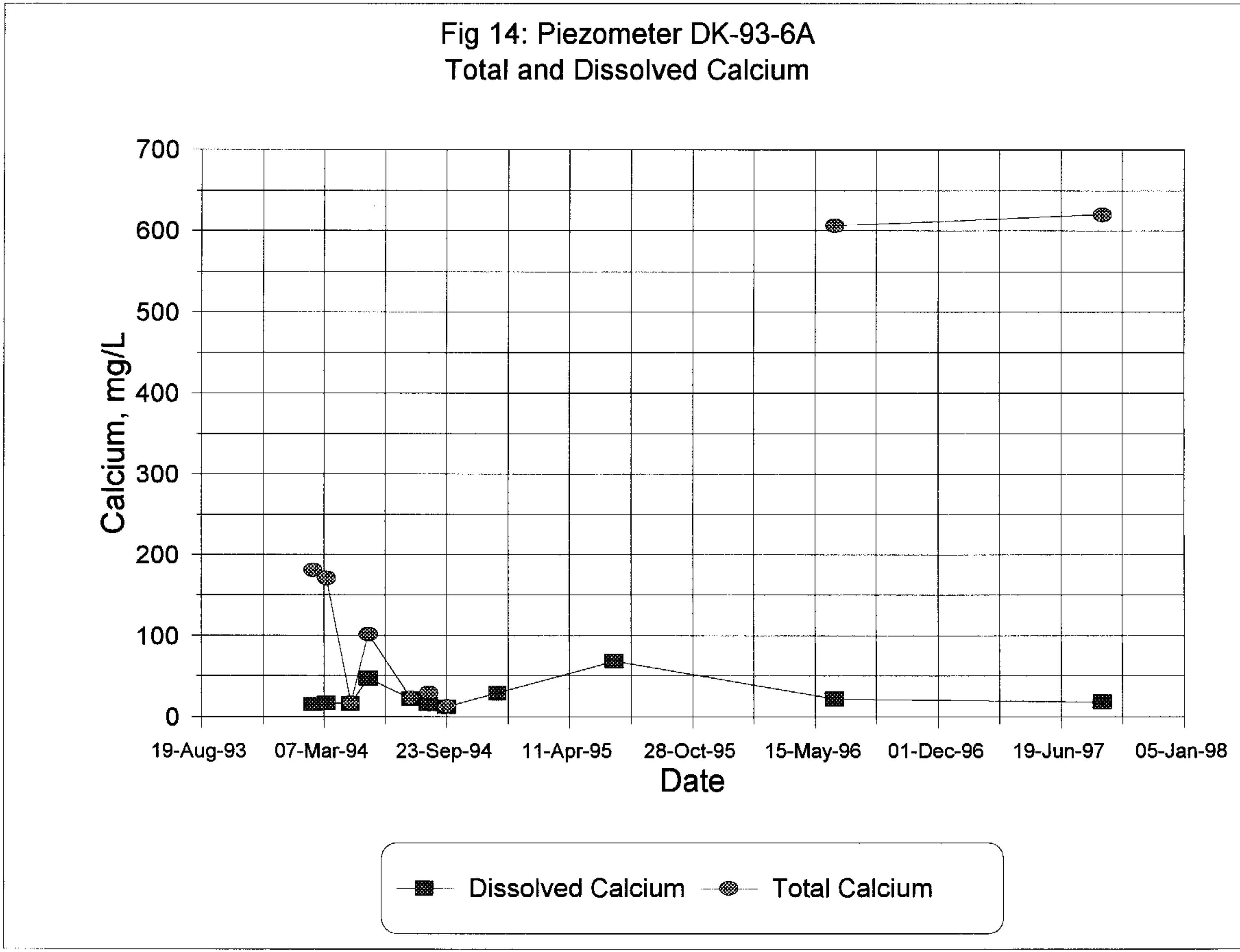
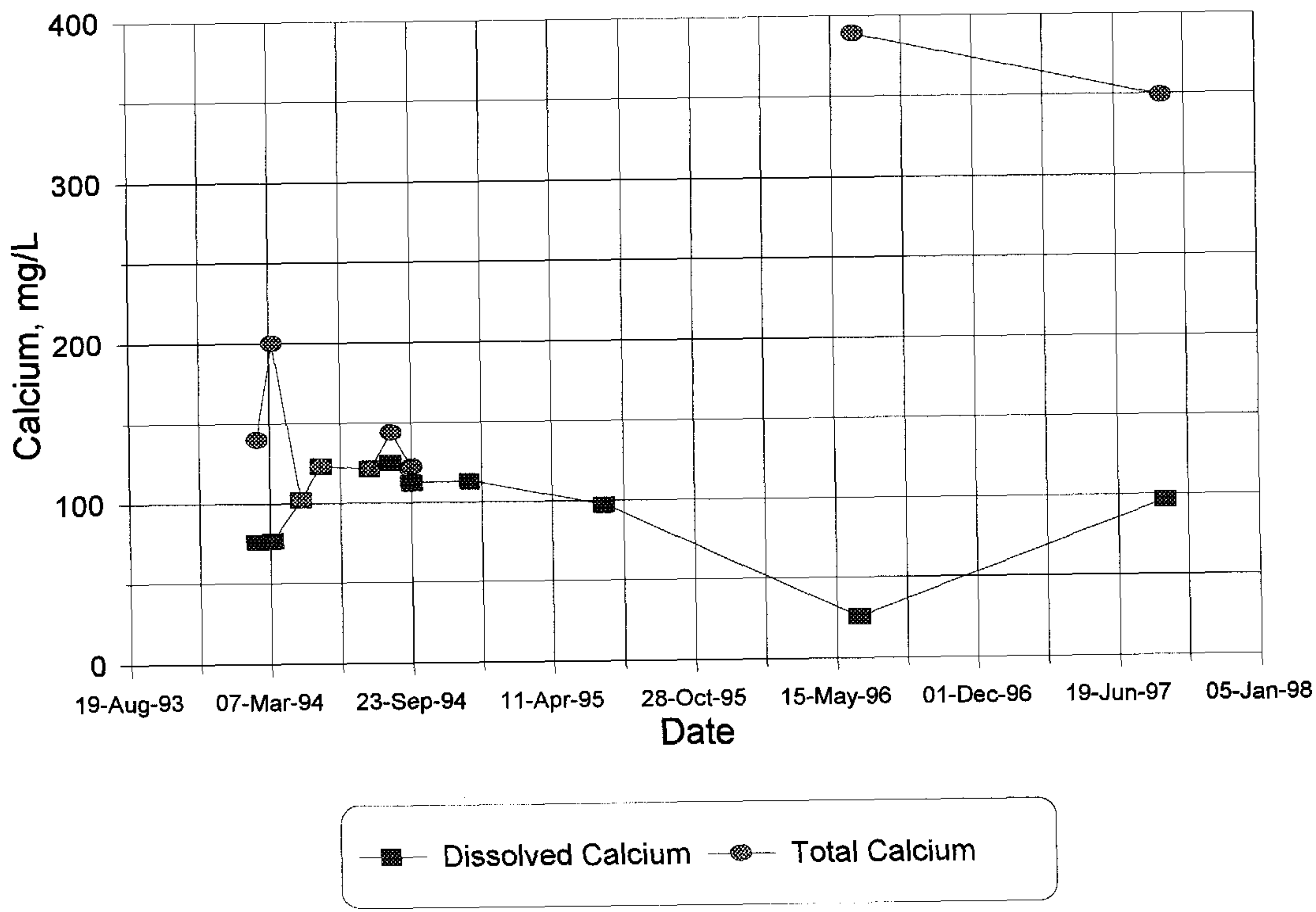


Fig 14: Piezometer DK-93-6A
Total and Dissolved Calcium
700
600
500
400
300
200
100
0
Calcium, mg/L
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Calcium
Total Calcium

Fig 15: Piezometer DK-93-6C
Total and Dissolved Calcium



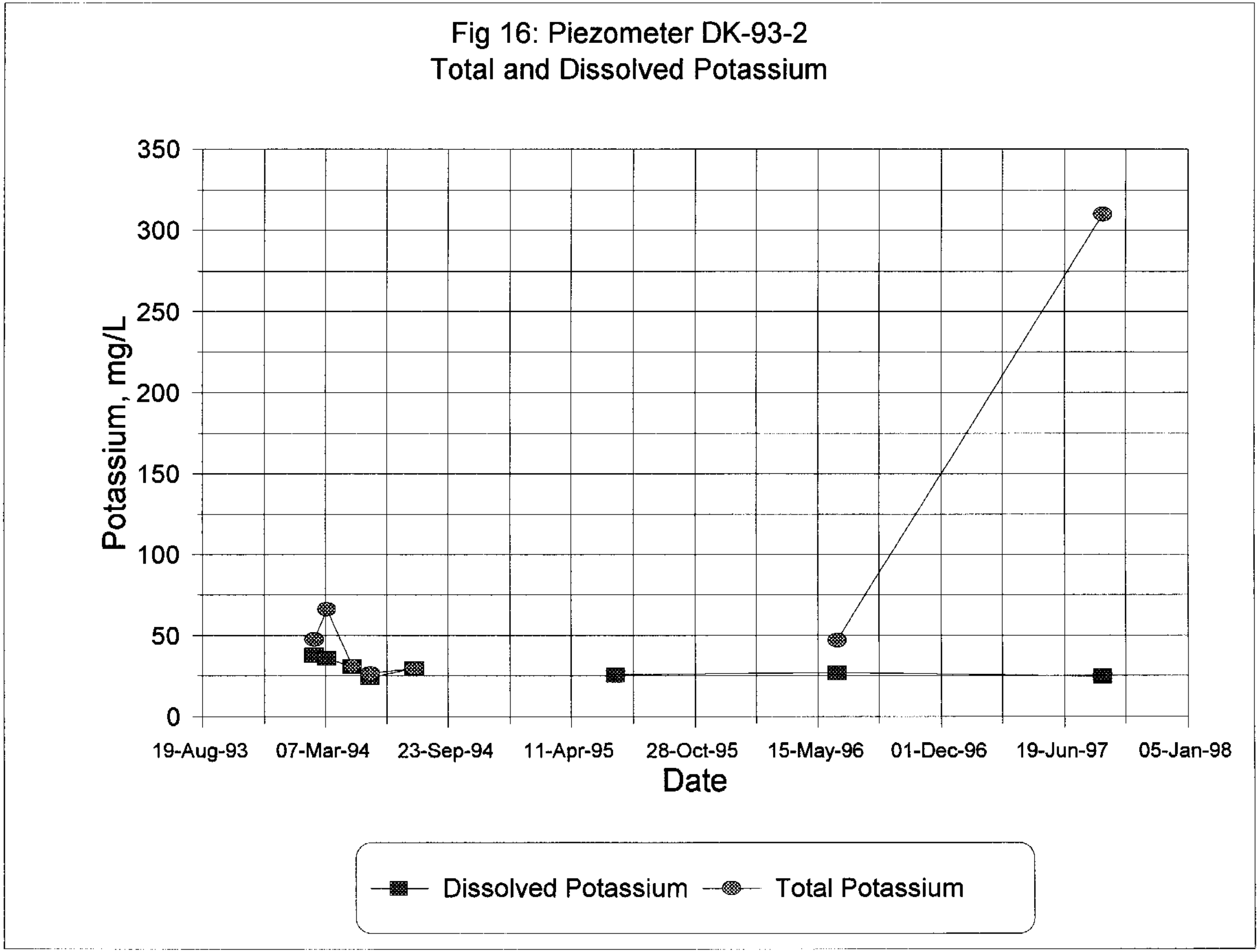


Fig 16: Piezometer DK-93-2
Total and Dissolved Potassium
Potassium, mg/L
350
300
250
200
150
100
50
0
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Potassium
Total Potassium

Fig 17: Piezometer DK-93-3A
Total and Dissolved Potassium

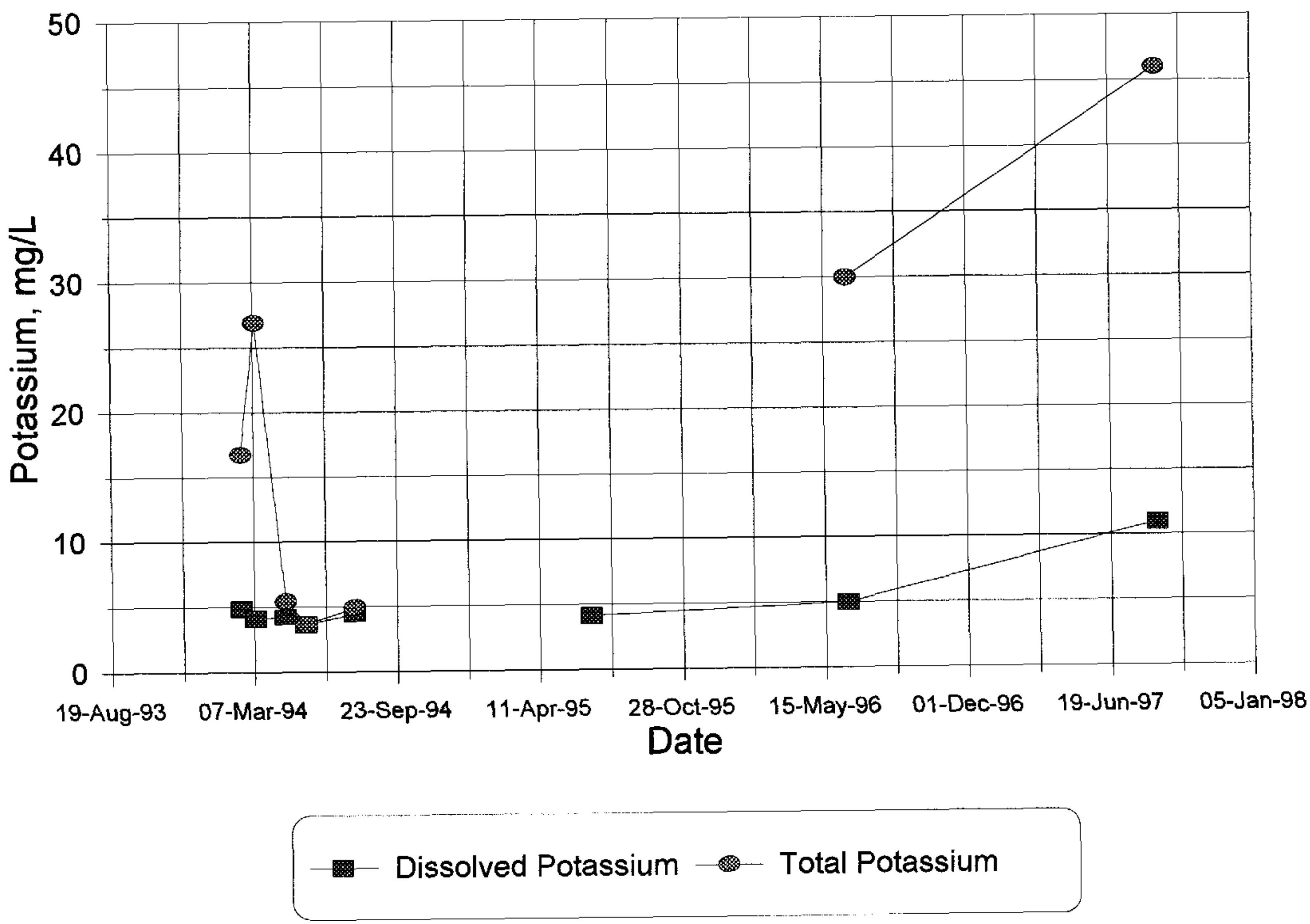


Fig 18: Piezometer DK-93-3B
Total and Dissolved Potassium

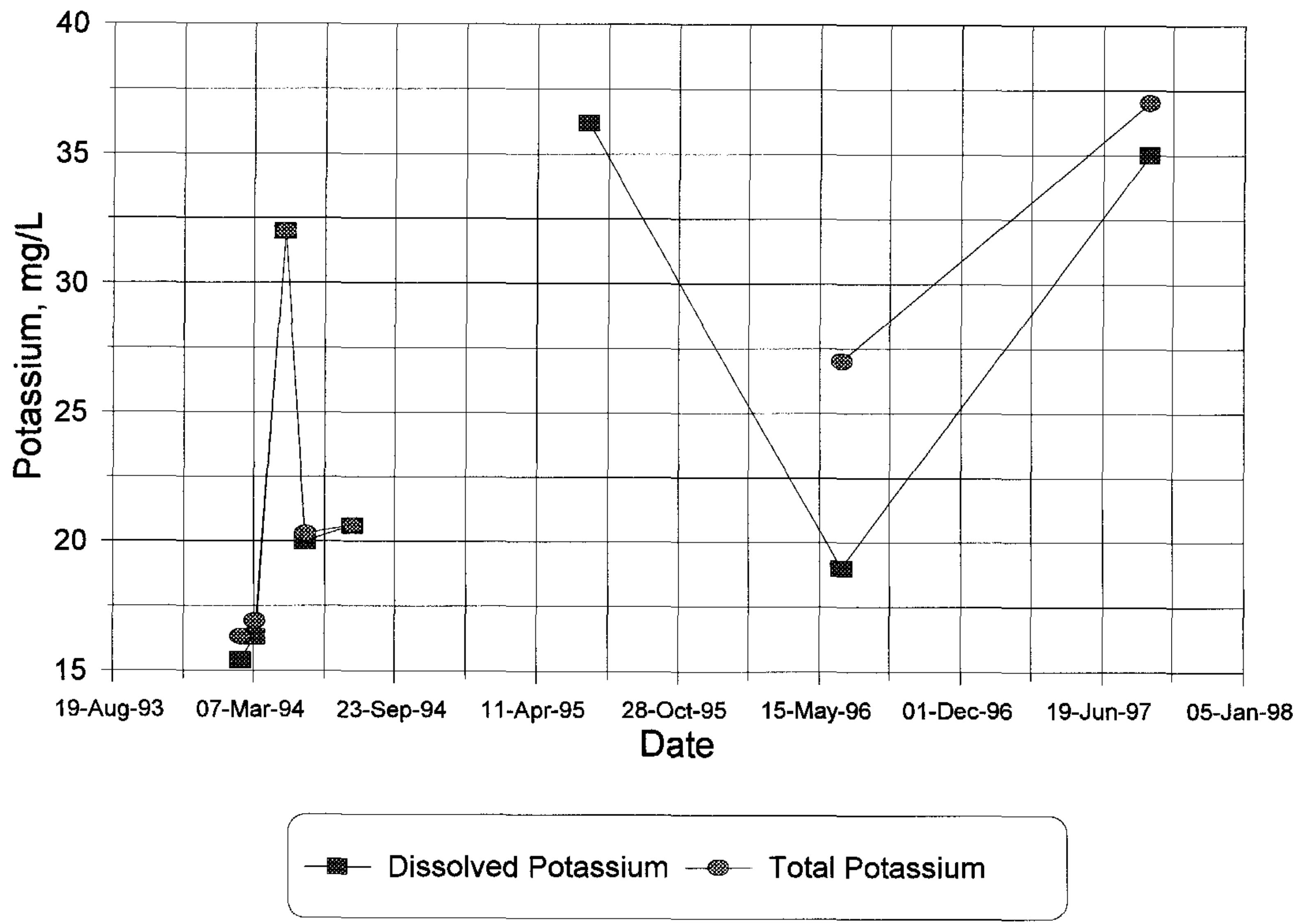


Fig 19: Piezometer DK-93-6A
Total and Dissolved Potassium

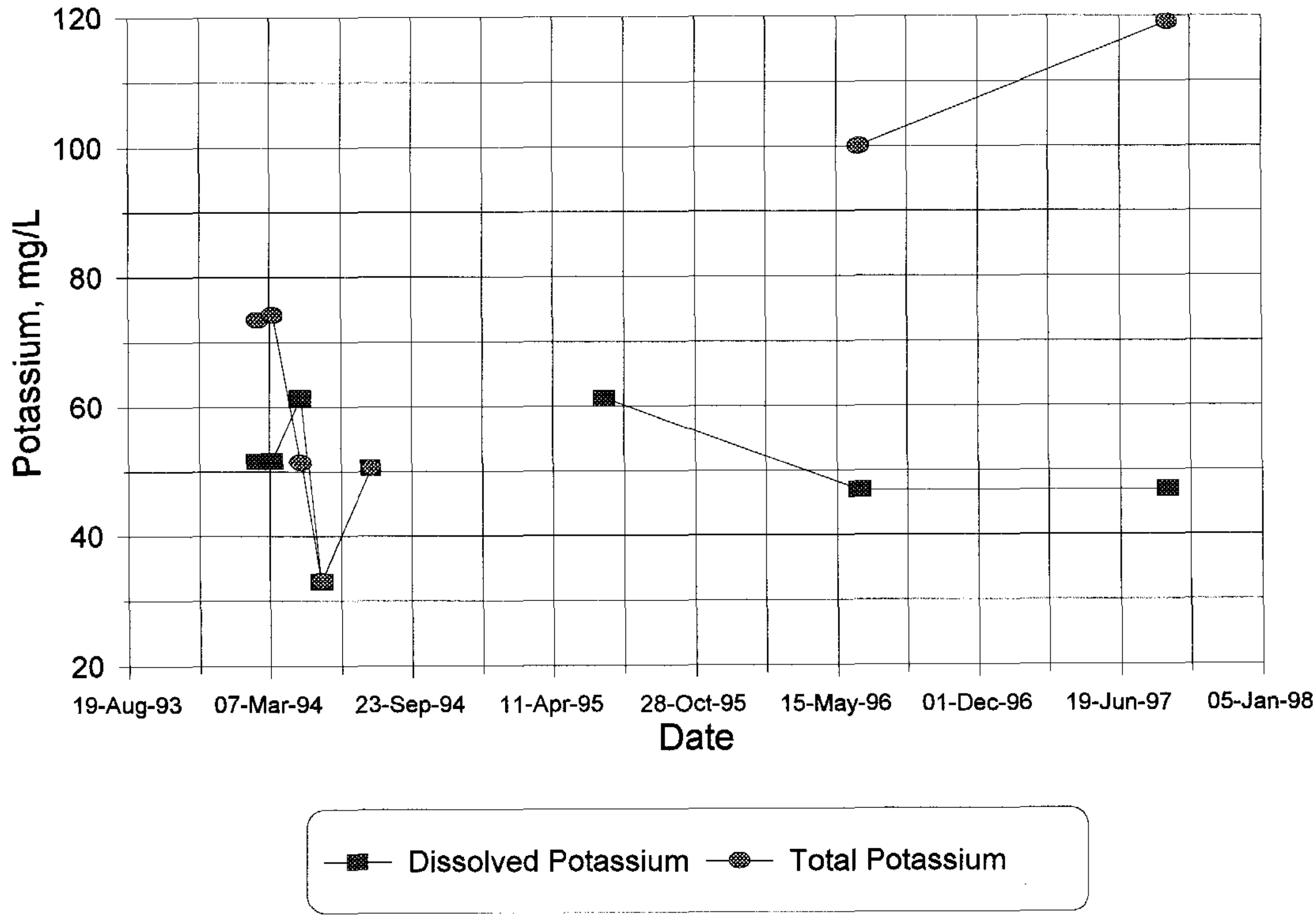
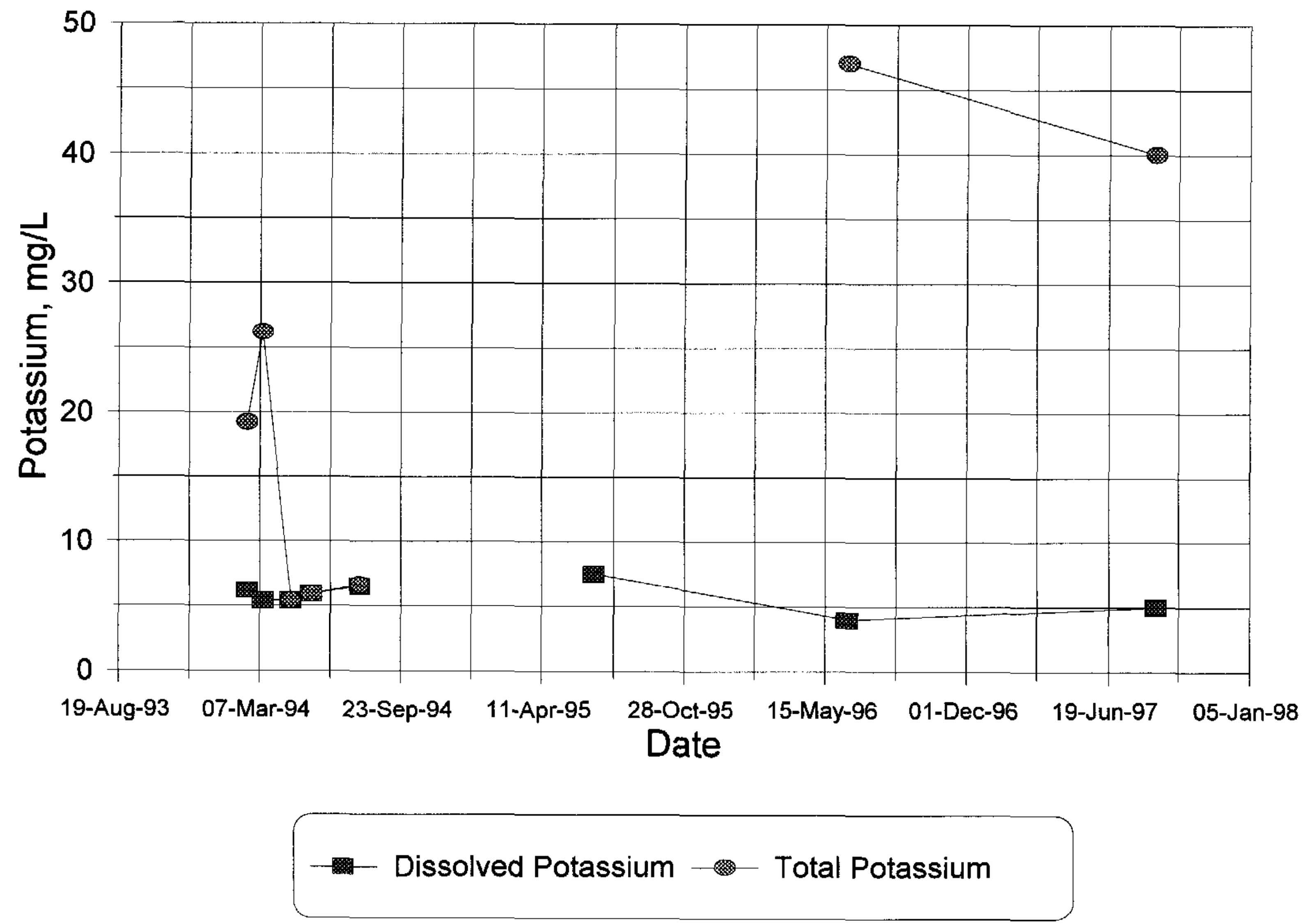


Fig 20: Piezometer DK-93-6C
Total and Dissolved Potassium



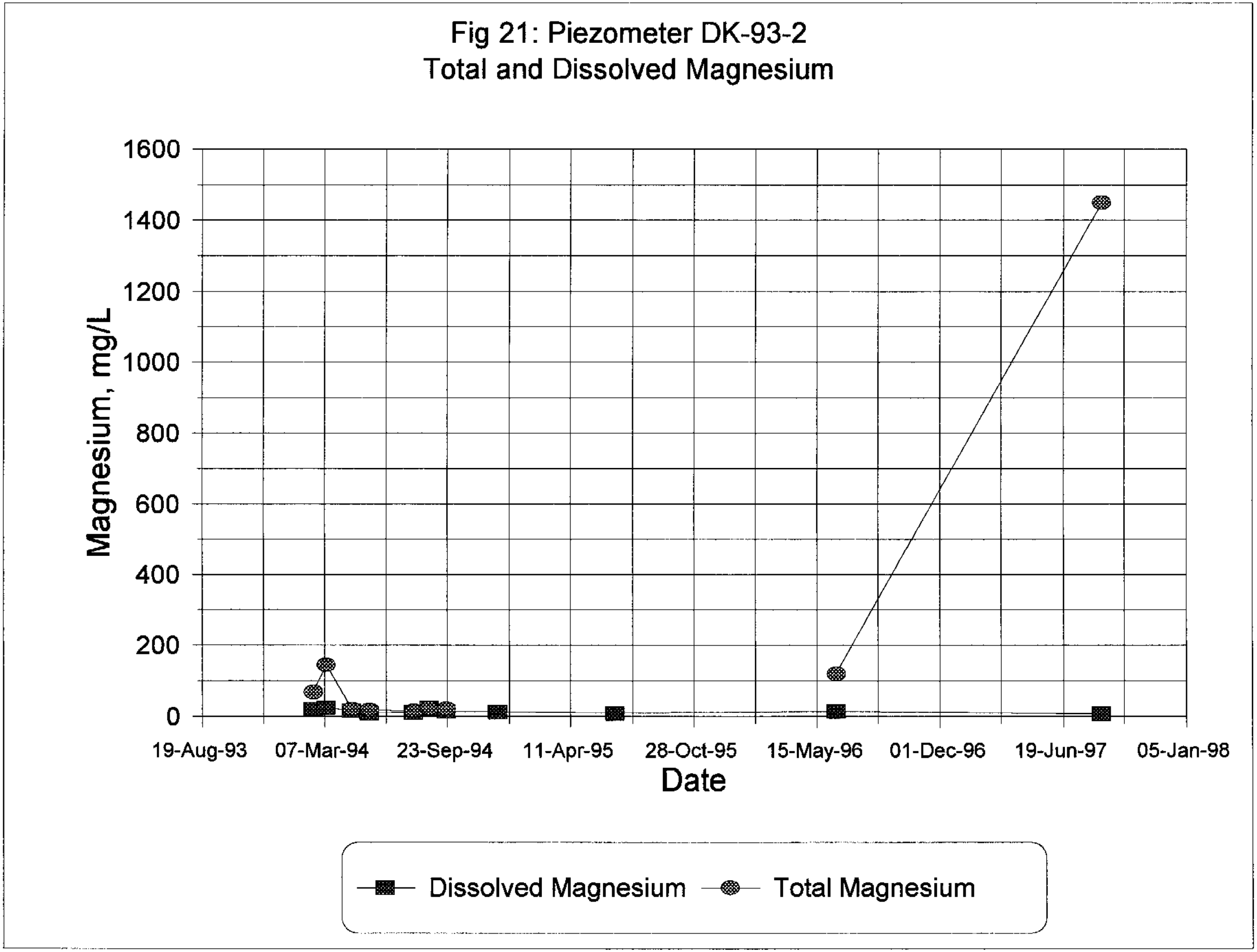
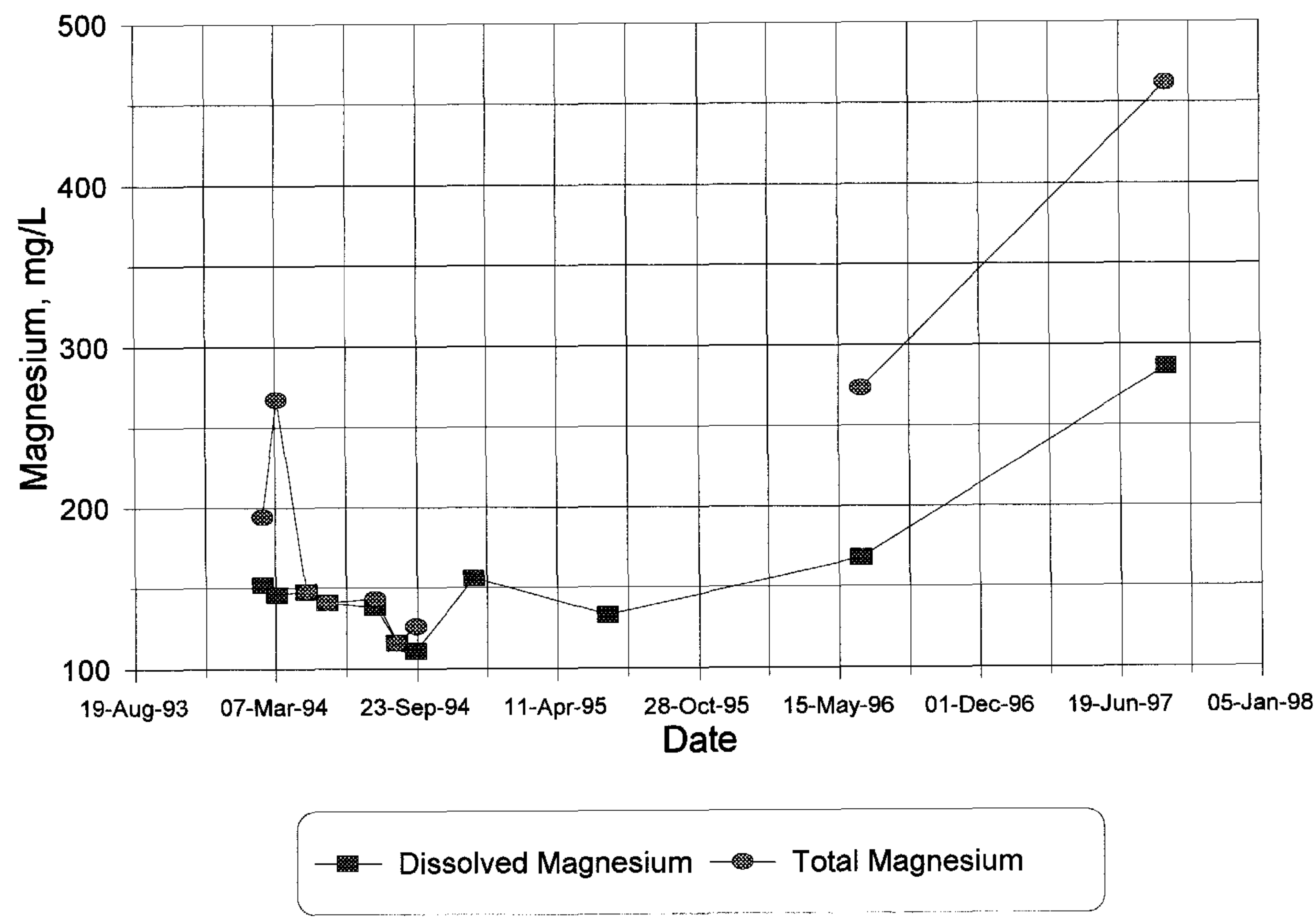


Fig 21: Piezometer DK-93-2
Total and Dissolved Magnesium
Magnesium, mg/L
1600
1400
1200
1000
800
600
400
200
0
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Magnesium
Total Magnesium

Fig 22: Piezometer DK-93-3A
Total and Dissolved Magnesium



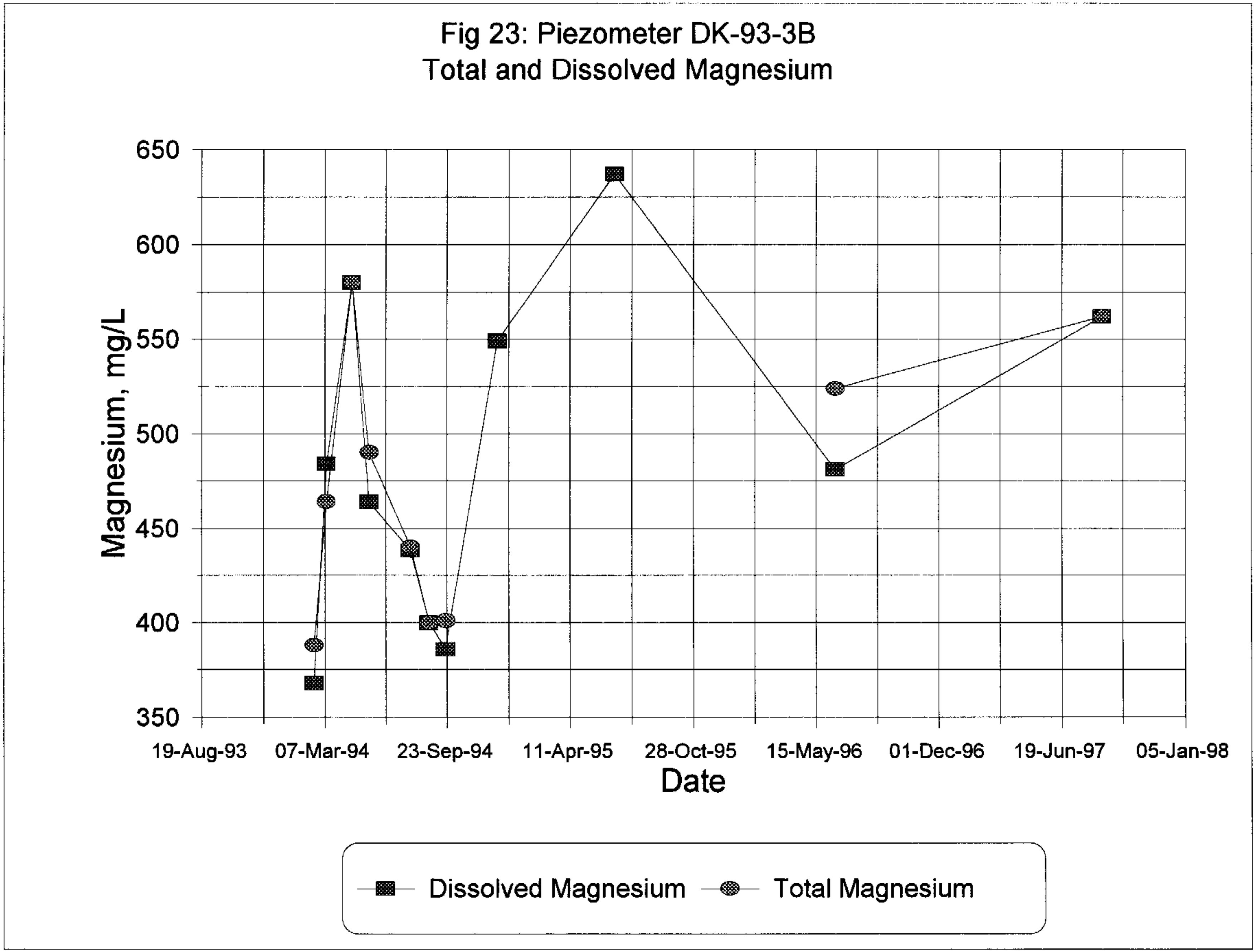


Fig 23: Piezometer DK-93-3B
Total and Dissolved Magnesium
650
600
550
500
450
400
350
Magnesium, mg/L
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Magnesium
Total Magnesium

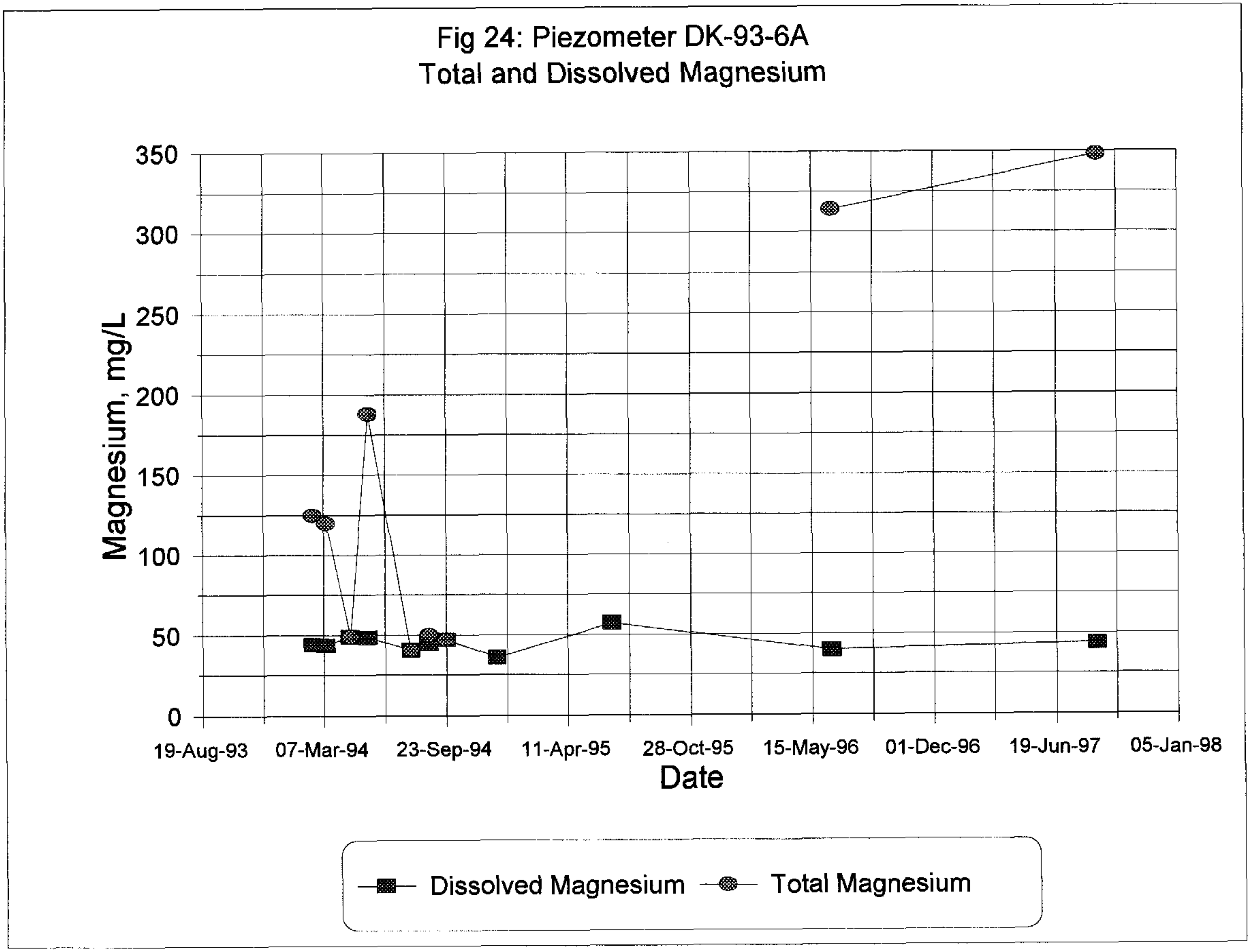


Fig 24: Piezometer DK-93-6A
Total and Dissolved Magnesium
Magnesium, mg/L
350
300
250
200
150
100
50
0
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Magnesium
Total Magnesium

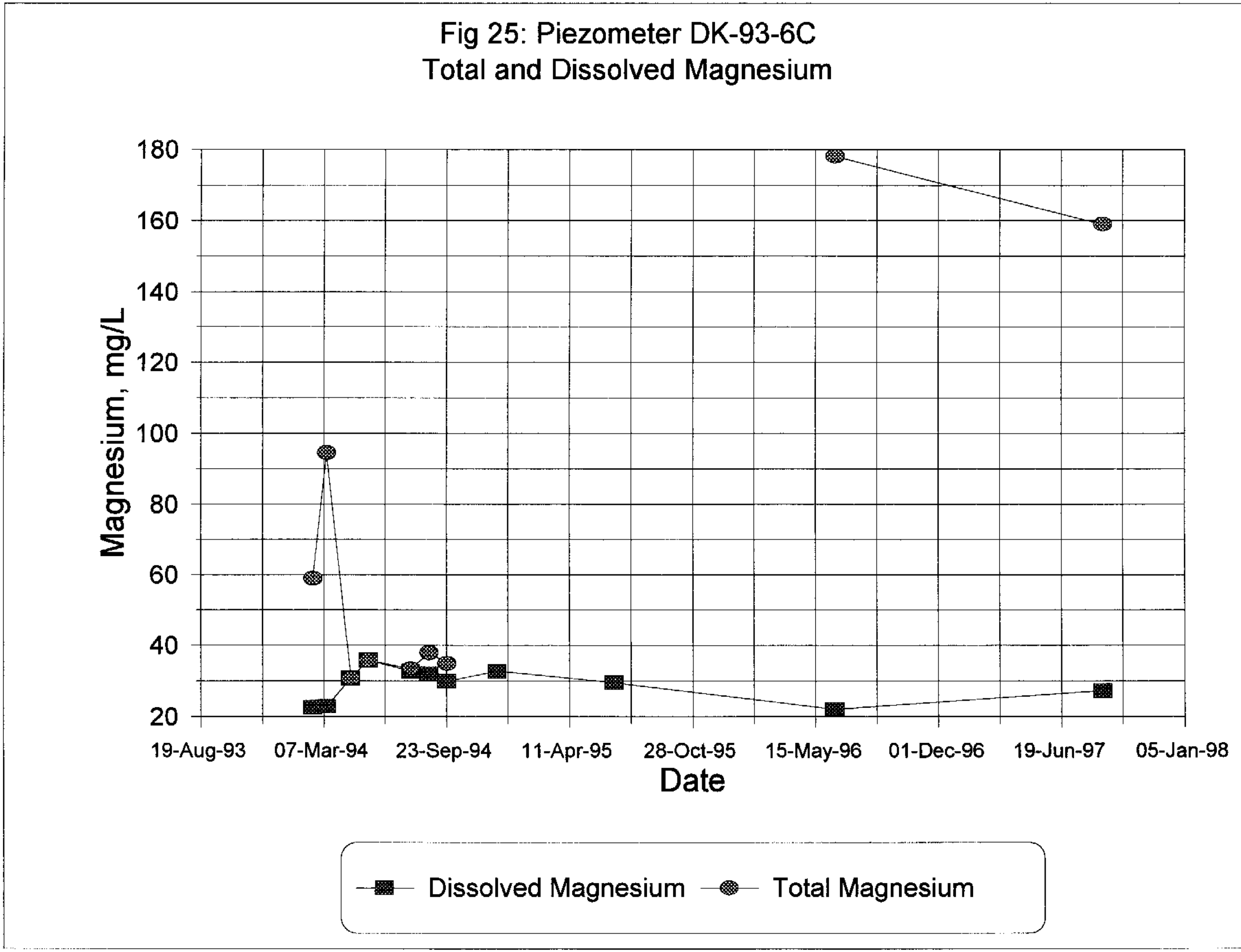


Fig 25: Piezometer DK-93-6C
Total and Dissolved Magnesium
Magnesium, mg/L
180
160
140
120
100
80
60
40
20
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Magnesium
Total Magnesium

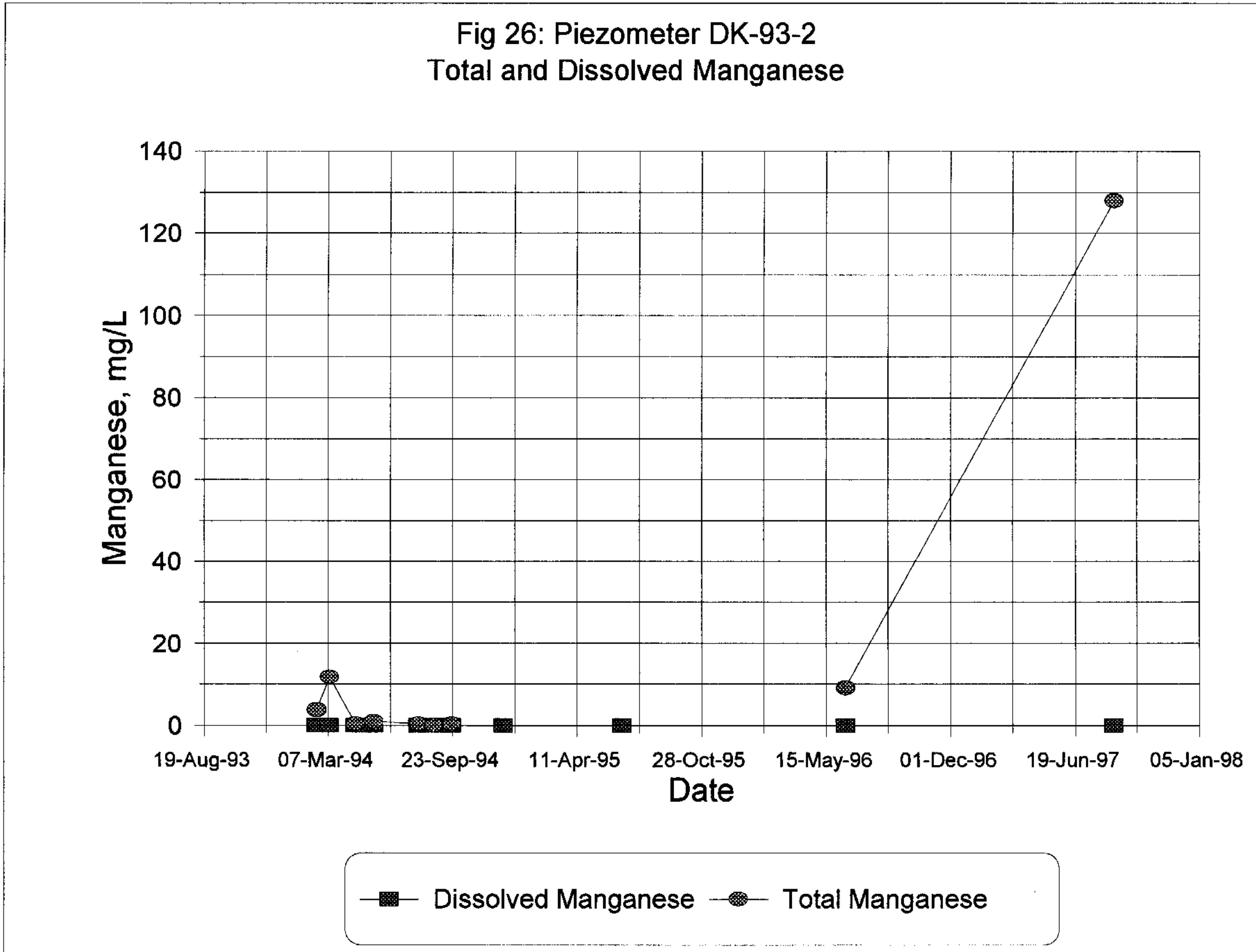


Fig 26: Piezometer DK-93-2
Total and Dissolved Manganese
140
120
100
80
60
40
20
0
Manganese, mg/L
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Manganese
Total Manganese

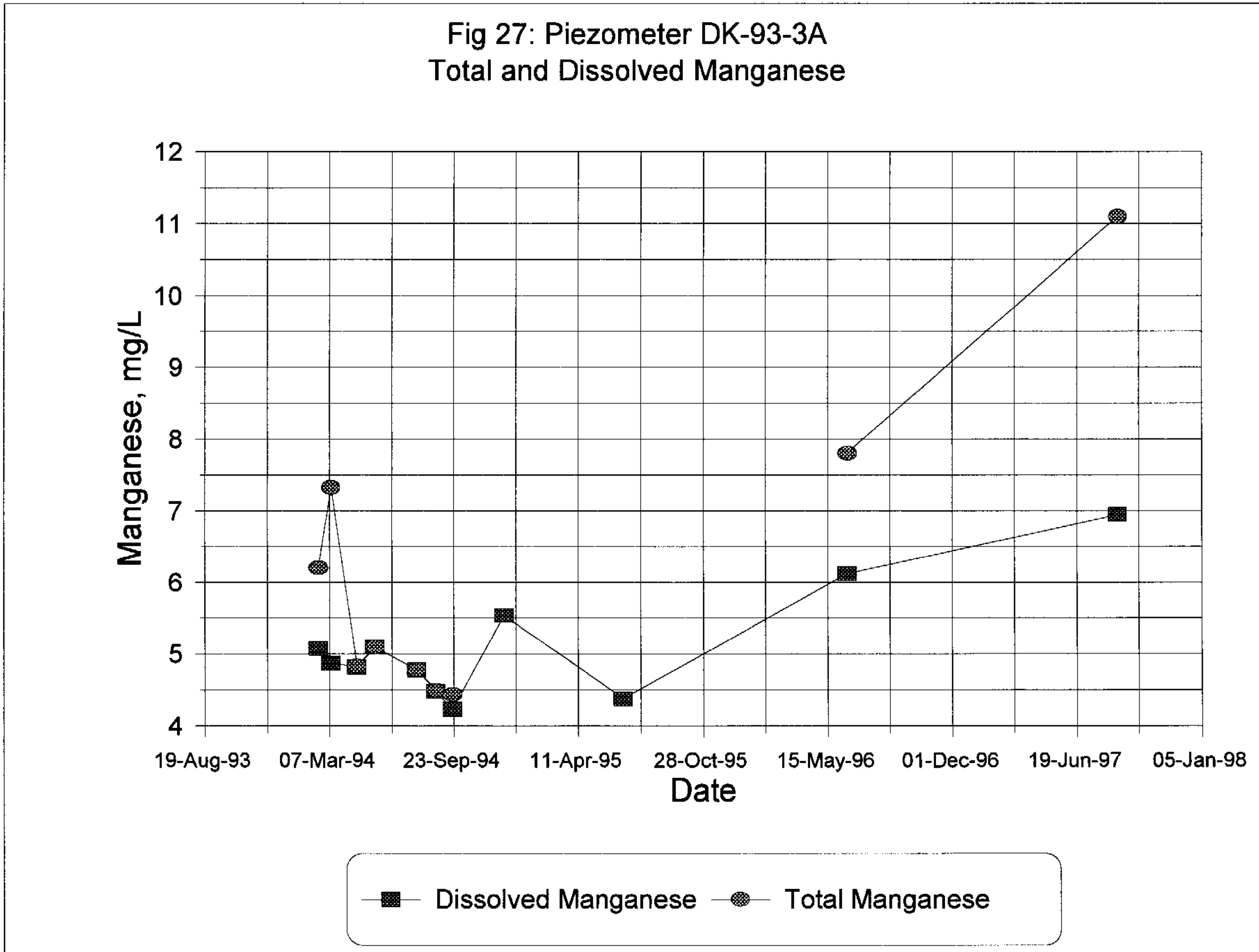
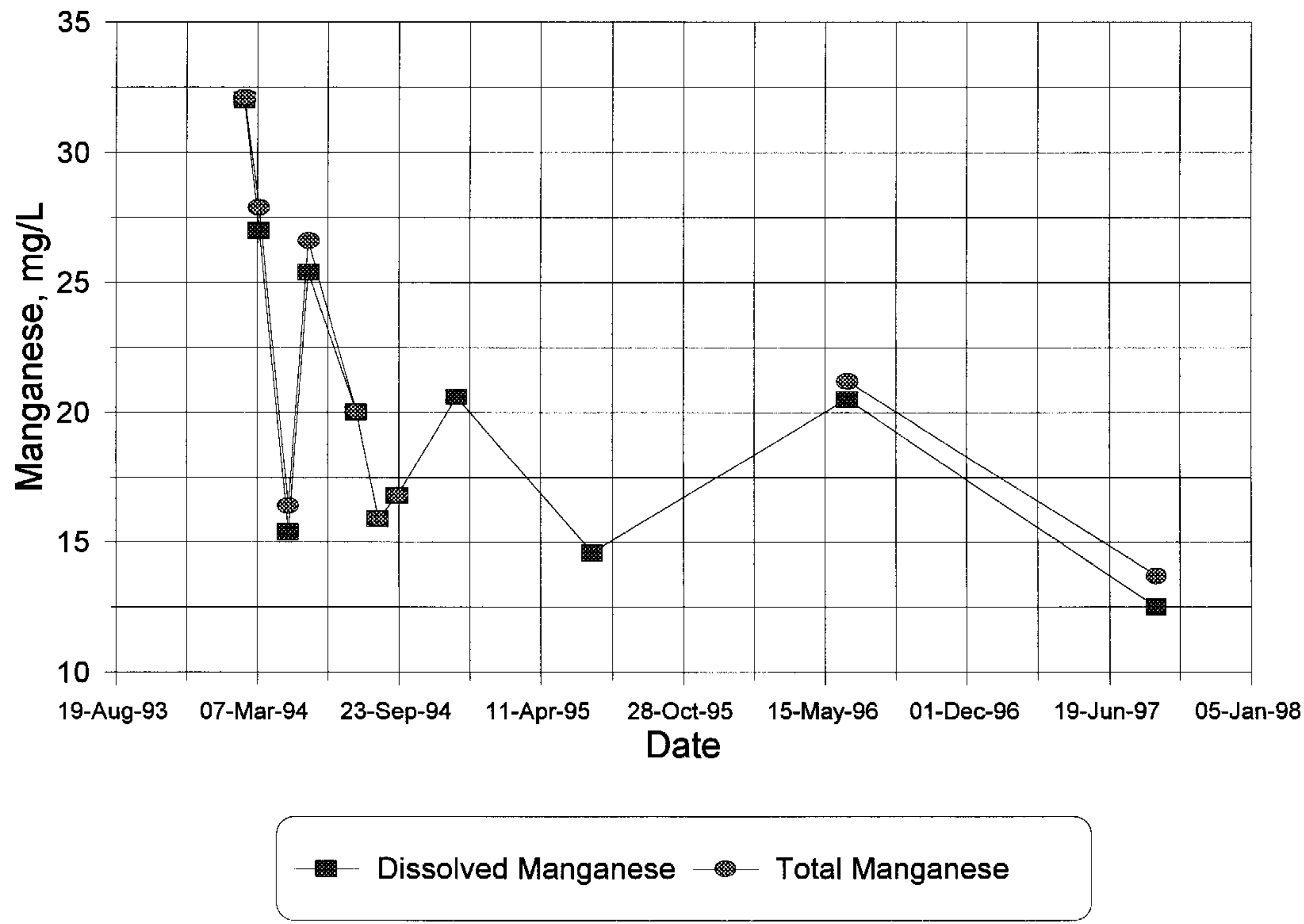


Fig 27: Piezometer DK-93-3A
Total and Dissolved Manganese
Manganese, mg/L
12
11
10
9
8
7
6
5
4
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Manganese
Total Manganese

Fig 28: Piezometer DK-93-3B
Total and Dissolved Manganese



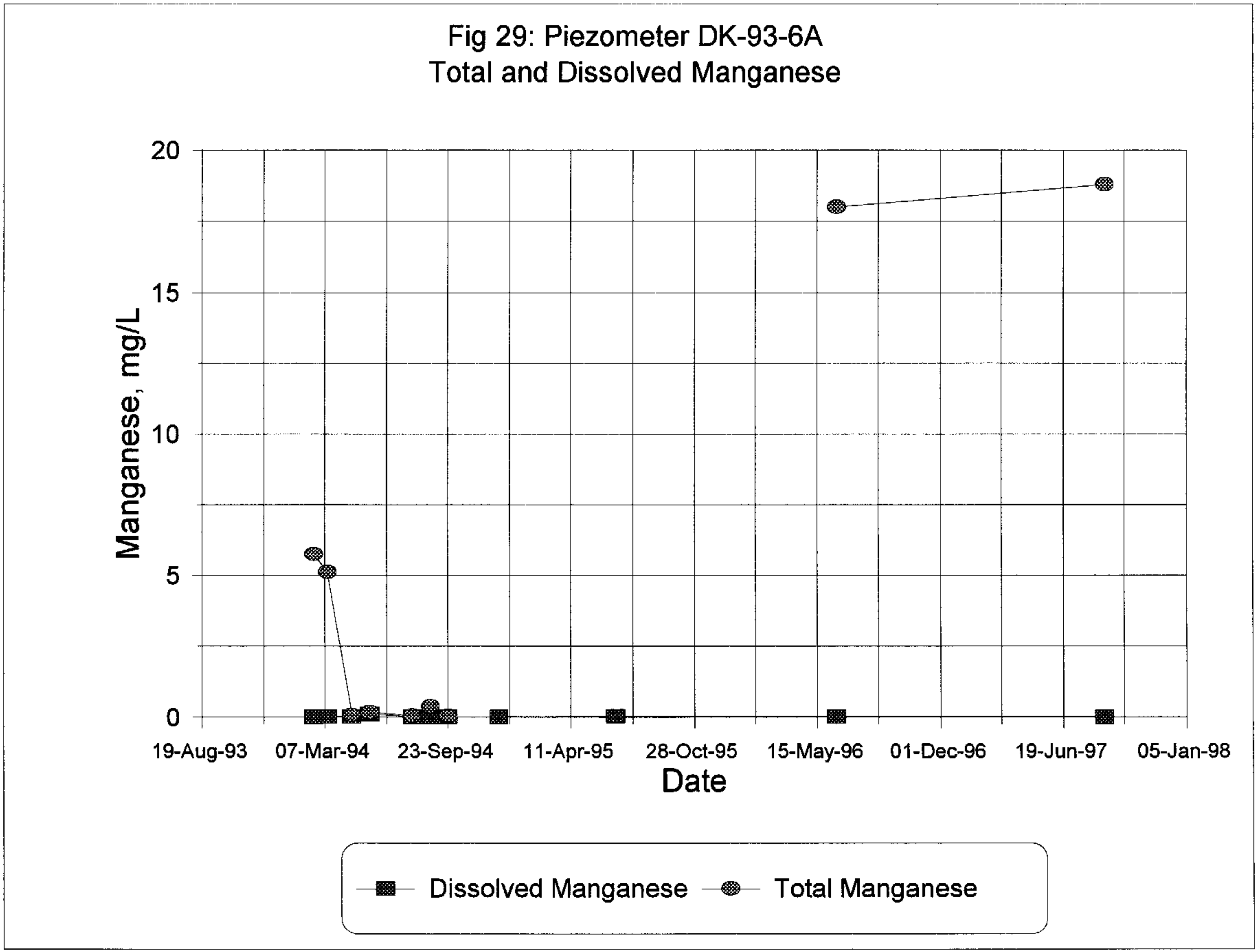
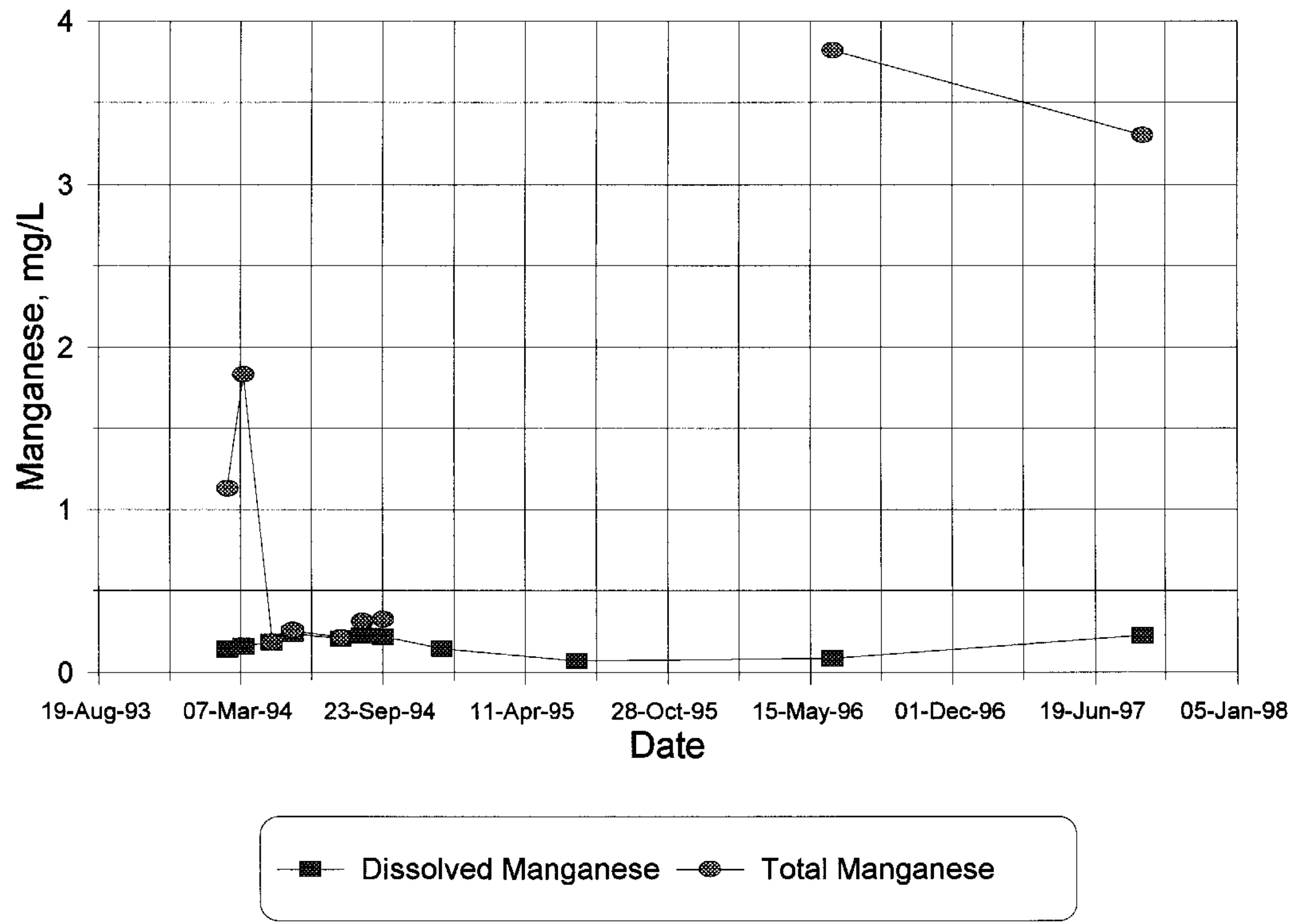


Fig 29: Piezometer DK-93-6A
Total and Dissolved Manganese
Manganese, mg/L
20
15
10
5
0
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Manganese
Total Manganese

Fig 30: Piezometer DK-93-6C
Total and Dissolved Manganese



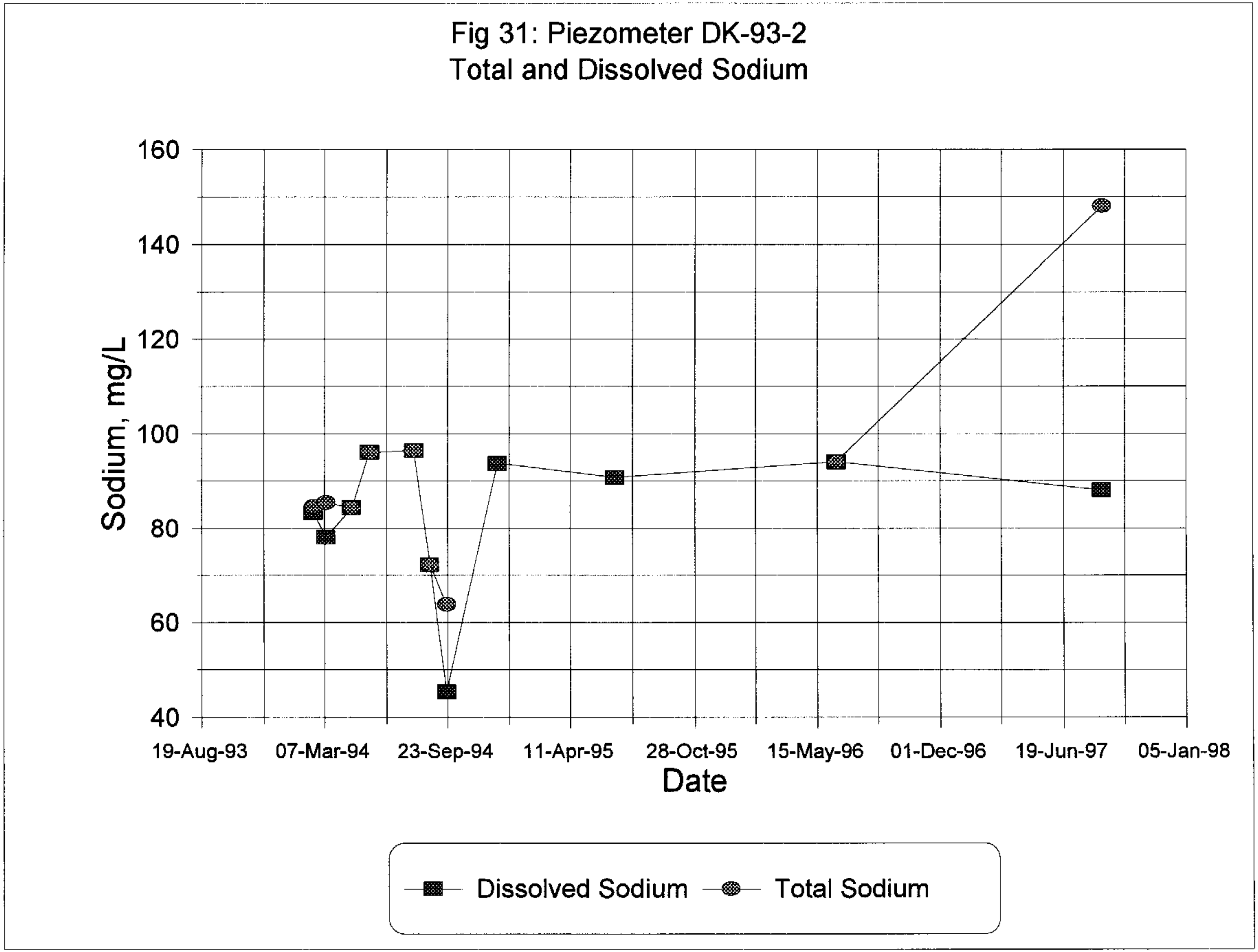


Fig 31: Piezometer DK-93-2
Total and Dissolved Sodium
Sodium, mg/L
160
140
120
100
80
60
40
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Sodium
Total Sodium

Fig 32: Piezometer DK-93-3A
Total and Dissolved Sodium

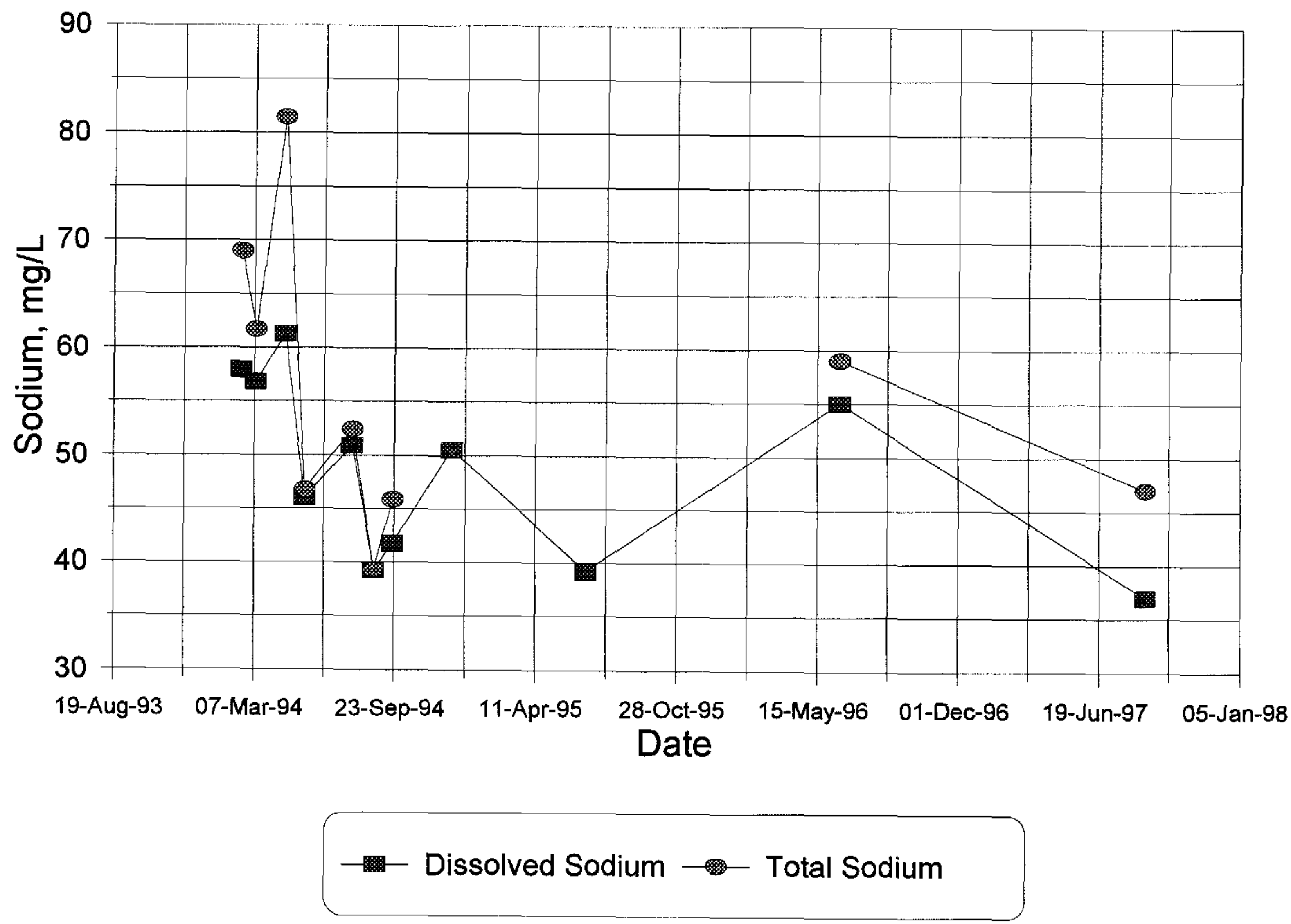


Fig 33: Piezometer DK-93-3B
Total and Dissolved Sodium

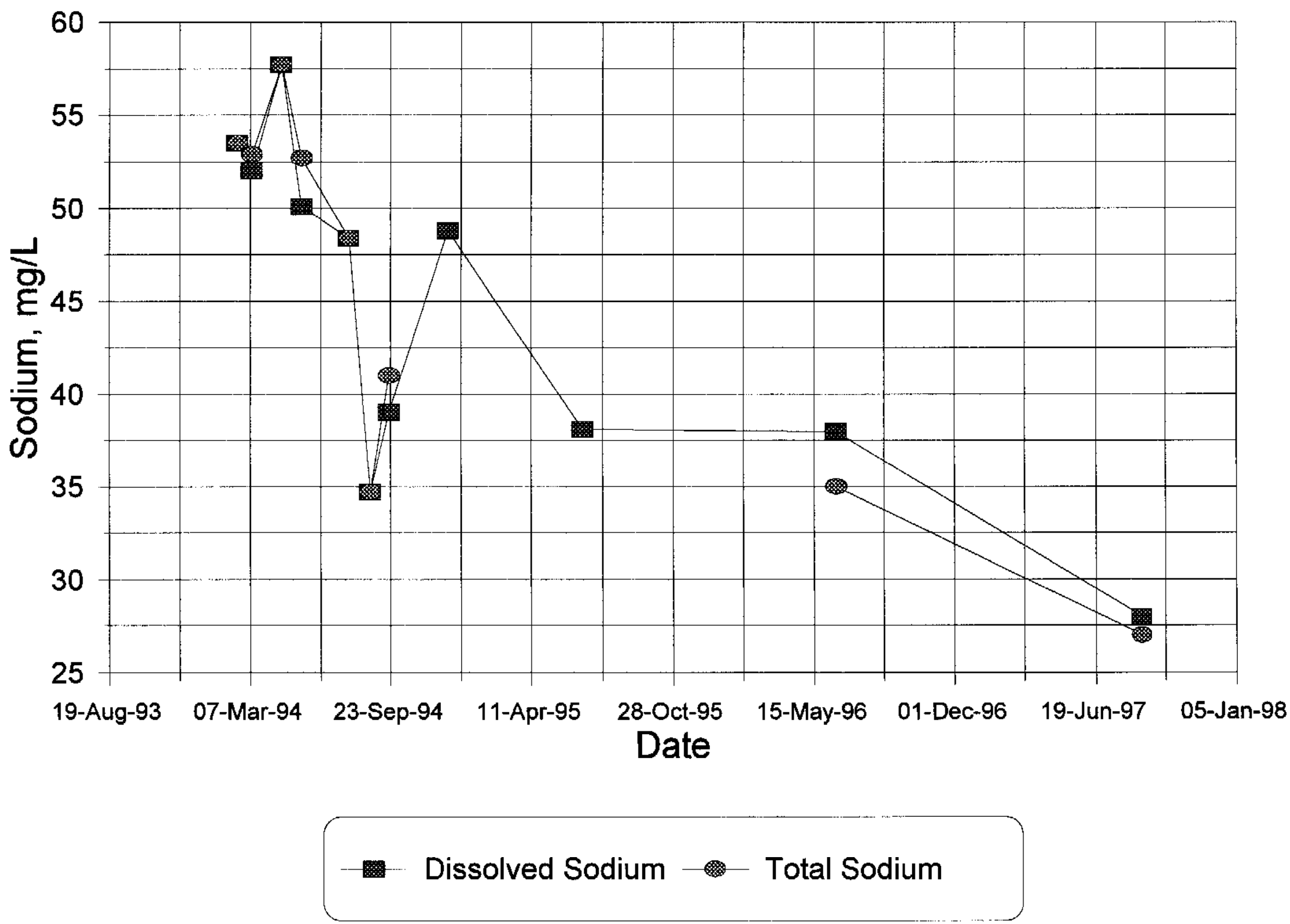
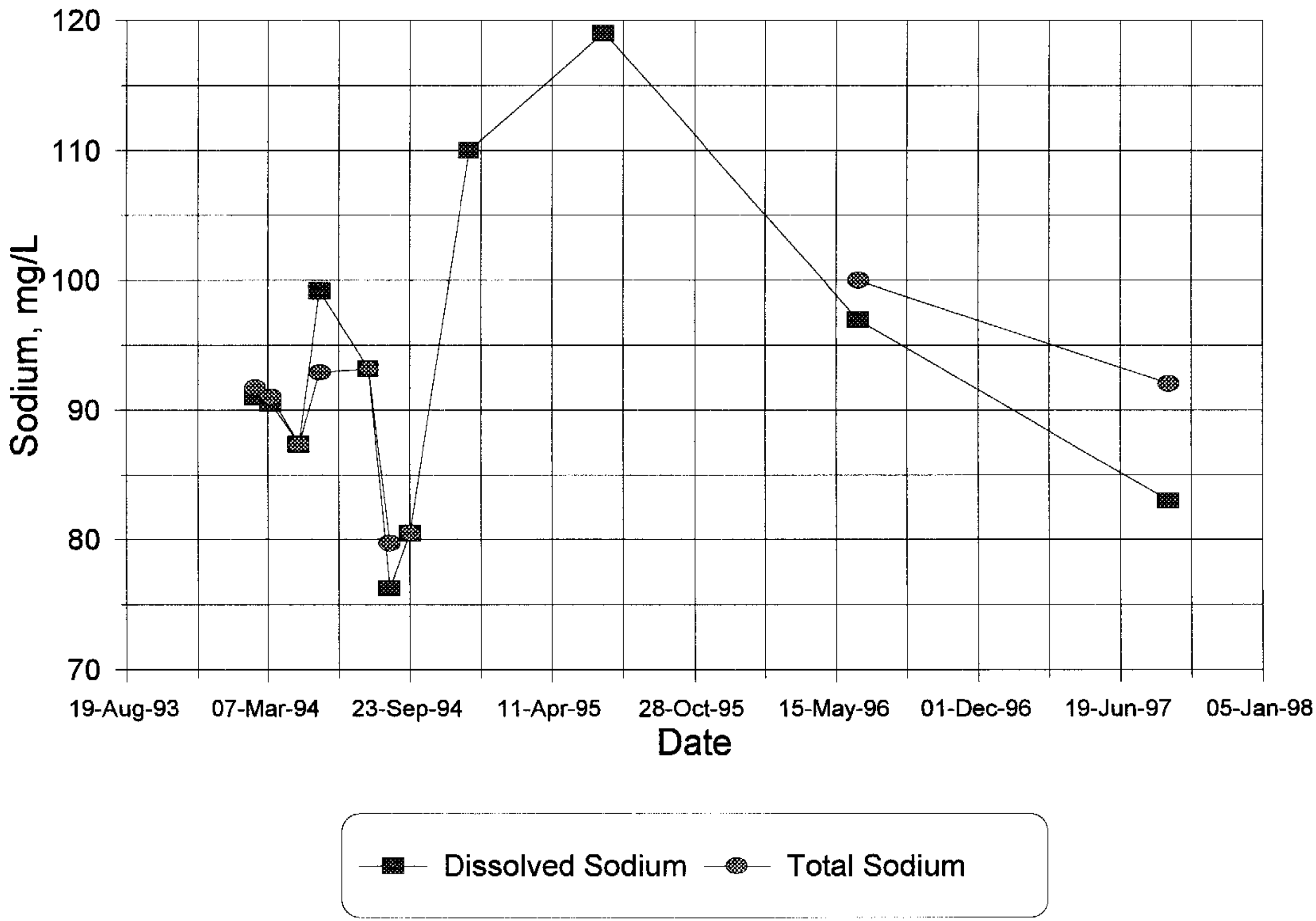


Fig 34: Piezometer DK-93-6A
Total and Dissolved Sodium



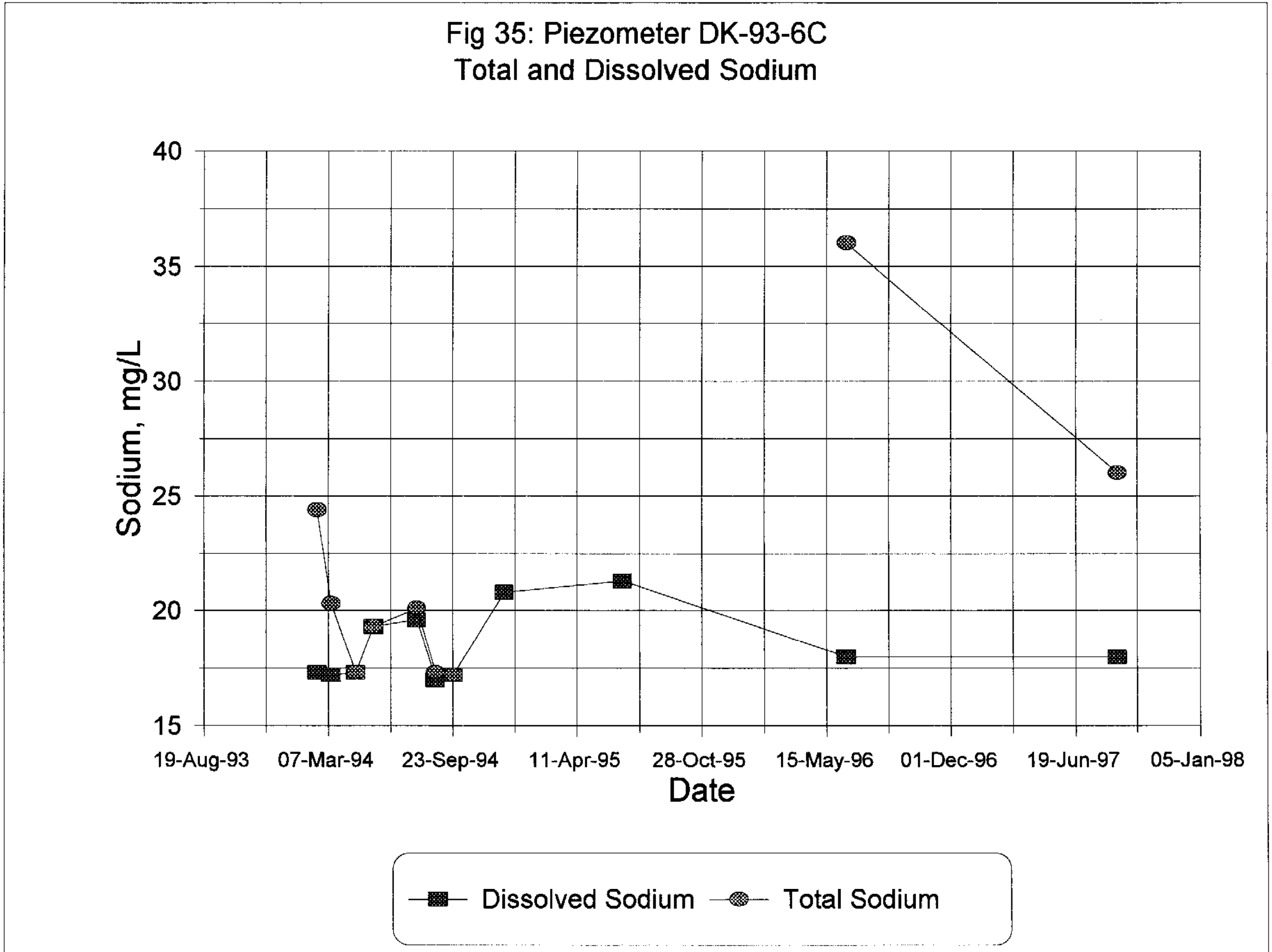


Fig 35: Piezometer DK-93-6C
Total and Dissolved Sodium
Sodium, mg/L
40
35
30
25
20
15
19-Aug-93
07-Mar-94
23-Sep-94
11-Apr-95
28-Oct-95
15-May-96
01-Dec-96
19-Jun-97
05-Jan-98
Date
Dissolved Sodium
Total Sodium