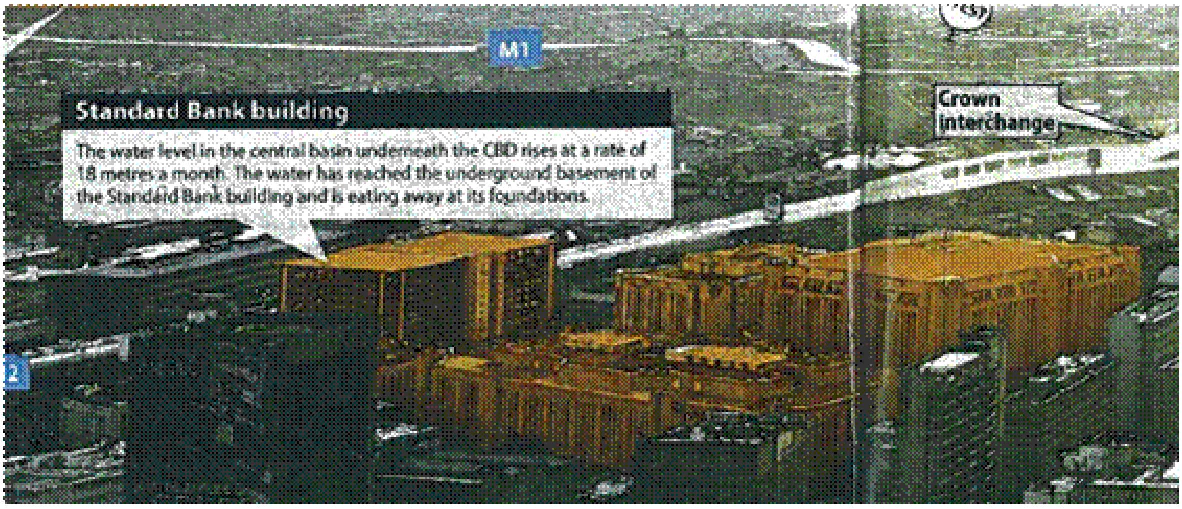

('You Magazine', March 2011 claiming that the basement of Standard Banks new administration building is already being flooded by acidic mine water)

Desktop assessment of the risk for basement structures of buildings of Standard Bank and ABSA in Central Johannesburg to be affected by rising mine water levels in the Central Basin

Final Report (Volume I of III)

Attention of:	*Mr. Rory R. Roriston* (Standard Bank, Rosebank)
	Mr. Nival Surban (ABSA, Johannesburg)
Submitted by:	Mine Water Research Group, NWU, Potchefstroom Campus, Geography, School of Environmental Sciences and Development; Team leader: Prof. Dr. Frank Winde
Status of report:	Confidential, requester only
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Quotes from media

BEELD 24.04.07.3

Geboue in Jhb kan meegee oor water

■ Suur myn-H_20 kan fondasies onstabiel maak

Elise Tempelhoff

Randfontein. – Suur mynwater wat besig is om ondergronds aan die Witwatersrand op te stoot, kan binne 'n dekade die fondasies van hoë geboue in die Johannesburgse middestad só onstabiel maak dat dit in duie sal stort.

Een van dié hoër geboue is volgens dr. Anthony Turton, leiernavorser oor water van die Wetenskaplike Nywerheids- en Navorsingsraad (WNNR), glo reeds hierdeur aangetas.

Hy weerhou dié gebou se naam om "histerie te voorkom".

Daarbenewens gaan die suurmynwater waarskynlik ook in Nigel se hoofstrate afstroom en staalstrukture in fondasies "aanval" wat geboue sal laat ineenstort.

Dít omdat die water, wat langer as 'n eeu deur mynhuise uit die dolomitiese kompartemente van die Witwatersrand gepomp is, besig is om teen 'n baie vinnige tempo terug te keer. Dié water is nou besoedelde suur-, radioaktiewe water omdat baie myne reeds gesluit het.

Turton het met *Beeld* gepraat nadat 'n brief van hom Donderdag op 'n vergadering aan lede van die Randfontein Environmental Action Group (REAG) voorgelees is.

Turton sê navorsing het bewys suur mynwater gaan waarskynlik oor minder as tien jaar die fondasies van geboue aan die Witwatersrand oorstroom.

Dr. Olaf Pollmann, 'n Duitse ingenieur wat navorsing oor mynre-

saaklik" dat die regeringsdepartemente waterwese en bosbou, omgewingsake en toerisme en minerale en energie, nóú vennootskappe met mekaar en die private sektor sluit om dié "uiters ernstige saak" aan te pak.

Beeld verneem daar is reeds enkele maande gelede 'n regeringstaakgroep (RTG) tussen dié drie departemente aangewys, maar dit kan nie op dreef kom nie omdat hulle nie kan besluit wie die leiding behoort te neem nie.

Volgens Turton is die mure van die "baie hoë gebou met 'n baie diep fondasie" in die Johannesburgse middestad waar die suur water reeds begin deurslaan, getoets. Die uitslag dui daarop dit is vol swaelsuur.

Dié water, wat met verskeie gevaarlike swaar metale waaronder arseen en nikkel besoedel is, gaan nie verdwyn nie. Daar moet dus nou gesoek word na oplossings.

"Dis 'n kwessie van tyd voor dié aspek 'n besprekingspunt op die internasionale aandelemarkte gaan word. Aandeelhouers kan begin om hul bestuurders te druk om hul gemors op te ruim," sê Turton.

'n Woordvoerder van die departement van minerale en energie, het gesê die regering reguleer die myne en stuur gereeld aanmanings oor die water wat na die oppervlak deursypel.

■ In 2002 het suur mynwater by Harmony Gold se Randfontein Estates-myn begin uitloop (sowat 'n kilometer van die Robinsonmeer) en in die Tweelopiespruit beland.

Die giftige water het tot die dood

... Volgens Turton is die mure van die „baie hoë gebou met 'n baie diep fondasie" in die Johannesburgse midestad waar die suur water reeds begin deurslaan getoets
(24 April 2007; E Tempelhoff, Beeld p. 3)

... According to Turton are the walls of the „very high building with a very deep foundation" in central Johannesburg where the acidic water already starts seeps through...

Statement made by Dr. A. Turton (CSIR) claiming that flooding has already taken place when in fact the water table remained at the same level where it was since 1976, i.e. nearly a kilometre below the surface of the CBD of Johannesburg.

Table of contents

Condensed overview on findings

1 Background

In August 2010, Prof Winde, head of the Mine Water Research Group at the Potchefstroom Campus of the North-West University was approached by two major banking groups to assess the geotechnical risks to their buildings in the Johannesburg CBD posed by the rising water in the Central Basin Mine void. This request was in response to a number of partly sensational media reports based on the concern of some scientists that acid mine drainage will severely affect the stability of the CBD infrastructure, especially high rise-buildings with deep basements (e.g. Van Vuuren, 2011 verbatim quoting Prof. TS McCarthy from the University of the Witwatersrand in Johannesburg). In this regard the new Standard Bank administration building featured prominently. Other reports predicted adverse consequences that included the flooding of the tourist mine shaft at Gold Reef City, health effects, formation of sinkholes in the CBD due to AMD solution of dolomite and pollution of the Vaal Barrage. A variety of dates when the water will surface were predicted with most targeting mid-2012.

Given the short period left to decant and pressurized by NGO's, activists, private industry and the media Government established an Inter-Ministerial Committee (IMC) on Acid Mine Drainage that, in turn, appointed a team of 27 experts to advise Cabinet on a course of action. The team of experts reported back twice during late 2010, advising that the water level of the Central Basin should be maintained below what was termed 'Environmental Critical Level' by pumping, if necessary, from underground pump stations. The discharge would be neutralized before being released. After the report was tabled to cabinet in early 2011 a total of R 225 m were allocated to address the problem.

AMD stems from the gradual closure of mines along a 44-km-long E-W running zone South of Johannesburg known as 'Central Rand' since the 1970s. After Durban Roodepoort Deep (DRD) located in the far western part of the Central Rand closed active underground operations in 1995, the only remaining active mine was the over 100-years-old East Rand Proprietary Mines (ERPM) in the far eastern part of the goldfield. The successive closure of mines resulted in the flooding of those voids where pumping stopped as water naturally continued to flow into the mine void. Following an accident at the pumping shaft of ERPM as last pumping mine, the system of interconnected mine-voids (termed 'Central Basin') started to fill. Initially the flooding was uneven across the Central Basin as the connections between the different mines were at different levels resulting in hydraulically separated sub-basins. However, since mid-2010, all the mines are hydraulically interconnected forming a single basin except for a small eastern section of ERPM that was disconnected from the other mines following a R29m plugging programme that originally aimed to extend the life of the mine to 2011.

2 Scope of project

A desktop study by the Mine Water Research Group evaluated available data from a number of studies done for different purposes, from August 2010 to April 2011. A scenario-based assessment model was developed to determine the risk a full mine void would pose to the CBD infrastructure.

Somewhat exceeding the scope of a desktop study, primary data supplied by the mining industry were also used, notably time series of pumping data and current mine water level measurements. The main parameters investigated included a variety of elevation data sets, mine void characteristics (volume, structure, shape, interconnectivity, closure rate), sources, pathways, quality as well as volume estimates of the ingress. Where necessary interrelationships between these parameters were analysed with the ultimate objective to predict at what elevation the final mine water level in the completely flooded basin will be in relation to basements in the CBD assuming that no intervention will take place to keep the water artificially at a lower level (‘Do-nothing scenario’).

3 General findings

- As dolomitic ground instability, in the form of catastrophic sinkholes affecting central Johannesburg, featured prominently in media reports (even providing elaborate sketches on how this will happen) we would like to point out that no dolomitic formations occur below the Johannesburg CBD.
- Since June 2010 the water level measurements across the Central Basin reflected a single water level that demonstrates sufficiently high interconnectivity between the different sub-voids to allow for a single decant point (e.g. a shaft) to control the water level of the entire Central Basin.
- The average water table rise since June 2010, when all sub-basins finally merged into a single large basin, reduced from an average of 0.55 m/d in the former central sub-basin to 0.37 m per day in the larger basin.
- The impact of rainfall on the rate of rise is less drastic and direct than previously suggested (exponentially rising rate where proposed) as rainfall-dependant ingress accounts for only a relatively small portion of the total ingress into the void. This explains the rather moderate effect even the exceptionally heavy rains that occurred since late December 2010 had on the observed rise of the mine water table causing a temporary increase by 37.5% (= 0.12 m/d).
- The lowest-lying shaft that is still open and connected to the Central Basin will serve as decant point. Provided that all ingress can overflow (i.e. the full decant volume be accommodated) this shaft will ultimately control the elevation of the

final mine water level in the Central Basin. At an estimated decant rate of 30-40 Ml/d this is most certainly the case. At the current status quo this decant point will be the Cinderella West Ventilation shaft at ERPM (near Boksburg) located at *1613.7 mamsl*.

- Given the average water table rise per day, the elevation of Cinderella West shaft, and assuming that the exceptionally high 2010-2011 rainfall will not be exceeded the predicted decant date will take place mid September 2013, more than a year later than predicted.
- In addition to decant from the shaft some mine water may diffusely seep out of the flooded mine void towards low lying areas below the decant level such as valleys of nearby streams. This is particular likely to happen where transmissive geological features such as weathered dykes, fault lines, fractures etc. provide a pathway that hydraulically links the flooded void with the surface. The E-W profile in Figure 1 indicates some potentially affected stream valleys in the far east of the Central Rand.
- Where the Main Reef outcrop zone, which historically was mined from surface to a depth of 40 m, is topographically low enough mine water rising in the underlying, deep void may spill over and saturate the unconsolidated fill material frequently found in this zone.
- Where this results in liquefying the base of large tailings dams, or other structures it may pose a geotechnical hazard. As failure of slimes dams in the Central Rand area have occurred in the past and, for example, where tailings deposits have been incorporated as used to support the M2 highway for example the risk associated with the underground (and thus invisible) diffuse outflow of mine water into the disturbed outcrop zone should be further investigated.

4 Key findings

- *Flooding risk*: Using the pile levels of the ABSA tower East as the deepest of the Bank buildings considered in the Johannesburg CBD; it was calculated that the maximum elevation to which the mine water table can rise in the Central Basin mine void is 90 m below the base of these piles. For the new admin building of Standard Bank, which according to the latest issue of You Magazine is already being flooded, the safety margin is 106 m. This water level height was determined using the surface elevation of the lowest shaft connected to the filling void (Cinderella West shaft of ERPM at 1613,7 mamsl) as the ultimate height of the water table in the mine void. As this safety margin can accommodate all possible uncertainties associated with the data used and the conceptual model it was concluded that ***no risks of mine water flooding any basement structure in the CBD of Johannesburg exist*** (Fig. 1).

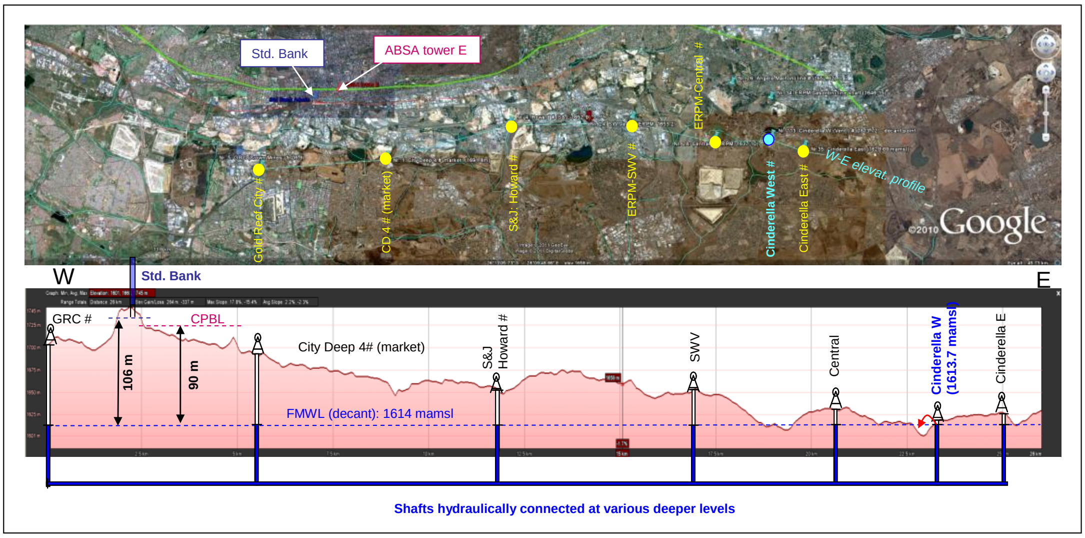

Fig. 1: The elevation of 7 open shafts along a E-W profile in relation the decant level and the pile base levels of ABSA (CPBL) and Standard Bank indicating a safety margin of 90 m and 106 m respectively.

- *Estimated decant volume:* The analysis of pumping volumes of the Central Basin mines and comparing these with ingress sources, indicated that decant volumes from natural and artificial sources will be in the order of 30-40 Ml/d in contrast to the 60–100 Ml/d cited in some scientific studies and most media reports. It is expected that, as the mine void fills, some ingress sources will finally be below the recovered mine water table and thus no longer able to contribute to ingress. Therefore, post-flooding ingress volumes are generally expected to be lower than pre-flooding pumping volumes which are commonly used to estimate future decant rates.

  The reduction of ingress is larger the higher the mine water table rises. Thus, a decant level closer to surface may result in a (significant) natural reduction of the volume of water that needs to be treated reducing long-term costs. If this concept of natural decant reduction were to be utilized a balance between the benefits of reduced decant volumes/ treatments costs and possibly increased risk to surface structure needs to be found. Considering only a partial reduction of the diffuse groundwater-related ingress from the fractured aquifer through the Main Reef outcrop zone a decrease of some 4 Ml/d is estimated. This does not take the reduction of stream-loss into account which is likely to occur in some streams especially in the low-lying eastern part of the Central Rand.

  The much reduced prediction of future decant volumes is likely to influence the treatment options, as the feasibility of certain processes is dependent on a high volume throughput.

- *Additional ingress sources:* The possible contribution of tailings reclamation activities on surface has, so far, not been considered. Tailings reclamation is particular pronounced in the Central Rand, where it indeed started in the mid-

1980s as this is the oldest of all goldfields where many old slimes dams with relatively high gold grades occur. The associated impact on the flooding of the mine void is likely to be considerable as the old mine dumps are hydraulically mined using high-pressure water cannons. This introduces large volumes of additional water into a highly disturbed area where surface mining and subsequent filling resulted in exceptionally high infiltration rates. In some cases reclaimed tailings along with process water have, for decades, been disposed of underground, increasing ingress while simultaneously reducing the void space.

- *Plastic void closure:* The Western Basin, which is currently decanting, was used as an analogy for what can be predicted for the Central Basin due to similarities between the two basins. The theoretical mining void volume of both basins is reasonably well known from mining records. However, analyses of water table rise vs. ingress volume indicated that the mine void of both basins has been reduced by approximately 75%. The most likely cause of this reduction is the time- and depth-related plastic closure of stopes which, in turn, significantly reduces the exposure of unmined ore as a major cause of AMD.
- *Decant water quality:* The water quality, using uranium (U) as the indicator for the past 9 years of water decanting from the Western Basin has, for example, improved from over 6000 μg/l U during the initial phase of the decant to presently 100-200 μg/l U, much faster than predicted. One of the reasons could be that less U is available in the void to be mobilized.

  In addition, the pH of the Western Basin void water directly at the outflow point is close to neutral suggesting that little U, and other heavy metals, can be liberated through dissolution of minerals by acidic mine water. Thus, the main sources of decant pollution are perhaps not underground but on surface relating to tailings and other mining residue deposits covering a large percentage of the surface catchment of this basin. This has implications for the Central Basin, where much of the small headwater catchments of streams are covered by mining residue, resulting in much of the ingress arriving in the void already being polluted. A significant proportion of the void water is attributable to tailings-polluted stream water lost to the mine void where old surface diggings as well as transmissive geological features such as dykes, faults and fractures are crossed. Direct seepage from slimes deposited on top or up-gradient of the mined outcrops zones may also contribute considerably to ingress especially where these dams are still active as e.g. at the Nasrec site where reprocessed tailings are still deposited.
- *Decant-induced stream pollution:* Where the mine void fills to levels above these streams, it may reverse the stream loss and stop further ingress of polluted stream water by keeping the latter on surface. As the decant and the surface run off both display similar poor quality, the net pollution effect of mine water diffusely entering the stream may not be as dramatic as predicted. In fact, much of the environmental damage observed in the Tweeloopiespruit at the West Rand may

be caused by the spatial concentration of polluted water from a large area which is discharged into a single, small stream that has almost no dilution capacity. This concentration is caused by the mine void collecting all infiltrating water that would otherwise drain into various streams across the area and discharging it via a single outflow point. Thus, it may be possible to resolve the pollution problem associated with mine water spilling from the flooded mine void on surface rather than underground. Given the technological difficulties for preventing underground pollution, there is reason for optimism.

Due to the above and the fact that during mining pumped mine water was disposed of in nearby pans and rivers, it is suggested that the Central Basin mine void decant will, in the long-term, not have a much greater impact on the river systems than was the case during active mining. In fact, if the water in the mine void stratifies, as observed elsewhere, cleaner water may decant on surface after the initial flush of highly polluted mine water subsided.

- *Groundwater pollution and associated radon exposure*: Where the rising mine water will come into contact with the surface aquifer, U contamination is to be expected. The associated radon risk needs to be assessed especially for informal settlements where the radioactive gas (formed ongoingly through the radioactive decay of uranium) easily can accumulate in low-lying, poorly ventilated shacks which often lack a concrete floor that could limit a radon influx. As a leading cause of lung cancer in uranium miners, radon exposure constitutes a severe health risk.
- *Radon exposure via shafts:* Since shafts are directly connected to the mine water and act as preferred conduits for equalizing barometric pressure differences between the void and the surface, it is likely that radon can reach the surface relatively quickly and well before it decays (the half life is 3.8 days) even in areas where the water table will remain deep below the surface. Thus, radon is likely to already escape from shafts well before the flooding of the mine void is complete. This renders shafts potential hot spots for Radon exposure of surrounding areas. With over 100 shafts distributed across the mining belt the potential for Radon exposure is considerable. The identification of affected areas may be difficult where old shafts have been covered with soil, or other material. These invisible, odourless radon hot spots are particularly dangerous.
- *Decant impacts on dolomite:* If AMD is allowed to decant freely from the Central Basin, it will have, depending on where possible diffuse decant occurs, an approximately 5-km-long surface pathway before it reaches the nearest dolomite outcrop. It has been proposed that increased rates of dolomite dissolution will result in the formation of potentially catastrophic sinkholes as experienced in the Far West Rand goldfield. In this context it should be noted that the primary cause for sinkhole formation in mining areas is not the accelerated dissolution of dolomite by acidic mine water but the lowering of the groundwater tables

following the dewatering of karst aquifers in an attempt to reduce ingress into the underlying mine void. By lowering the groundwater table as sub-surface erosion base surface water percolating through sediment filled pre-existing karst voids can erode the unconsolidated fill material into deeper lying karst receptacles. This increased rate of subsurface erosion is the actual source of sinkhole formation and thus unlikely to occur south of the Central Rand.

How far acidic mine water would indeed increase dolomite dissolution is unclear. Attempts by gold mines to use the abundantly available dolomite for neutralizing AMD failed as the dolomite chips were, within days, armored with inert coatings of iron hydroxides preventing acid neutralization and further dissolution of the dolomite chips.

5 Conclusions and recommendations

Given the shortage of water in Gauteng as the most water-stressed province in South Africa that relies, heavily, on water imported from Lesotho at great costs, the anticipated decant from the mine void should be seen, not as a threat, but rather as an opportunity of using water which for years went unused to fill the void. Untreated acidic mine water has been used in the past by municipal sewage works in the Central Rand to aid nitrification. Given the number of sewage works in Johannesburg and the volume of sewage to be treated this alone could perhaps accommodate all the decanting water resulting in no treatment costs while saving clean water otherwise used for this purpose. This is one example from a range of other possibilities that should be explored in order to arrive at low cost, low-energy solutions that are sustainable in the long-term as opposed to the currently proposed high-cost, high-energy, end-of-pipe pump-and-treatment-option subsidized *ad infinitum* by society.

In this context it appears that the AMD report to the Inter-Ministerial Committee and Cabinet, concerning the Central Rand, lacks a thorough analysis of available data and leaves many crucial aspects superficially covered. This includes key issues such as the volume of the expected decant, the compilation of sources of the ingressing/decanting water, water quality and relationship to rainfall, the rate of rise of the mine water table and date of decant as well as the spectrum of associated risks.

On the one hand it becomes difficult to avoid the impression that the report is a pre-mature, somewhat hasty response to a largely media-and interest group-driven campaign that appears to have inflated, misrepresented and exaggerated possible risks associated with the filling of the mine void. It has been suggested, in public, that this may be aimed at creating a ‘knee jerk’ reaction by Government where a sense of urgency prevents sound scientific analyses and streamlines approval for an expensive ‘solution’ favoured by some role players from the private sector.

On the other hand it needs to be stressed that the conclusion that the ‘do-nothing option’ would not result in any flooding risk for the CBD, nor in much greater pollution of surface water should not be used as an excuse to continue avoiding the much needed rehabilitation of mining-affected environments in one of the most densely populated areas of South Africa.

In order to pro-actively address the identified risks of flooding-induced subsidence of structures in the low-lying outcrop zones and the exposure of residents especially in informal areas to radon even before flooding is complete it is recommended to urgently conduct in-depth investigations to quantify these risks.

0 Technical information

(i) Composition of the research team

An overview on core members of the research team is provided in Tab. 1.

Tab. 1: Core members of the research team including their function, responsibilities and research focus within the project

Name of researcher (qualif.)	**Function in team**	**Main research focus/ responsibility**
Prof. EJ (Leslie) Stoch (DSc Agric.)	Principal researcher	Secondary data retrieval, capturing and analyses (contact to DRD), media reporting analysis, alternative solutions, report review
Mr. Ewald Erasmus (BSc Hons. Geol./ Geochem, Pr. Sc. Nat.)	Principal Researcher	Geology, Hydrogeology, Hydrochemistry, secondary data retrieval, literature study, concept development administrative project support , report review
Mr. Emile Hoffmann (M/ PhD- student Potchefstroom)	Researcher	Geographical Information Systems (retrieval and evaluation of elevation data, spatial data processing and DEM generation, GIS modeling, Google Earth based analyses, map generation)
Mr. Aljoscha Schrader (M Geography, PhD- student)	Assistant	Archive work, literature retrieval, data capturing
Prof. Frank Winde (Dr. rer. nat. habil., Pr. Sc. Nat.)	Principal Researcher/ Team leader	Hydrology, hydrochemistry, data analyses, conceptual model development, data interpretation/ evaluation, project administration, quality control, report writing,

(ii) Lists of Abbreviations

(excluding titles and commonly used abbreviations)

ABSA	Amalgamated Banks of South Africa
AMD	Acid mine drainage
CB	Central Basin
CBD	Central business district
CD	City Deep Gold Mine
CD-NGI	Chief Directorate National Geospatial Information (new name)
CDSM	Chief Directorate Survey and Mapping (old name)
CGIS	Corporate Geographical Information System
CGR	Crown Gold Recovery
CGS	Council for Geosciences
CM	Crown Mines
CMR	Consolidated Main Reef Gold Mine
Cnr.	Corner of two street crossing each other
CPBL	Critical Pile Base Level
CR	Central Rand
CRG	Central Rand Gold Ltd.
CSIR	Council for Scientific and Industrial Research
DEM	Digital Elevation Model
DRD	Durban Roodepoort Deep Gold Mine Ltd.
DWA	Department of Water Affairs
E	East
EB	Eastern Basin
ECL	Environmental Critical Level
EMPR	Environmental Management Plan Report
ER	East Rand
ERGO	East Rand Gold and Uranium Operation
ERPM	East Rand Proprietary Mines Ltd.
FEV	Far East Vertical shaft
FWR	Far West Rand
FMWL	Final Mine Water Level
GE	Google Earth
GIS	Geographical Information System
GM	Gold Mine
GW	Groundwater
IMC	Inter-Ministerial Commission
JHB	Johannesburg
JSE	Johannesburg Stock Exchange
Kby	Kimberley
Lidar	Light detection and ranging

MAP	Mean Annual Precipitation
M	Mega (1 million)
Mio	Million
MiR	Middle Reef
Ml	Mega litres (1 million litres)
MR	Main Reef
MRL	Main Reef Leader
N	North
NR	North Reef
PBL	Pile Base Level
R	Reef/ reef package
Rn	Radon
RGM3	Rand Gold Mine Milling and Mining
S	South
SEV#	South East Vertical shaft
SD	Slimes Dam
SR	South Reef
SRTM	Shuttle Radar Topography Mission
SSR	South South Reef
Std. Bank	Standard Bank
SWV#	South West Vertical shaft
t	ton(s)
vs	versus
V	Vertical (shaft)
W	West
WITS	University of the Witwatersrand
WL	Water Level
WR	West Rand
WB	Western Basin

(iii) Used units and symbols

#	shaft
/	divided
:	divided, ratio
µg/l	microgram per litre (= parts per billion, ppb)
a	years
cm/d	centimetres per day (rate of water level rise)
d	days
ft	feet (= 0.3 m)
ha	hectares
km	kilometres
km^2	square kilometres (= 1 million m^2 = 100 ha)
m	metres
m^2	square metres
m^3	cubic metres
mamsl	metres above mean sea level
mbd	metres below datum
mbs	metres below surface
Ml	million litres
Ml/d	Megalitres per day (1 Ml/d = 1000 m^3/d = 1 million litres per day)
mm/a	millimetres per year (e.g. for rainfall volume over an area = l/m^2 x a)
Mm^3	million cubic metres
oz	ounce
t	ton(s)
t/a	tons per year
x	times

(iv) List of tables

(Total of 28 x tables)

(v) List of figures

(Total of 75 x figures)

(vi) Glossary of key concepts

Elevation

Alternative term: ‘altitude’, height of a site above a certain benchmark level. For surface areas this benchmark is usually the mean sea level (msl) and elevations are given in metres above msl (mamsl). For deep level mining however, the reference to the mean sea level frequently results in elevations below the msl which can lead to confusion between m above msl and m below msl. Therefore an alternative benchmark has been introduced termed ‘datum’. This is an elevation set at 6000 ft above mean sea level well above the highest surface area in the Witwatersrand goldfields allowing to express all depths in metres below datum (mbd) avoiding possible confusion associated with using mamsl.

Elevation data

Data from different sources indicating the altitude of sites above benchmark heights. For topographic maps most data sets are given in mamsl. Elevation data are generated either by trigonometric surveys producing the most accurate and reliable data for single points such as trigonometric beacons, shafts etc. or by remote sensing techniques that can cover larger areas. The later elevations are usually referenced by using ground based surveys and are generally less accurate. In many instance area elevations are derived from interpolations between different points of known elevations (‘spot heights’). Most of the interpolations are now done by computerised programmes generating Digital Elevation Models (DEMs) which allow for any point of the covered surface area to determine its (interpolated) elevation.

Aquifer

Rock containing freely available water in fractures, pores etc. Since the water is stored underground this water is termed groundwater.

AMD

Acid Mine Drainage – natural water that comes in contact with oxidised pyrite (iron sulphide, FeS_2) exposed to oxygen by mining (either in the mine void or on surface in the form of tailings, rock dumps etc.). In the presence of oxygen the reduced sulphur forms sulphate which in turn generates sulphuric acid that turns the water acidic. Parts of reduced iron from the sulphide change from ferrous to ferric Fe and binds to hydroxide (OH^- a natural dissociation product of water). The formation of iron hydroxide releases additional H^+ ions that increase the acidity of the water. The freshly formed FeOH is insoluble and precipitates as yellow boy covering surfaces in streams with an orange-brown layer that soon crystallizes forming a rock like sediment.

Datum
See elevation

Mine void
Complex network of vertical and horizontal tunnels used to access underground ore deposits. The overwhelming majority of the void volume is made up of the excavated ore body (mined reef) while shafts, haulages etc. (termed 'access infrastructure 'or 'off-reef development') in deep level gold mining commonly account for 10-15% of the total void.

Mine water
Water accumulating in the mine void resulting in the gradual flooding of the void.

Service water
Water added by mines to underground operations for drilling, cooling, dust suppression, cleaning etc. additionally to naturally ingressing water.

Ingress
Inflow of surface water or groundwater into the underground mine void.

Decant
Overflow of mine water from the flooded mine void onto surface or into adjacent sub-voids.

Mine void flooding
Natural process of surface and groundwater seeping into the mine void and gradually filling up available empty space, stops when connecting link to surface is reached via which all ingressing water can overflow.

Basin
In this report used to refer to a system of abandoned hydraulically interlinked mine voids. Basins are named after the associated goldfield e.g. the Central Basin is the mine void of the Central Rand.

Sub-void
In the context of this report an underground mining void generated by a particular gold mine which forms part of a larger system of mine voids, which may or may not be hydraulically connected.

Sub-basin
A number of hydraulically interlinked sub-voids within a larger basin.

Stope

The part of the mine void where the ore is excavated accounting for some 85 to 90% of the total void volume. Stopes are the least permanent structures of the void and tend to close soon after active mining stops.

Reef

A concept is used somewhat ambiguously. In stratigraphy a reef refers to a layer of rock containing the commodity of interest often only contained in smaller layers within the strata which are also termed reefs. An example for the CR is the Main Reef which is the term for a series of reef bands such as the North Reef, South Reef and Main Reef Leader as well as the name of one of the reef bands. In mining terms a reef refers only to the part of a reef band that contains payable Au grades commonly confined to a small part of the total thickness of the reef band.

Outcrop

The zone where the underground reef intersect with the topographic surface

Tailings

Milled and leached ore from which the Au is removed and which is subsequently hydraulically deposited on so called slimes dams or tailings dams. At Au grades of 5-10 g/t the mass extracted from the ore is negligible. Thus tailings are a good representation of the mined out volume. However, owing to the milling which reduces the density of the original ore from 2.65 to 1.45 t/m³ the volume of tailing deposited on surface is larger than the volume of the mine void.

Tonnage

In the context of this report the mass (in tons) of milled ore (tailings).

Slimes dams

Hydraulically deposited tailings.

Creep

In the context of this report the ability of highly stressed hard rock to plastically deform through bulk expansion without forming cracks or fractures.

Boundary pillar

Remaining rock / ore between underground mines that marks the mine lease boundaries projected vertically down from surface.

1 Introduction

Various news media (print, television and radio broadcasting) reported since late 2009, based on statements from scientists, that acidic mine water in the flooded mine voids underneath Johannesburg will soon reach the surface with a range of number of adverse consequences. Apart from the flooding of an underground tourist mine at the Gold Reef City complex and potential health effects on residents exposed to acid mine drainage (AMD) this also included predictions that mine water will eventually compromise the geotechnical stability especially of high rise buildings in the central business district (CBD) of Johannesburg (JHB). This, in turn, was due to the acidic nature of mine water presumably able to corrode underground structures of concrete and steel and thereby destabilising the affected buildings. In this context the new administration building of Standard Bank in the southern part of the CBD was specifically mentioned.

Following a Card Blanche report end of July 2010 Standard Bank approached Prof. Winde to assess the extent of such risks. After an initial meeting at the Rosebank branch of Standard Bank it was agreed to conduct a desk-top study into this matter. Shortly thereafter Prof Winde received a similar request from ABSA headquarters in Johannesburg. In order to avoid duplication and pool resources the two banks were put in contact with each other and subsequently agreed on a joint study. This study commenced mid August 2010.

Owing to the urgency of the matter an intensive initial research phase followed that primarily focused on the quantification of the proposed flooding risk for underground building structures. The first results of this investigation were presented on 15 September 2010 at a meeting in the Standard Bank branch in Rosebank (Johannesburg). Following the introduction of the developed conceptual model underlying the risk assessment the major findings regarding a possible flooding risk were discussed. Based on various sets of preliminary elevation data it could be shown that the risk for underground structures of the selected bank buildings in the CBD of Johannesburg to get in contact with acidic mine water from the flooded mine void is very low. This is mainly owing to sufficiently large elevation differences between the lowest levels of the buildings structures (pile base level) and the highest possible elevation the water table in a completely flooded mine void is likely to reach.

Possible uncertainties were addressed in the phase following the first presentation. In addition this phase was also used for assessing the accuracy of used elevation data as well as obtaining new elevation data sets with improved vertical resolution. The latter included the purchase of Lidar data with 0,25 m and 1m vertical resolution for the entire study area, surveyed shaft elevations as well as latest water level elevations from the different sub-voids.

A second interim report was provided on 24 January 2011 at a meeting in the headquarters of ABSA Bank in central Johannesburg. At this meeting it was agreed to finalize the project and submit a written report on the project results by mid March 2011.

Delays of several months in delivering the ordered Lidar data (from the two Metro municipalities of Johannesburg and Ekhuruleni) as well as subsequently discovered quality problems with the delivered data sets caused a delay in compiling the report. Also, the latest data on mine water levels in the different sub-voids were only made available end of March 2011. Further delays were caused by the belated release of the report of the Expert team of the Inter-Ministerial Committee on Acid Mine Drainage (IMS- AMD) that was analysed as part of the project.

2 Project aim

The main aim of the project is to quantitatively assess, via a desk-top study, the proposed risks that mine water from the flooded Central Rand mine void may destabilise selected buildings of Standard Bank and ABSA in the CBD of Johannesburg through flooding and corroding underground structures required for the geotechnical stability of the buildings in question.

In addition, a number of issues directly or indirectly linked to the above risk assessment are to be addressed including the identification of the main ingress sources, their respective contribution to the filling of the void as well as determining the dynamics of the mine water table and its governing factors.

Lastly, a section was added outlining other risks possibly associated with the flooding of the mine void which may directly or indirectly impact on Standard Bank and ABSA. This includes geotechnical impacts on the Bank's property elsewhere in the study area (i.e. outside the CBD) as well as on the stability of highways and slimes dams in the immediate vicinity to the CBD owing to flooding related seismicity and subsidence.

Also, the risks associated with negative public perceptions following media reporting on widespread environmental pollution in Johannesburg and collapsing surface structures are discussed regarding their potential to adversely affect the corporate image and business confidence.

3 Methods of investigation

Before reviewing any existing literature, data or other sources of information on the issue it was decided to independently develop a conceptual risk model during an initial brainstorming exercise in order to avoid being contaminated by pre-existing ideas, and arrive at a fresh and ideally original approach to the problem.

Subsequently, a review of pertinent literature was conducted essentially aimed at verifying the developed concept of the model as well as the individual model components. Apart from a few published sources most of the relevant information is unpublished and had to be retrieved from various sources. Most of the sources are in the form of consultant reports, EMPRs, data sheets, maps, satellite images, personal communication, etc. While most available sources were dealing with understanding the reverse process, i.e. the influx of water into the mine void (recently termed 'ingress'), many provided relevant data and insights that informed our model on mine water ingress and the associated flooding risks for buildings. In exceeding the scope of a desk top study, primary data have been generated too. This includes the quantification of certain ingress areas per mine void as well as the calculation of pumping-volume ratios used to assist in identifying ingress sources for the different sub-voids.

As the developed risk assessment model is essentially based on differences in elevation the retrieval of accurate and reliable elevation data was of crucial importance for the project. This includes elevation data for the topographic surface across the entire study area, the underground structures of the selected buildings as well as water levels in different parts of the mine voids, hydraulic void structures such as links between different sub-voids, collars of monitoring shafts, collars of potential decant shafts etc. The data have been subjected to quality assessments using surveyed spot heights such as trigonometric beacons provided by the CD-NGI on 1:50,000 topographic maps of the study area as well as shaft collar elevations of five monitoring shafts (surveyed by the individual gold mines) as benchmarks. Elevation data of the lowest lying underground structures (pile base levels) for the selected buildings of Standard Bank and ABSA in the CBD of JHB were requested from both banks. While in both cases detailed plans of the underground structures were provided they did not display the requested elevations. Attempts to retrieve the PBL indirectly via depth calculations also failed. Subsequently estimates provided by McLoyd (2010) were used.

In addition, topographic positions for geological and mining data such as mine lease areas, outcrops of different reefs, location of monitoring shafts etc. had to be determined. As no coordinates were available this was frequently difficult to achieve. In most instances scanned maps were imported into Google Earth and/or GIS and subsequently georeferenced using visual comparison. However, maps from different sources indicating outcrops of reefs and mine lease areas were found to differ from each other resulting in some degree of uncertainty regarding the exact geographical location, shape and area of lease area for example. While shp files indicating old shafts for the whole Central Rand had been obtained, the lacking specification of names in the associated feature tables made it difficult to precisely identify the five monitoring shafts. Using Google Earth assisted to visually confirm the location of 4 of the five shafts. Coordinates of Howard shaft at Simmer & Jack as well as coordinates of the other monitoring shafts were eventually supplied by DRD.

Finally the elevation data have been imported into EXCEL and GIS for further modelling. In order to assist with the visualization of the different final mine water levels and their potential impact on the underground structures of the bank buildings a digital elevation model (DEM) has been used.

4 Sources of data and information

The following sources of information have been consulted:

(1) Unpublished literature

(2) Competent persons

(3) Published literature

(4) Newspaper articles

Tab. 4.1 provides an overview on sources of data and information used in the project.

Tab. 4.1: Contacted institutions and sources of data and information used in the project (in no specific order)

Project topic	Name of contact person	Affiliation	Type of data/ information
Mine void flooding	Mr. Kruger, K.	DRD	- monthly results of water level monitoring of 2 different shafts including provision of data of collar elevation of these shafts provided to Prof Stoch
WL mine void	Mr. Schweitzer, J		telephonic enquiry of E Erasmus regarding water level data in the Central Rand mine void
Ingress information	Dr. Coetzee, H.	CGS	Refusal to provide data or information as CGS Project no 5512 is still embargoed as work in progress (telephonic enquiry by L Stoch)
Seismicity	Mr. Chichowicz, A. Mr. Steyn, J.	CGS	Telephonic enquiry by E Erasmus regarding the seismicity effects of void flooding
Elevation of pile base level	Mr. MacLeod, N.	Standard Bank consultant	Provision of building plans for new admin building of Standard Bank in JHB CBD, estimate of pile base levels
Location of buildings/ Elevation data	Mr. Surban, N.	ABSA	Map indicating the location of ABSA buildings in the CBD of JHB, Provision of building plans for new admin building of Standard Bank in JHB CBD, estimate of pile base levels
Elevation data	Mr. Olivier, P.	Esor Franki Ltd.	Contructing company of ABSA complex, Pile base length of ABSA towers, conditions of bedrock/ groundwater presence
Elevation data, Mine void characteristics	Mr. Labuschagne, V. (cell phone: 082 494 3945)	DRDGold, chief surveyor ERPM	Elevations and coordinates of monitoing shaft used by DRD Gold, pertinent info to mine void connectivity, repeated teelphonic enquiries by E. Erasmsus and F. Winde
Elevation data	Mr. Duesimi, R. Tel. 021/ 658 4372	CD-NGI	Accuracy of CD-NGI elevation data, telephonic enquiry by E.Hoffmann
Elevation data	Security personnel of new administration building	Standard Bank	Height of basment, no of a above ground storeys, telephonic equiry by E. Erasmus
Eelvation data	Engineer at pile constructing company	Asakheni Consulting Engineers	Length of piles used at ABSA towers, general conditions of bedrock below basements in the CBD of JHB, telephonic equiry by E. Erasmus
Background information	Mr. Gravette, R.	Standard Bank	Provision of PowerPoint presentation to Cabinet by WRC and CGS
Topographic elevation data	Ms. Khosa, B.	Corporate GIS (CGIS) of Johannesburg Metro Municipality	Sale of lidar data (25 cm vertical resolution, ground level points) for 49 x tiles covering 9 km² each in the central and western part of the study area, totaling 441 km²)
Elevation data	Mr. Mathebula, T.	Corporate Geo-informatics of the Ekurhuleni Metro Municipality	Provision of lidar data (1 m vertical resolution, ground level points including ground control points and town survey marks) for the eastern part of the study area covering 500 km²
Topographic info and elevation data	Chief Directorate National Geospatial Information (CD-NGI, formerly CD Survey and Mapping)		Provision of digital contour data of 5 m and 20 m intervals for the topographic 1:50000 map sheets nr. 2627 BB; BD and 2628 AA; AB; AC; AD

5 Conceptual risk assessment model

5.1 Definition of 'risk'

While the term 'risk' is often used inconsistently and ambiguously and may change its meaning depending on the context used in, it can be generally defined as the exposure to the possibility of loss, injury or other adverse events. For the purpose of this report 'risk' is defined as the *probability* of a potential *hazard* (i.e. something that can cause harm) to actually result in an adverse event (modified after Wikipedia, 2011)

In the context of this investigation 'hazard' describes the (proposed) potential of acidic mine water from the completely filled mine void to flood basements and underground structures of selected buildings of Standard Bank and ABSA in the Johannesburg CBD. The flooding is thought to compromise the integrity and geotechnical stability of the associated building through the (proposed) ability of acidic mine water to corrode underground support structures such as piles.

5.2 Risk factors

The potential of this hazard to translate into a risk is determined by the following factors:

(i) The (vertical) height difference between the lowest underground structure (in this case the base of the concrete piles, termed pile base level or PBL) of the key building closest to the mine void (termed 'critical PBL' or CPBL) and the (predicted) final mine water table (FMWT) in the mine void once it is completely flooded.

(ii) The horizontal distance between the key buildings and potential outflow points of the flooded mine void.

(iii) The permeability of rocks separating the key buildings from the flooded mine void, i.e. the ability of the separating rocks to allow mine water to migrates from the void towards the buildings e.g. via fractures, underground mine workings, faults, dykes etc.

(iv) The ability of mine water to indeed corrode the underground concrete and steel structures associated with the identified key buildings.

5.2.1 Elevation difference between CPBL and FMWT

The elevation difference between the CPBL and FMWT is considered to be the most critical of all factors, as this difference determines whether or not mine water may be able to flow from the mine void to the bank buildings once the former is completely flooded. Ignoring for a moment all other possible factors controlling the probability of basement flooding, the latter can only possibly occur if the FMWT is high enough above the CPBL for the resulting hydraulic head to drive mine water from the void towards the banks. To be conservative, a flooding risk is already assumed if the FMWT is equal to the CPBL. Should this not be the case, i.e. the FMWT is below the CPBL, mine water could principally not flood the banks, i.e. no risk exists. However, in order to cater for possible data uncertainties it is assumed that the higher the CPBL is above the FMWT, the lower the risks will be (even though it should be zero as soon as the CPBL is marginally above the FMWT) as the associated safety margin increases.

For determining the lowest PBL the surface elevations of the identified key buildings were determined (using different sets of elevation data to ensure accuracy), from which the depth of the basement levels and the length of the deepest reaching supporting piles were subtracted. The latter values were determined using construction plans and information sourced from companies involved in the construction of the piles and the buildings.

For determining the lowest PBL the surface elevations of the identified key buildings were determined (using different sets of elevation data to ensure accuracy), from which the depth of the basement levels and the length of the deepest reaching supporting piles were subtracted. The latter values were determined using construction plans and information sourced from companies involved in the construction of the piles and the buildings.

5.2.2 Horizontal distance between key buildings and the mine void

Apart from the water table height related risks, the horizontal distance the banks are located from the mine void is also important, as the rocks in between could act as a potential obstacle for decanting mine water, to potentially reach the basements of the key building should a hydraulic gradient allow for such a scenario. Therefore, the closer an identified key building is to the mine void, the higher would the potential be for being flooded. For that reason the critical PBL is not necessarily the lowest PBL elevation but – if no or only small differences between the PBL of key buildings are found – the PBL of the building closest to the mine void would act as the CPBL.

5.2.3 Hydraulic connectivity between key buildings and mine void

This relates to the permeability of the rock separating the basements of key buildings from the mine void. In the absence of any detailed information it is assumed that this is the same for all key buildings. However, where special features such as connecting faults, dykes or underground workings exists, it increases the potential for flooding. In such case the PBL of the most exposed building will be used as the critical PBL (CPBL).

5.2.4 Corrodibility of underground building structures

Although (to our knowledge) no dedicated study has yet been conducted conclusively proving that for local conditions the quality of the final decanting mine water would indeed corrode solid concrete structures such as piles and basement casings, and if so at what rate and what period of time would be required for serious damage to occur, a conservative approach is applied. Hence, any contact between the mine water (acidic or otherwise) from the flooded mine void and the supporting underground building structures of the selected buildings is defined as a *hazard* regardless for how long and to what extent the contact occurs.

5.3 Conceptual model for determining the final mine water level

While the PBL is a fixed value that can be determined with a reasonable degree of certainty (even though some technical problems were encountered in retrieving the elevation in mamsl from provided building plans) this is somewhat different for the elevation of the final mine water table. As the filling of the mine is still in progress at the time of writing, the elevation of the final water level in the completely flooded mine void can only be predicted. This prediction in turn, is essentially based on a conceptual understanding of how the mine void fills and what governs the final water table elevation in the mine void once flooded. The main purpose of this report is to describe the facts, assumptions and data that underpins the final risk assessment as well as to point out associated uncertainties, and where possible to quantify them.

A first basic assumption for determining the final mine water level in the completely flooded mine void system (FMWL), is that mine water cannot possibly rise much above the level of the surface openings of the mine void (entrance level), as it would thereafter flow out onto the surface (termed 'decanting') preventing a further rise of the mine water level. In terms of the associated flooding risk that would mean, for example, that no basement flooding is possible if the surface (or entrance) level of the mine void is generally lower than the CPBL. Since this assumes that no anthropogenic intervention takes place artificially keeping the mine water level at a lower level (e.g. through pumping), the highest possible water level in the mine void represents a worse case scenario. As this do-nothing option of natural decant results in the highest possible risk for basement flooding to occur, the general approach of the applied risk assessment is conservative.

This conservative approach also applies to the handling of data uncertainty especially with regard to error margins associated with the different types of elevation data. Except for cases where elevation data are completely unreasonable, those elevations will be used that represent the highest possible flooding risks. That means, that CPBLs will be based on the lowest available elevation data (in this case on Lidar data as will be discussed later in the report) while the FMWL in turn will be based on the data set indicating the highest elevations (in this case Google Earth data as discussed later). This too allows for additional safety margins in assessing the risk of basement flooding.

Assuming that the mine void behaves similar to a confined aquifer, the water level of the highest-lying point or zone where water enters the mine void (‘ingress point’) defines the ‘piezometric surface’ i.e. the level to which water in a borehole or shaft could possibly rise (Fig. 5.1).

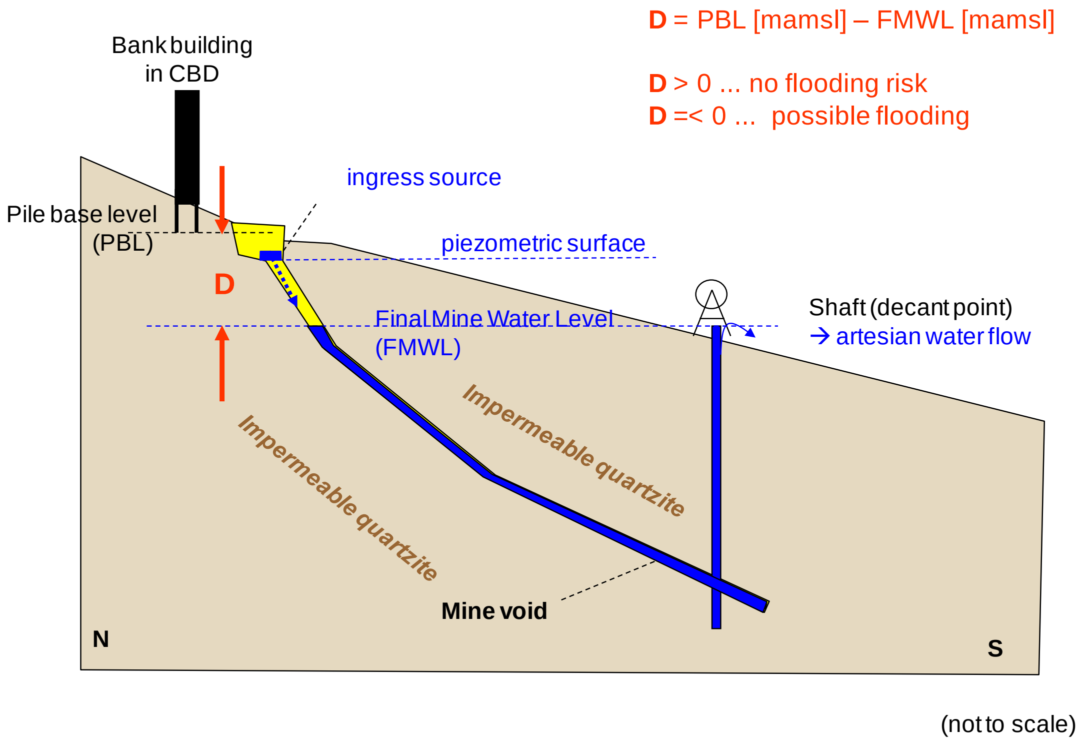

Fig. 5.1: Simplified N-S cross section of the mine void illustrating the concepts of the critical pile base level (CPBL), maximum final water level in the mine void (FWL) and the artesian water flow through lower lying shafts/ boreholes owing to hydraulic head at the highest-lying ingress source. (The depicted relation between the PBL and MFWL constitutes a no-risk scenario. This is for demonstration purpose only and may not reflect actual conditions at the study sites.)

For determining the final mine water level (FMWL) several factors needs to be taken into account in addition to the elevation of the ingress sources, including:

(1) The *elevation of potential outflow points* such as shafts, boreholes or topographically low-lying areas such as river valleys connected to the mine void (Are some shafts or other outflow point located below the MFWL and would they be able to accommodate the volume of out-flowing mine water?).

(2) The *hydraulic properties of the mine void*: the main question for assessing the flooding risk is whether the water level across the 44 km-long can be controlled by a

single outflow point. This requires that the hydraulic interconnectivity controlling the lateral flow across the different sub-voids is equal or higher than the hydraulic conductivity of the near surface substrate that controls the vertical inflow into the mine void. Given a scenario where lateral interconnectivity is sufficiently high across the entire basin (i.e. all interconnected sub-voids) the slope of the water table (i.e. the hydraulic head between the ingress area and the decant point) needs to be established in order to arrive at an in between WL elevation that can be used for assessing the flooding risk for the bank buildings.

(3) Furthermore, in order to predict when the mine void will be completely flooded as well as to understand what factors govern the flooding dynamics (indicated by the changing rates of rise of the mine water table) the volume and hydraulic properties of the mine void needs to be established. In this regard a conceptual model was developed taking the void geometry, shape and volume as well as time-depended changes of the latter due to closure processes into account.

(4) *The final decant volume*: in order to assess to what degree potential decant points could accommodate the ultimate flow of water emanating from the flooded mine void, the volume of the latter needs to be determined. Commonly, decant volumes are assumed to be equal to ingress volumes with the latter often being derived from pumping volumes of mines before flooding took place. Sometimes this is also calculated based on estimates for the contribution of the different ingress sources. In this report it is proposed that the final decant volume after the complete filling of the void will most probably be lower than the ingress volume during the flooding. This is mainly due to a range of ingress sources likely to be cut off as the mine water level rises above them. Apart from preventing inflow from sources that are now on the same level as the mine water table, the rise in mine water levels may also result in reversed flow directions with mine water recharging former ingress sources.

(5) In addition to the general situation along the full strike length of the 44 km-long Main Reef outcrop the PFWL needs to be determined at large scale i.e. directly around the selected buildings. This takes the possibility of localised groundwater flow inversions due to the micro relief or the effects of slimes dams into account.

Once the PFWL has been established with a reasonably degree of confidence this will be used for assessing the flooding risk for underground structures of the selected bank buildings.

6 Characterisation of study area

6.1 Location

The location of the Central Rand (CR) goldfield as defined by the various lease areas of individual gold mines. Owing to amalgamation of mines, change in ownership and names etc. it is difficult to depict the location of all 52 mine that at one stage were active in the CR. For the situation in 1955, when the CR was at its peak Au production, mine boundaries area shown in Fig. 6.1.

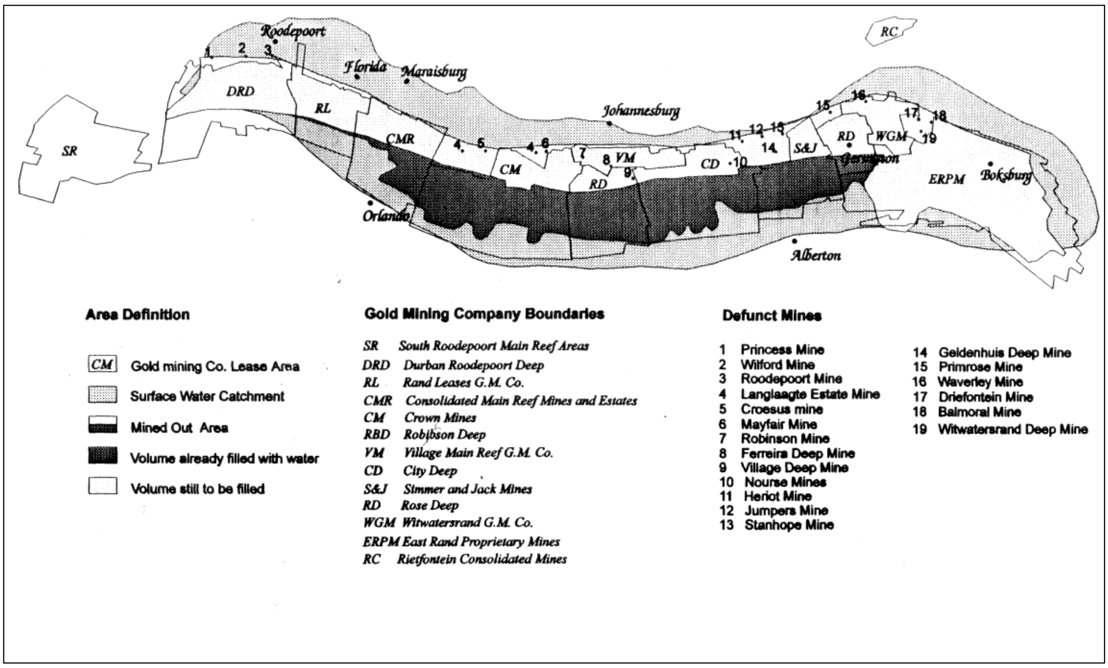

Fig. 6.1: Overview on mines which historically have been active in the CR (adopted from Scott, 1995)

Further consolidation reduced the number of mines thereafter. The mine boundaries for 2003 are indicated in Fig. 6.2.

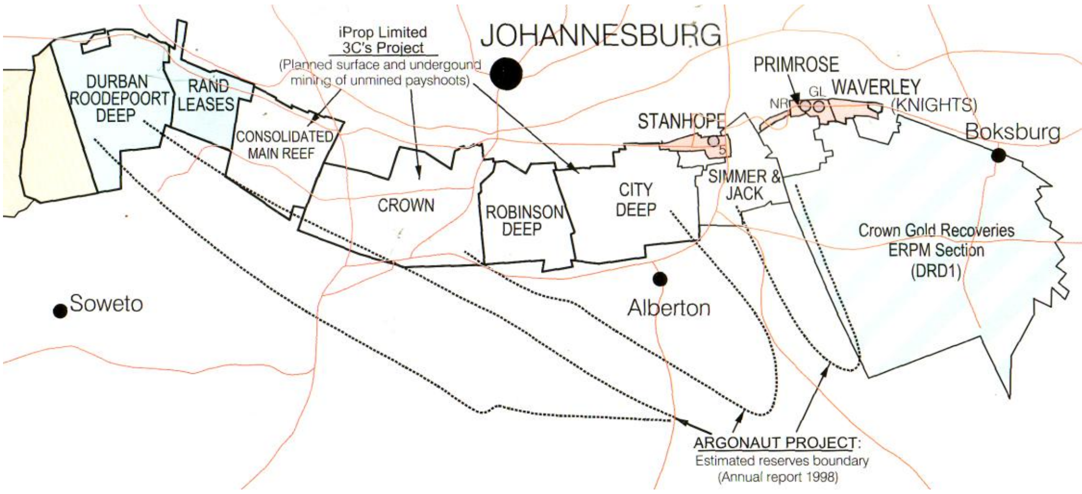

Fig 6.2: Mine lease boundaries including the prospecting area for the Argonaut project (adopted from Barker & Associates, 2003)

The historical changes in mining ownership seem to have resulted in a considerable degree of uncertainty regarding the exact number and shape of the individual lease area as illustrated by differences in the various references. In this report a map compiled from various sources is used.

According to Barker & Associates (2003) the CR is currently made up by a total of 11 x mine lease areas forming a 44 km – long belt running from Roodepoort in the west via Central Johannesburg to Boksburg in the east with an average width of some 2-3 km. The names of the various gold mine lease areas are given in Fig. 6.1 and 6.2.

The western boundary of the CR is marked by non-Au containing rocks of the Witpoortje horst resulting in a 2,5 km-wide gap to the nearest mine lease area in the West Rand NW of DRD.

After deep-level mining stopped here in 1998 the associated mine void (termed 'Western Basin') filled up within a span of 4 years and started to overflow in September 2002 with an average flow rate of 20-30 Ml/d impacting mainly on the Twee- Loopie Spruit that drains to the north of the continental divide.

A less clear distinction seems to exist between the CR and the FWR as the DRD shares a common boundary with Harmony GM (Doornkop section) in the SW. Doornkop is still an active deep level GM which is hydraulically linked underground to a number of active deep level mines in the South including the Cooke section of Harmony, Ezulweni Uranium mine (originally Western Areas GM later Harmony 4 shaft) and GFL Ltd.'s South Deep mine. As some of the latter GMs operate below dolomite they have to pump large volumes of ingressing dolomitic groundwater. Like many other mines in the FWR some of theses mines (such as South Deep) still have considerable life-spans of several decades left.

In the east the CR borders on the East Rand goldfield with surface mine lease areas of ERPM and Cons. Modderfontein being only some 700 m apart. Despite this vicinity on surface there is no underground connection between the voids in the two mining basins which are separated by solid rock of the so-called Boksburg gap of some 8 km width (V Labuschagne, 2011). Thus, the possibility of a proposed hydraulic link between the 2 basins can be discarded. (link proposed during the DWAF ingress study to centralise mine void water outflow)

Since 2009 all active underground operations in the Eastern Basin have stopped after the last operating GM (Grootvlei) was taken over, first by Pamodzi Gold and later Aurora mining, both whom experienced financial problems. Owing to the latter, Aurora switched of underground pumps in 2010 which delivered some 60-70 Ml/d of mainly dolomitic water to the Blesbokspruit.

Since then the eastern basin, which is largely covered by water rich dolomite, is currently in the process of being flooded. Decant will affect the Blesbokspruit and the Marievale wetland protected under the RAMSAR convention.

Owing to the unmined rock of the Boksburg gap there is no hydraulic connection between the Eastern and Central basins.

To the north the CR is defined by the outcrop of the MR, north of which only non auriferous strata are found. One exemption is the Rietfontein Cons. Mine in the eastern part of the CR forming an isolated island to the north of the mining belt.

The southern boundary of the CR is defined by the southernmost south dipping reef, and the location of shafts needed to access the deepest possible reefs.

However, below the depth to which the reefs were mineable when mining ceased, the reefs do continue. Improved technology and a significantly higher Au price compared to the 1970s when most mines closed, prompted DRD in 1998 to explore the feasibility of accessing these reefs through ultra deep mining (Argonaut project). After mining rights apparently expired under the new mining law, the Argonaut project appears to have been abandoned. Thus a large scale reactivation of deep level mining in the CR in the near to medium future is unlikely.

6.2 Natural conditions

6.2.1 Geology, relief and hydrography

As topographic elevation is of crucial importance for the risk assessment the factors determining the relief of the study area as well as direction and rate of water flow on surface and underground are briefly discussed.

Located on the large interior plateau of the Highveld that covers most of South Africa the base of the up to 200 m high E-W running ridges and hills of the Witwatersrand are located at an elevation of approximately 1700 mamsl.

The regional drainage pattern is largely controlled by the outcrop of weathering resistant, quartzitic rocks of the Witwatersrand Supergroup forming the continental divide that separates the Limpopo watershed in the N (draining towards the Indian Ocean) from the Orange basin in the south discharging into the Atlantic Ocean. To the south of the Witwatersrand hills E-W running outcrops of weathered dolomite result in morphologically low lying areas that determine the course of major drainage lines such as the Wonderfonteinspruit (to the west of the CR), the Klip River (in the south) and the Elsburg Spruit and Blesbokspruit in the SE (Fig. 6.3).

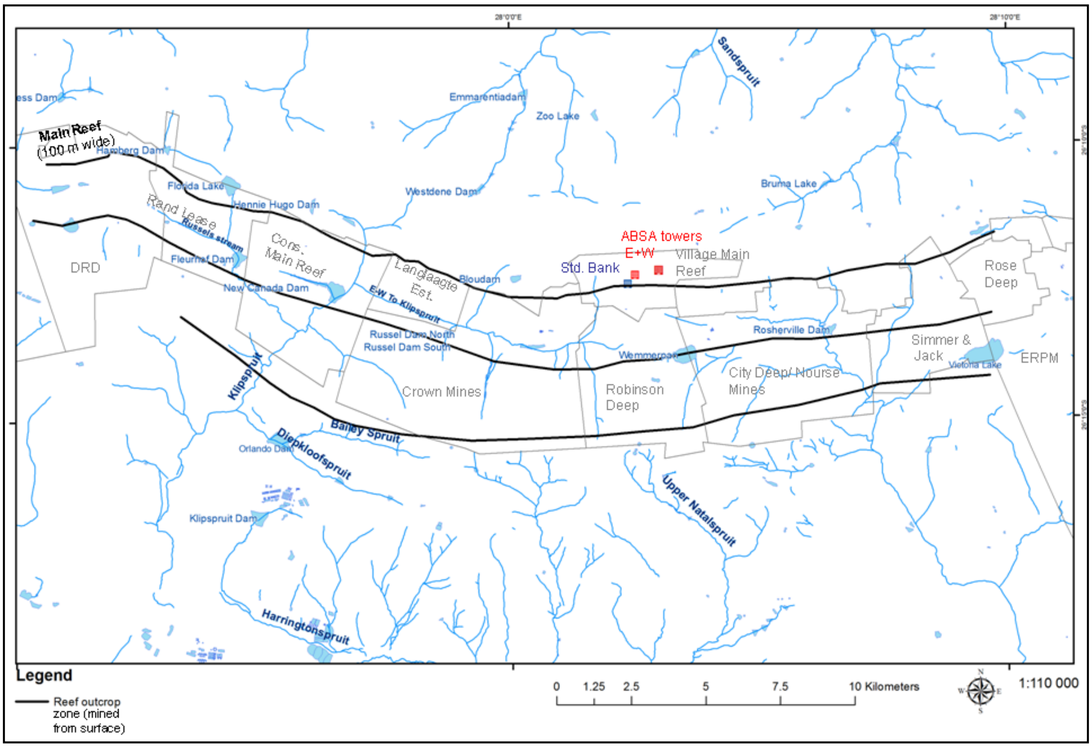

Fig. 6.3: Hydrography of the study area

The natural relief of the Witwatersrand (Afrikaans for 'Ridge of white waters') is illustrated by the network of streams and rivers draining the area. This drainage, in turn, largely reflects the underlying geology and the varying resistance the different geological formations offer against erosion. It is of importance to the project that most of parallel running elongated ridges such as the Linksfield and Kensington ridges are formed by weathering resistant non-Au-bearing rocks of the West Rand Group (Orange Groove and Hospital Hill respectively) while most of the mined gold reefs of the Central Rand Group (i.e. Main Reef-, Bird- and Kimberley reefs of the Johannesburg and Turffontein formations) outcrops in low-lying areas.

Therefore, the mined outcrop zone that marks the surface entrance to the mine void is generally found in low-lying areas, often located below urbanised slopes to the north. One exception is the Elsburg Reef, a coarse weathering resistant conglomerate, which as part of the Turffontein Group forms the South Crest ridge. Owing to the often low Au-contents in the CR this reef however was only mined locally, while in the WR it was mined for its high uranium grades (Fig. 6.4).

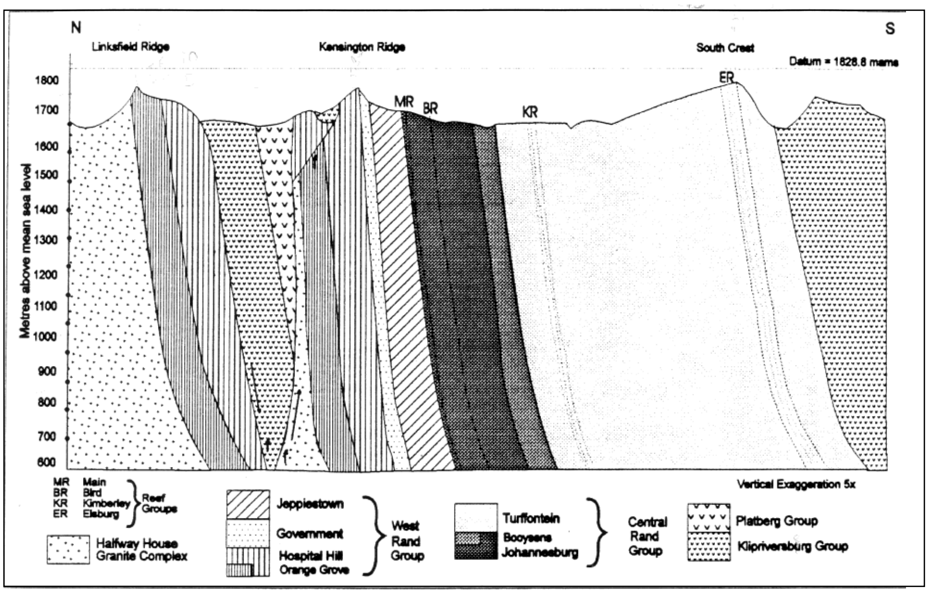

Fig. 6.4: N-S geological cross section through central Johannesburg (adopted from Scott 1995)

6.2.2.Geohydrology

Reports of the early mining period indicate that many pans and swamps have occurred in the area and that miners were able to source water from hand dug wells implying a generally shallow water table (Scott, 1995). Almost exactly 100 years later, when the tented digger camp developed into one of the largest urban metropolitan areas in Africa, the average water table below Johannesburg dropped to 15m below surface. This was determined by 37 x boreholes drilled during the drought period 1985/86 by the JHB municipality on municipal land such as parks, sport fields, fire stations and bus depots to overcome water restrictions. While water was struck at an average depth of 40m below surface (ranging from 9m to 108mbs), the average standing water level (also termed rest water level) in the drilled boreholes recovered to 15 m below surface (ranging from 0.5m to 40m), This recovery is typically found in fractured aquifers where hydraulic heads in higher-lying areas drive the groundwater along the fracture network intercepted by drilling to fill the newly drilled boreholes to the head levels. Only in 4 x boreholes were the rest water level lower that the water strike level.

Groundwater above the mining belt is stored in a low yielding fractured aquifer developed in the quartzites of the West Rand Group. During excavations of high rise buildings in the CBD, groundwater was found e.g. at the (old) Standard Bank Centre (300 m N of the MR outcrop) at 23m and at the Carleton Centre (500 m north of the MR outcrop) at 30m below surface resulting in pumping of 0,18 Ml/d from wells around the excavation pit for dewatering during the first 6 month of construction (Brink, 1979). At some other high-rise buildings in the CBD, permanent dewatering is necessary such as the Sanlam Centre (Life Towers) and 11 Diagonal Street (both located 400 m N of the MR outcrop) (Scott, 1995).

Owing to the relative low position within the natural relief the mining belt is bound to receive above-average volumes of surface water runoff from the adjacent slopes. Due to the fact that these slopes are relatively steep and highly urbanised with much of the natural surface being sealed by impenetrable materials (concrete, paving, tarred roads, roofs etc.) run off volumes are particularly high. The inflow of surface runoff into the mine void is likely to be particular prominent where the mined outcrop zone run in or near the valley bottom between parallel E-W-running ridges, receiving stormwater from the urbanised slope of the northern and southern ridge. Even where the outcrop zone is not directly located in the valley bottom but somewhat higher on the slope, ingress is likely to occur via fractures connecting streams in the valley with the S-dipping mine void (Fig. 6.5).

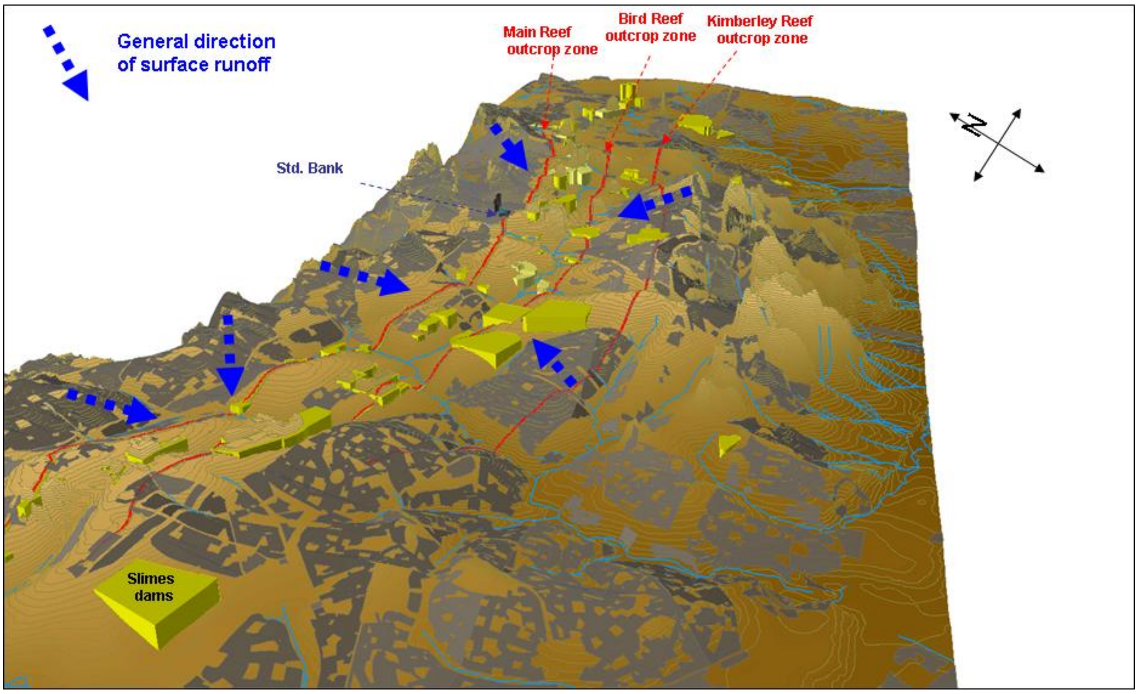

Fig. 6.5: 3D view of the valley situation

This mining belt follows the outcrop zone of the three mined reefs which initially have been mined by trenching and shallow incline shafts. This disturbance of near-surface rock and subsequent filling of mined-out surface areas as well as underground voids with unconsolidated material such as sand, rubble and mainly ash (derived from coal fired steam engine that were widely used at the time) allows for high rainwater infiltration rates into the mine void. A 1km-long mining trench dug by Anglo American along the MR outcrop at Simmer & Jack was later transformed in a municipal repository, also allowing for high infiltration rates

Given the CR topographic situation, the predominant flow direction is not N-S (following the general dip of the topographic surface) but E–W, maximising the contact length between (losing) streams /drainage lines and their associated wetlands/ floodplains and the underlying high-infiltration substrate and the mine void.

Most valleys, streams and drainage lines are geologically controlled and follow features such as faults, bedding planes, dykes and outcrops of less-erosion resistant formations. This further increases the infiltration rate for surface water. A depiction of faults and dykes occurring in the CR is shown in Fig. 6.6.

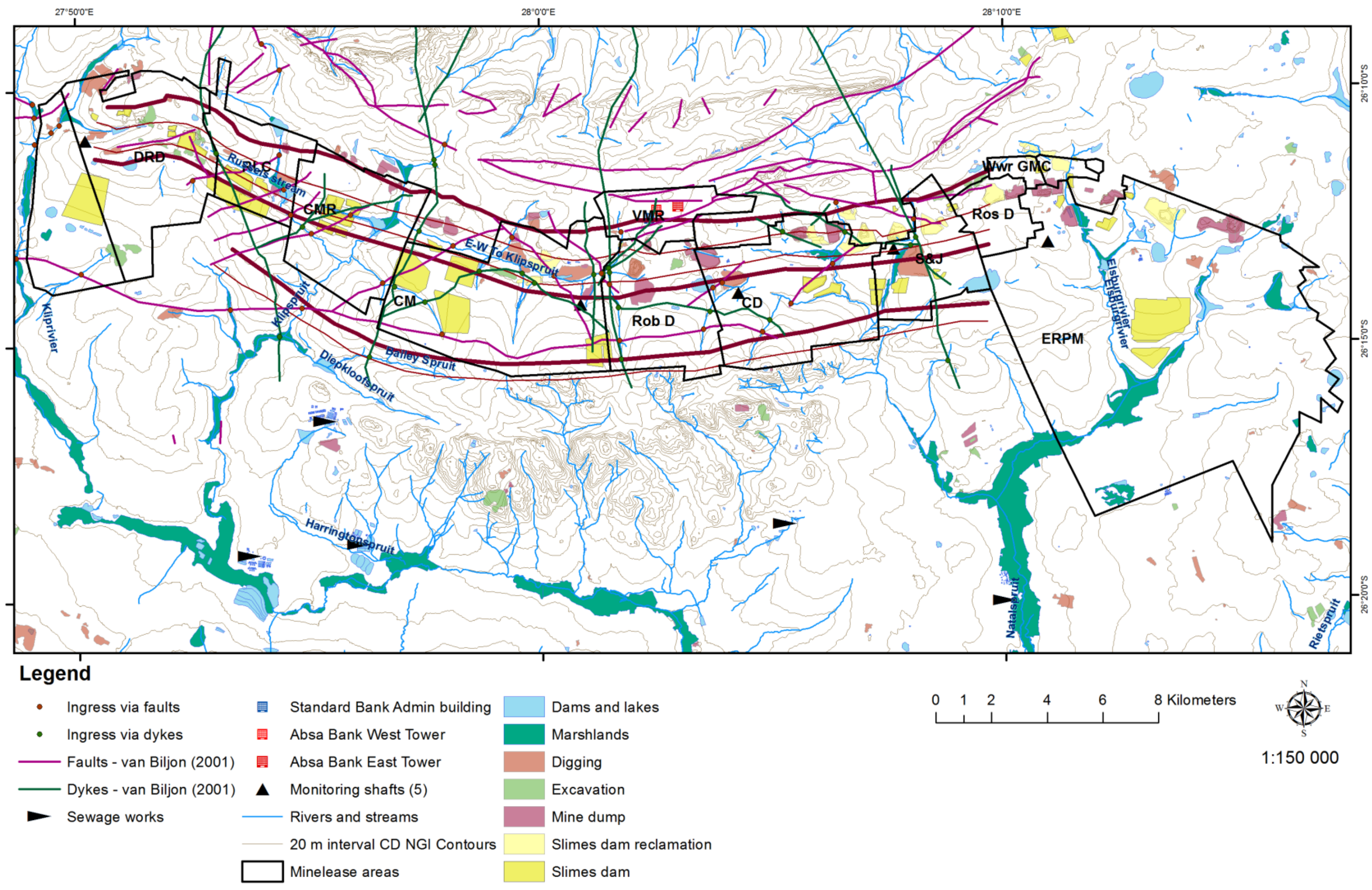

Fig. 6.6: Dykes and faults in the CR (based on van Biljon and Walker, 2001)

Apart from the relatively small E-W running streams with their limited catchment areas this also affects the generally somewhat stronger N-S flowing rivers that cross the outcrop zone (Fig. 6.7).

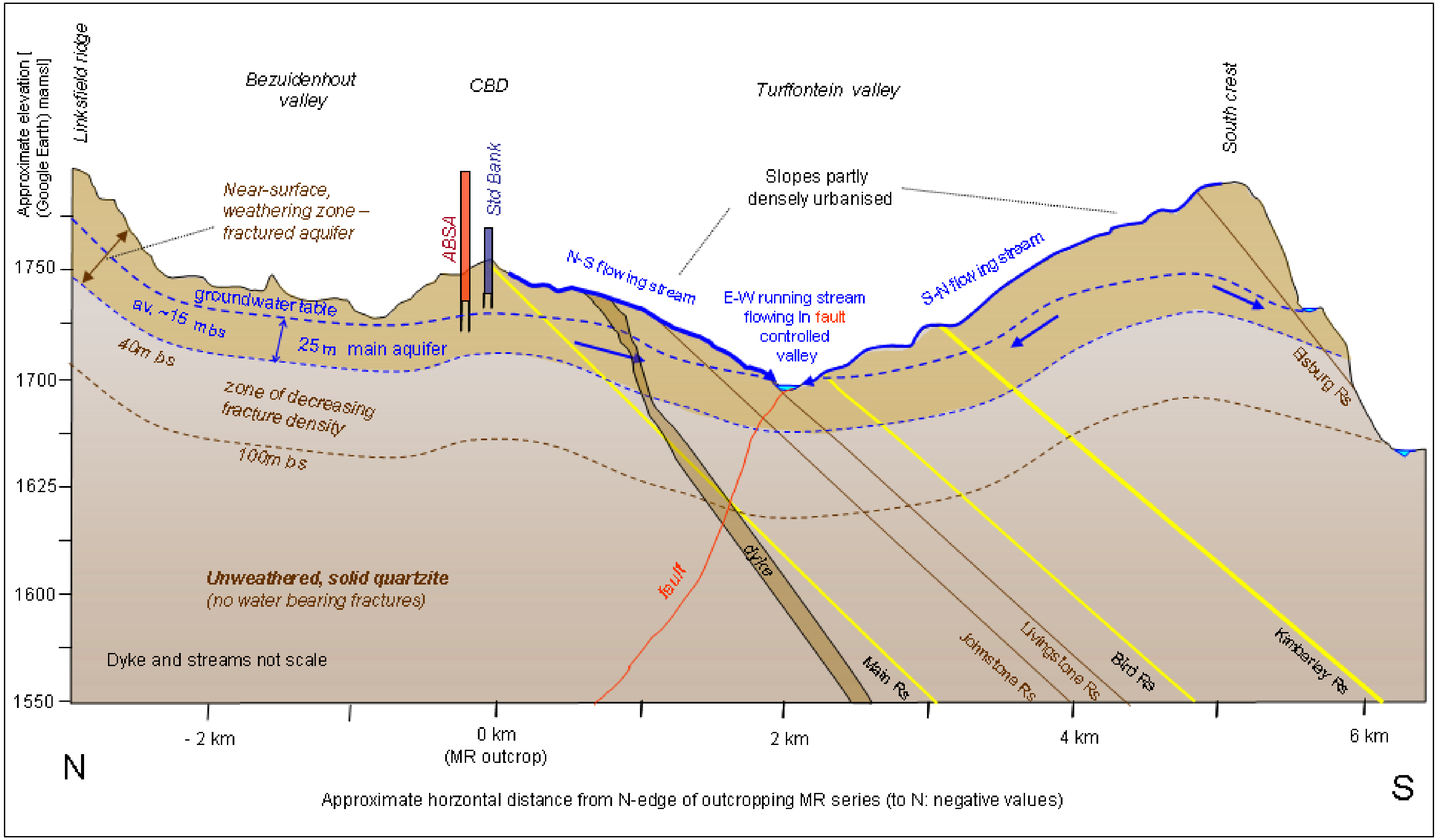

Fig. 6.7: Natural geological and geohydrological conditions in the central part of the Central Rand (based on a N-S elevation profile from Google Earth)

6.3 Mining and land use

6.3.1 Effects of mining

During early stages of mining much of the outcropping reefs was extracted by trenching (elongated open pit mining) and shallow incline shafts reaching a depth of some 30m (Scott, 1995). Below this depth however, the nature of the reefs changed from oxidised ore to significantly harder, reduced (pyrite-containing) ore posing challenges to the mine the ore as well as to extract the gold. While mercury was used to extract gold from oxidised ore via amalgamation this proofed to be not suitable for pyrite-containing ore threatening to prematurely end the gold–rush at the Witwatersrand. To overcome the challenge posed by harder and deeper ore from which Au was more difficult to extract, resources had to be pooled resulting in only larger consolidated mining companies surviving. This reduced the number of operational gold mines significantly (Scott, 1995).

Surface digging and subsequent filling of open pit areas with sand, rubble and later also municipal waste significantly increased the potential infiltration of rain and surface water entering the outcrop area (Fig. 6.8).

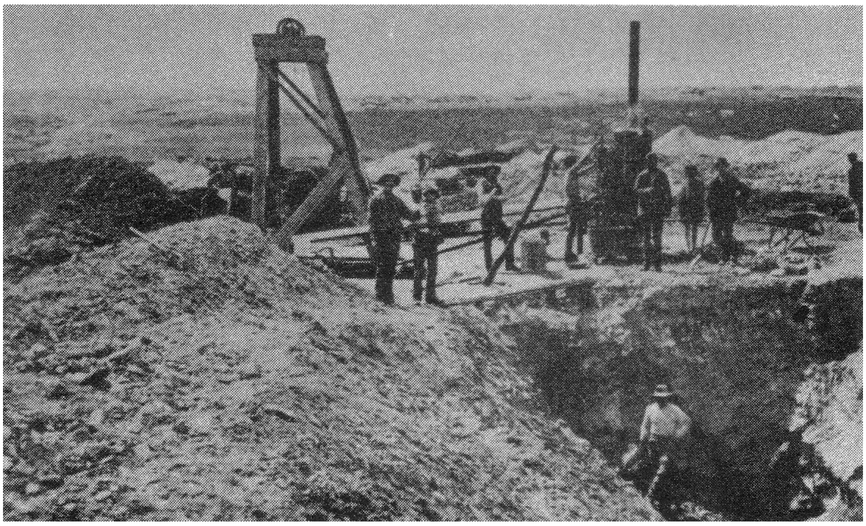

Fig. 6.8: Surface mining via trench digging (Photos: bottom – Editorial committee, 1986; top – Mendelsohn and Potgieter, 1986)

Following the outcropping reefs over the entire strike length of some 40 km the surface mining affected a zone of some 75-100 m wide and 30-50 m deep. Later additions of vertical shafts accessing the deeper lying and harder reefs some 300-520 m south of the originally mined MR outcrop further extended the mining zone. Since many shafts sunk before 1912 have not been lined they are likely to act as drains routing groundwater directly to the mine void. As mining progressed and even deeper lying reefs were mined a second row of shafts located some 1130 m from the original MR outcrop were added (Scott, 1995). Together with the supporting mine infrastructure such as hostels, metallurgical plants and tailings deposits etc. the average width of the mining belt grew to its current width of approximately 1-1.5 km (determined visually using satellite imagery in Google Earth) (Fig. 6.9).

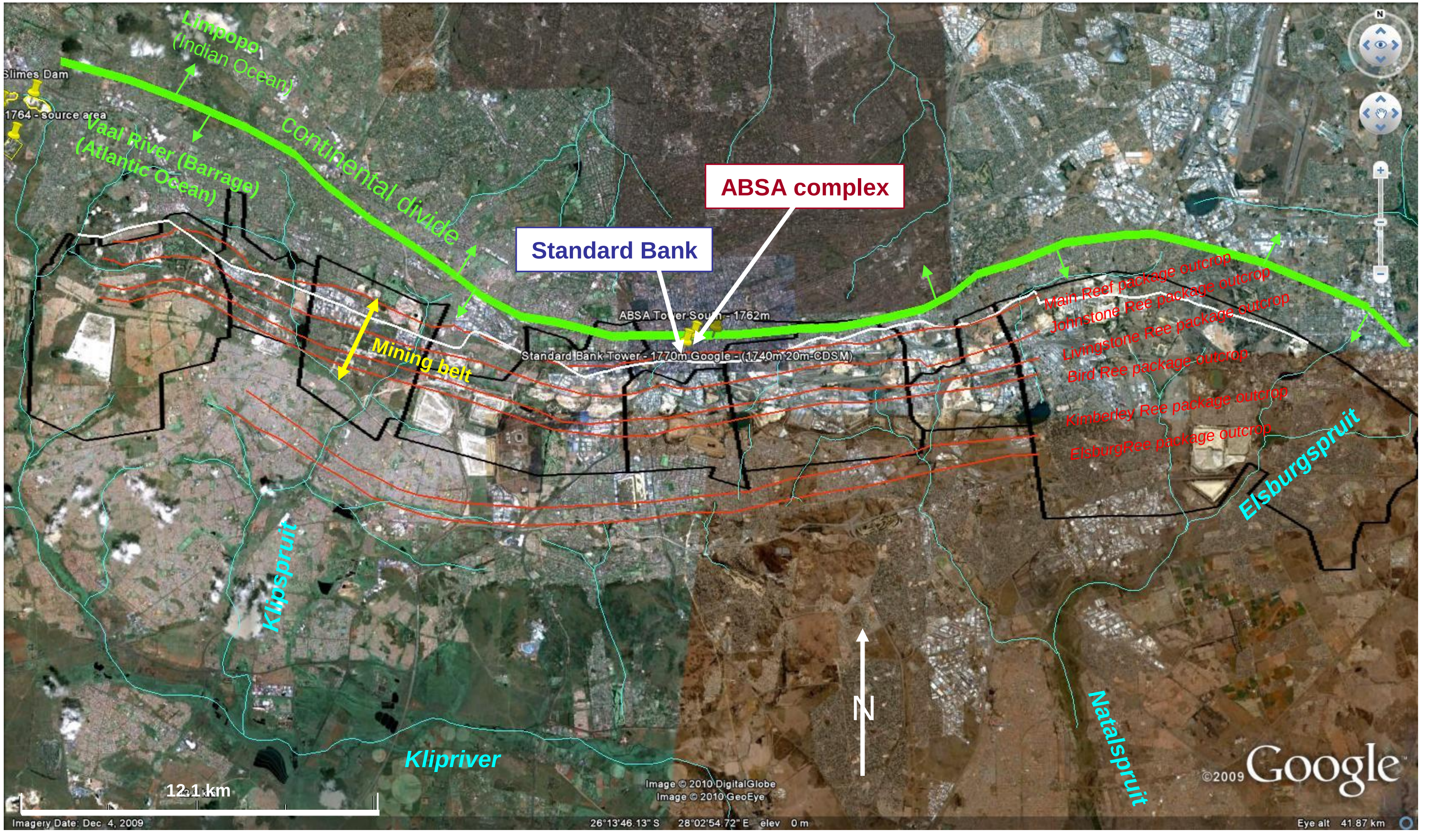

Fig. 6.9: The Central Rand with the outcropping reefs and the continental divide (map base: Google Earth satellite imagery)

Before 1930, mine workings were filled to a depth of several hundred metres with sand and rubble, but mainly with ash derived from the many steam engines used in stamp mills, hoists and locomotives (Scott, 1995). While this reduced the void volume that can be filled with water, it may also be significant in terms of the quality of the rising mine water. It should be explored to what degree the ash may buffer some of the acidity of the rising mine water.

Since water in the semi-arid interior soon became scarce as the progressing mining and milling required ever increasing volumes, many mines claimed ownership of the small local streams and altered the hydrological system to suit mining. Amongst others this included the construction of often poorly designed dams to store water for the dry winter season when many streams ceased flowing. Flooding of mine working due to the frequent failure of dams was a major hazard in the early days before water from elsewhere was brought into the area. Today many of these old (unlined) dams are still in existence recharging the underlying mine void with seepage.

Later all water needed by the mines was imported into the area sourced from the Zuurbekom dolomitic compartment as well as the Vaal River system (Vaal Dam and Vaal Barrage). Owing to pollution (largely derived from the Witwatersrand GMs), water abstraction from the Vaal Barrage stopped several years ago. Recently pumping of water from the Zuurbekom compartment also stopped owing to quality concerns.

As deep level mining progressed to depths of over 3400 m below surface and tonnages of milled ore increased, the leached and milled ore has been deposited in the vicinity of the shafts and metallurgical plants as slimes dams, further changing the hydrological properties of the areas.

The distribution of slimes dams in the Central Rand is shown in Fig, 6.10.

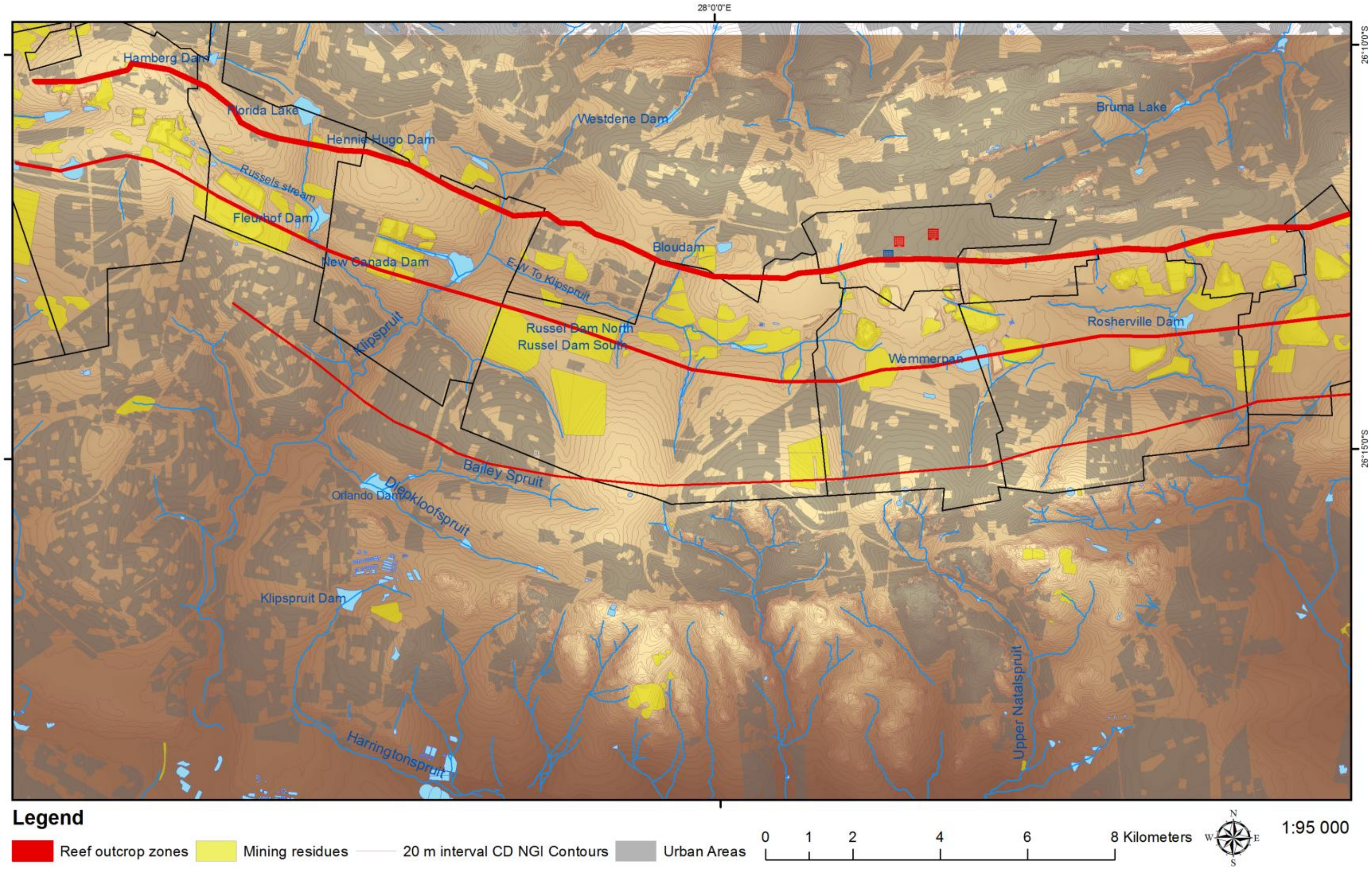

Fig. 6.10: Slimes dams in the Central Rand

Apart from creating an artificial relief that changes the run-off characteristics of the original surface topography, many slimes dams have been placed on top of natural drainage lines impacting directly on drainage patterns, and later also on the water quality of affected streams. Furthermore, owing to the fine grained nature of the mining residues, water filling the pore space between the solids forms a water table some meters below the surface of the sand dumps and slimes dams, which is well above the natural water table level (Fig. 6.11).

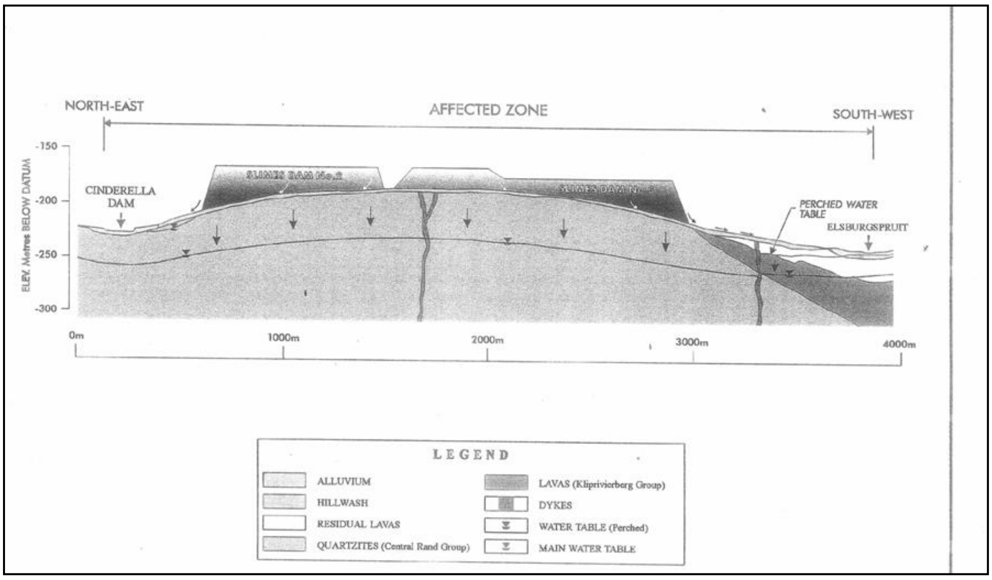

Fig 6.11: Slimes dam cross section depicting the impact of seepage on the elevation of the underlying groundwater table (adopted from the EMPR of ERPM, 2001)

This, in turn, drives a continuous flow of seepage that may recharge the underlying mine void via the highly penetrable substrate on which they are based. In some instance slimes dams in the CR have failed owing to the collapse of shallow underground mine workings (Scott, 1995).

Another possible source of water entering the mine void is well-irrigated golf courses commonly found on each of the mine properties, often displaying a number of permanent water features such as little streams and lakes. With irrigation water commonly applied in the order of ca. 1700 mm/a, this effectively triples the rainwater volume these areas receive per annum.

The fact that a large percentage of the outcrop area is threatened by ground subsidence and covered with mining residues prevented large-scale housing or office parks to develop, even in areas close to the CBD where property prices are at a premium. The resulting gap in the densely populated JHB area is frequently filled by informal settlements often encroaching right up and in some places even onto slimes dams.

Compacted over the decades to stable structures some of the slimes dams have later been incorporated as supporting structures for the M1 Highway and other major roads. In the event of a possible rise in the local groundwater table following the completion of the mine void flooding this may pose a geotechnical hazard in such places where underlying material may become water saturated and liquefy (Fig. 6.12).

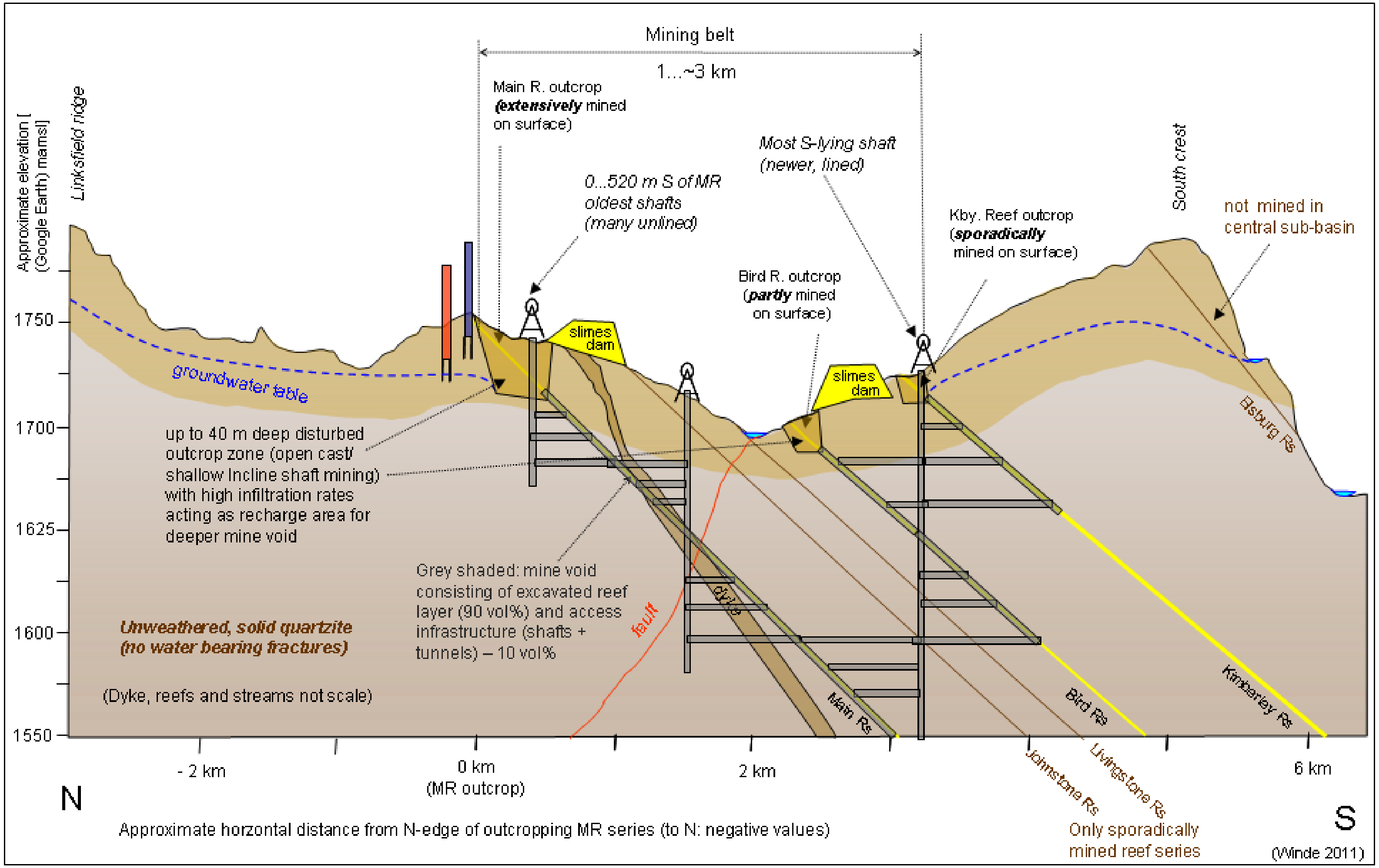

Fig. 6.12: Schematic N-S cross section through the central sub-basin at the CBD depicting impacts associated with surface and deep-level gold mining

6.3.2 Effects of surface tailings reclamation

Since the early 1980s, when the first tailing reclamation plant was commissioned (RM 3), many of these tailings have been reclaimed by various operators such as AngloGold's East Rand Gold and Uranium Operation (ERGO), DRD's Crown Gold Recovery (CGR) and several other smaller operators. This is usually conducted by hydraulic mining where high-pressure water canons are used to disperse the deposited slimes and bring them back into suspension, introducing large volumes of additional water into the high infiltration area. Furthermore, some operators like Village Main Reef had permission from the DWA issued in 1982 (although dumping of slimes was practiced before) to dump the reworked tailings as a slurry into the underground mine void reducing void volume while injecting additional water. The maximum deposition was stipulated as 120000t/month of which 50% had to be water and 50% solids (tailings) by weight, not exceeding a total deposition of 1 million m^3/a. At a density of tailings of 2.65 t/m^3 this would result in 60 Ml/month of water (=2 Ml/d) and 22.6 Ml/month of tailings (= 0.75 Ml/d) totalling an injected volume of ca. 2.75 Ml/d. According to EcoSat ERM (2000) a total average volume of 3.5 Ml/d additional water was assumed by ERPM to be pumped from its SW Vertical shaft. The mine was required to compensate ERPM for the additional volume of water that arrived at the SW Vertical shaft because of its underground tailings dumping.

Records of dumped tonnages for October 1991 (140923t) and 1992 (136503t) given in Scott (1995:121) indicate that these limits may have been regularly exceeded (in this case by 17 % and 14% respectively). Furthermore, the average percentage of water in both cases is also higher than stipulated (56% and 62% respectively) resulting in an even higher total volume injected into the mine void. Assuming an average of 141.000 t of dumped slurry with 60:40 water-solid ratio the mine pumped some 106 Ml/month (=3.5 Ml/d) into the mine void. Practised over at least 13 a (1982 to 1995) this resulted in a minimum of 10.16 million t of tailings injected into the mine void reducing the void volume by 3.83 million m^3. This exceeds the originally mined volume at Village Main Reef (1892-1920: 1.75 Mt = 0.7 million m^3) by over 5 times (5.46 x) suggesting that much of the dumped tailings also filled void space in adjacent mines such as Robinson Deep and City Deep. The amount of tailings dumped underground by VMR alone equals 0.9 % of the total tonnage of ore milled in the CR (1080.4 million t in 1984, Scott 1995). It is assumed that the tailings from surface operations are dumped in the upper most part of the void. Owing to the reduction of the mine void volume caused by gradual stope closure (discussed in a separate section below) the impact of slimes dumping on void volume reduction is more significant than suggested by the low percentage in terms if milled ore.

According to Scott (1995) signs of underground slimes disposal were also found at other operators such as the Badenhorst Mine which did not have the required permits. The additional injection of tailings and water in a 1:1 mixture (termed slurry) may explain some unexpected rises in water tables observed in the area (Scott 1995).

Other operators transported the slurry via long pipelines to reactivated tailings dams at Nasrec or the ERGO Mega-dam in the East Rand. Based on experience in the FWR such pipelines are prone to leakages, and other failures resulting in more water being introduced to the outcrop area. Regarding possible future pipeline failures it is noteworthy that DRDs CRG operations currently plans the construction of a 50 km-long multi million R pipeline from its met. plant at Crown Mines to the Ergo SD in the East Rand.

The areas cleared from tailings, especially those in close proximity of the CBD, were acc. to Scott (1995) in the late 1990 earmarked by Rand Mines Ltd. for development of middle income housing as well as office parks and industries. How far this has happened should be established as the soil of SD footprints remain contaminated to a considerable degree amongst other with U. With radon as one of the products of radioactive decay, chances are that the radioactive gas may accumulate in residential housing erected on footprints of mining deposits. Being a known cause of lung cancer radon could pose a significant health risk.

6.3.3 Effects of urbanisation

In search of employment by the mines and related sectors many people flocked to the area rendering Johannesburg the most densely populated and fastest growing urban area in South Africa.

The exponential growth in population and associated extension of built up area lead to a general increase of water volumes imported into the area (and discharged via sewage works) as well as changes of the natural local water balance due to effects associated with urban developments. The latter includes a drastic change of the run-off coefficient of the area towards reduced infiltration (i.e. diminishing groundwater natural groundwater recharge) and increased runoff owing to the growing coverage of natural soil with impervious layers such as concrete, tarred roads, roofs etc. (Fig. 6.13)

Commissioner Street, Johannesburg, 1887.

Standard Bank, Johannesburg, 1886

Fig. 6.13: Uncovered soil surface in JHB allowing for the infiltration of rainwater before urbanisation (Photo top: Mendelsohn and Potgieter, 1986; bottom: Brodie, 2008)

The decreasing groundwater recharge may be illustrated by the fact that the water table in the fractured aquifer underlying much of Johannesburg dropped significantly since the discovery of Au kick-started urbanisation. Reportedly, early miners living in tents on the flat Highveld before there were any signs of urbanisation were still able to source their water from hand-dug wells implying a shallow groundwater table close to surface (Scott, 1995). This water table dropped to an average of some 15 m below surface as determined by boreholes drilled some 100 years later throughout central Johannesburg (Scott, 1995) as mentioned earlier.

Simultaneously the number of residents increased exponentially from several hundreds in 1886 to currently 3.5 million requiring a corresponding rise of imported water volumes. As much of the consumed water is discharged again via sewage systems the majority of imported water finally flows into nearby rivers of the Central Rand as sewage- or industrial effluent. This transformed many of the formerly seasonal rivers into perennial water bodies. Van Biljon & Walker (2006) as well as other suggested that the increasing discharge of sewage effluents into formerly seasonal streams crossing the outcrop areas in the CR is, amongst others, responsible for increased ingress volumes. This, however, is unlikely to be the actual reason for the increased stream flow as the all sewage works discharging effluents are located either north of the continental divide or well below the areas where the outcropping reefs have been mined (Fig. 6.14).

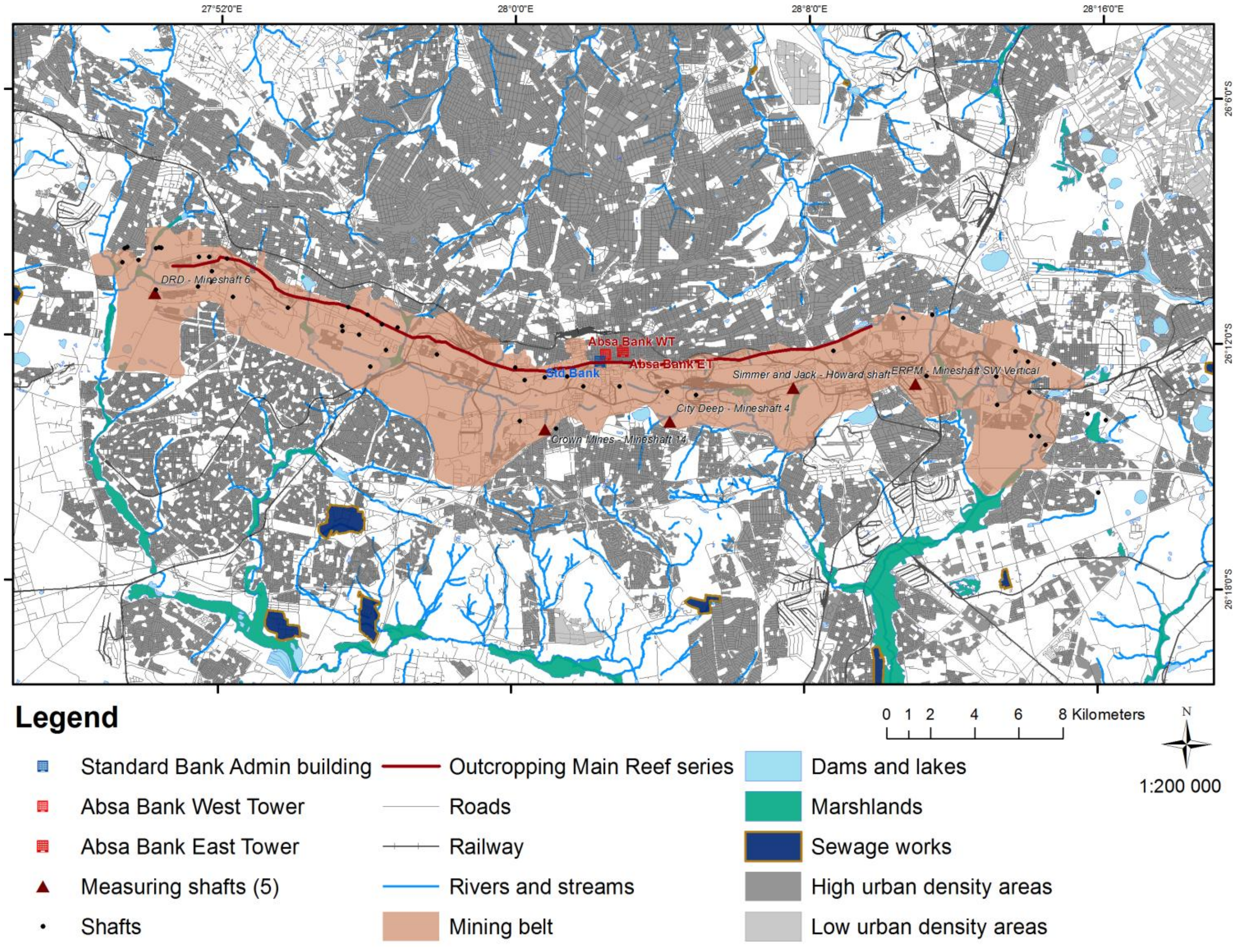

Fig. 6.14: Map of the study area showing the location of streams, urbanised areas as well as of sewage works

Thus, the increased volumes of discharged sewage effluent (apart from untreated raw sewage possibly received from informal settlements) can not account for the increased ingress volumes. Investigating the upper Natalspruit near Alberton Winde and Sandham (2005) found that much of the run-off consisted of seepage from tailings dams covering large proportions of the relatively small headwater catchment of the stream. Owing to elevated piezometric porewater surfaces forming within slimes dams and sand dumps, seepage flows continuously into adjacent rivers providing a baseflow also during the dry season. As almost all streams originating south of the continental divide are affected by seepage from mining residues, this is likely to be the actual cause of the change from seasonal to perennial flow regimes and thus the increased ingress. Of particular concern is that seepage from tailings and sand dumps is commonly of very poor quality being effectively acid mine drainage (AMD). This, in turn, means that much of the water entering the mine void is already highly polluted which will make it difficult to distinguish at a later stage between decanting mine water after the void is filled and polluted surface water that no longer flows into the mine void but directly into rivers.

Since most of the Johannesburg reticulation systems is gradient-flow based, the associated drainage infrastructure consisting of underground canals, pipelines and sewage works is typically concentrated in topographically low lying areas such as river valleys and drainage lines. As the MR series is a negatively weathering rock type, much of the mined outcrop zone runs in low lying areas and E-W running valleys rendering it potentially susceptible for water lost from reticulation systems. For the pressurised water reticulation in the Johannesburg metropolitan area an average leakage loss of 35-40% of the total input volume is reported. While such water loss somewhat counteracts the reduced natural recharge resulting from urbanisation and the associated sealing of the natural soil surface, it was obviously not enough to retain the original water table level below Johannesburg. Since many of the pipelines running across or parallel to the mining belt are embedded in unconsolidated fill material and exposed to increased levels of shear stress resulting from heavy mine truck traffic, collapse of underground mine workings, mining induced seismicity etc., it stands to reason that water losses due to stress-induced leakages and pipe failures may be particularly pronounced in the outcrop area.

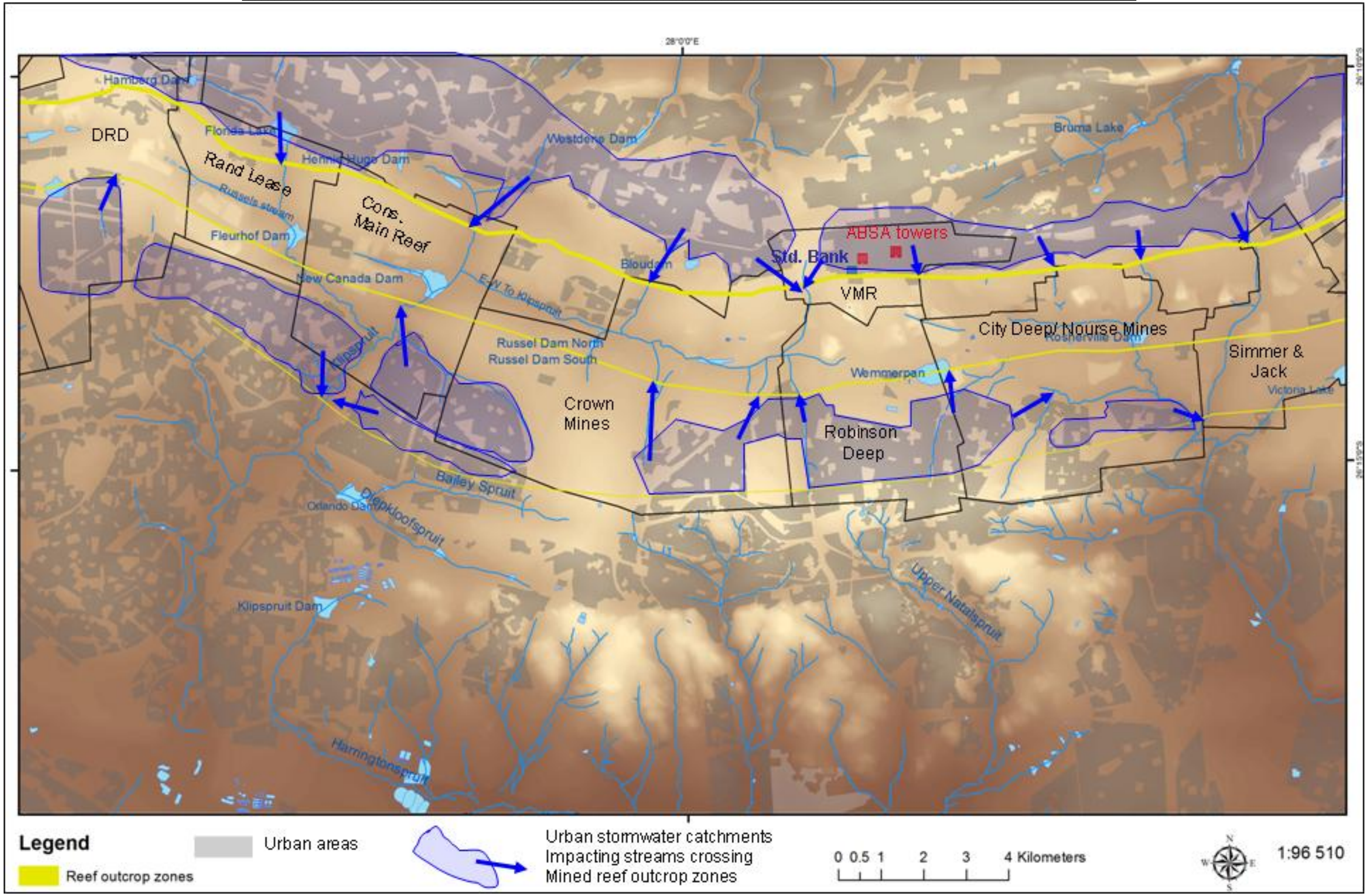
DRD
Rand Lease
Cons. Main Reef
Hamberg Dam
Florida Lake
Hennie Hugo Dam
Westdene Dam
Bruma Lake
Russels stream
Fleurhof Dam
New Canada Dam
E-W To klipspruit
Bloudam
Std. Bank
ABSA towers
VMR
City Deep/ Nourse Mines
Rosherville Dam
Simmer & Jack
Victoria Lake
Russel Dam North
Russel Dam South
Klipspruit
Crown Mines
Wemmerpan
Robinson Deep
Bailey Spruit
Orlando Dam
Diepkloofspruit
Upper Natalspruit
Klipspruit Dam
Harringtonspruit
Legend
Urban areas
Reef outcrop zones
Urban stormwater catchments
Impacting streams crossing
Mined reef outcrop zones
0 0.5 1 2 3 4 Kilometers
1:96 510

Fig. 6.15: Sloped urbanised areas generating stormwater run off that enters the mining belt and contributes to ingress into the mine void

Based on the measured surface area of the urban settlements draining into the mining belt (145 km²), the long-term mean annual precipitation over the area (730 mm/a) and the run off coefficient into account a first order approximation of the total stormwater run-off volume generated annually by the high-density urban catchments associated with the mining belt was calculated. The run coefficient was estimated as product of the average proportion of impervious surfaces in high density settlements (conservative estimate of 50% = factor of 0.5) as well as the average slope (medium: 1 m height difference over 50 m and less resulting in 70 % of rain running off rather than infiltrate: factor 0.75) suggesting that 35% of the rain would run-off (run off coefficient: 0.35). Based on these first order approximates the total volume of stormwater runoff from high-density settlements draining towards the mining belt is approximately 51602 Ml/a (101 Ml/d). Assuming that some 80% of the run-off is captured by underground stormwater drainage systems that discharge directly into nearby streams this accounts for 81 Ml/d averaged over the whole year. Since some 80% of MAP in the Highveld fall between October and March (6 x months) the average daily stormwater input into streams of the CR during the summer season is ca. 130 Ml/d and some 40 Ml/d during winter. Much of this water will arrive in the form of highly intense flash floods frequently exceeding the flow capacity of the small receiving streams causing temporary inundations of adjacent areas, which in turn may further increase the associated ingress. The rising stream water levels are likely to accelerate the rate of stream water lost to the underground through permeable stream beds. Winde (2005) found for a mining-impacted stream in SE Germany that relatively small stream level changes result in disproportionally increased infiltration of stream water into the Hyporheic zone below the stream bed. Since little or no evaporation can take place during the rain event (the air is normally fully water saturated i.e. displaying a relative humidity of 100%) and is very limited once the run-off entered underground canal systems where it is largely protected from evaporation associated losses (which under filed conditions account for well over 90% in much of South Africa) rare only of marginal, if any, importance in urban catchments.

The reminder of the stormwater run-off that is not captured by underground drainage channels is termed ‘diffuse run off’ and accounts for an estimated 20% of the total stormwater volume. Flowing over pervious and impervious surfaces which are not protected from evaporation the diffuse run-off component may be somewhat reduced by evaporation where longer flow distances are to be covered after the rain stopped. Averaged over the whole year (including the comparably dry winter season this type of stormwater run off accounts for an estimated 20 Ml/d (i.e. 32 Ml/d during summer and 8 Ml/d during winter). As much of this run off will flow across the lower lying disturbed reef outcrop zone it is likely to significantly contribute to the ingress into the mine void adding a strong seasonal component.

6.4 Summary

Since many of the mined reefs are comparably easily weathered, their outcrop is in low lying parts of the natural relief, which is an important aspect for the project as it reduces the potential flooding risk for higher lying urban areas that developed later. As the surface entrance to the underground mine void, the outcrop areas of the mined reefs now defines the level up to which the water in the mine void can possibly rise.

Mining increased infiltration of surface water due to the disturbance of natural rock and soil to a depth of approximately 50 m. Later it placed large deposits of mining residues such as sand dumps and tailings dams on this soil (approximately 40% of total area). Owing to the formation of an elevated piezometric surface formed by porewater within the mining deposits, especially sand dumps and tailings act as a continuous source of polluted seepage entering the mine void via the high-infiltration soil in the mined outcrop zone. Due to historical needs to use local water resources for mining and ore treatment, many streams in and upstream of the outcrop zone have been altered in their course and often poorly constructed dams been added as water reservoirs. These dams also act as continuous sources of water recharging the underlying mine void. Later, much of the water needed for underground mining and milling of ore has been imported into the area via pipelines from Rand Water. Leakageg of these pipes may also have contributed to ingress. As a newly established mine, (CRG) is operating in the CR and their input of service water (and perhaps also tailings via backfilling or dumping) needs to be quantified.

Since the early 1980s many slimes dams have attracted surface operations to extract the contained residual Au and U. Since the hydraulic reclamation of tailings deposits uses high-water pressure canons, this widespread practice in the CR introduced large volumes of additional water into the high infiltration zone. Furthermore, many of the surface operators dumped the leached tailings as a solid – water mixture directly into the underground mine void introducing even more water while reducing simultaneously the storage capacity of the void. As surface operations are still ongoing their contribution to the direct recharge of the mine void aquifer should be quantified.

Because of the position of the mined outcrop belt and its associated southern infiltration zones in the topographic lows of the CR, it stands to reason that gradient flow-based water and sewage pipes are concentrated in this zone. Since much of the soil in this zone is disturbed and unconsolidated, effects of heavy mine truck traffic as well as the collapse of shallow underground mine workings causing subsidence may have resulted in above-average leakage and water losses from affected pipelines and reticulation systems. With an average water loss of some 40 % of the input volume from pressurised water reticulation systems in the Johannesburg area, these leakages may add a significant volume of water to the ingress into the mine void. Moreover, since the MR outcrop zone is situated over large parts of its total strike length in valleys, stormwater systems from the urbanised northern and southern slopes are likely to drain directly into streams running across or parallel with the MR outcrop zone. Since urban areas generally generate large storm water volumes (somewhat less so from informal settlements where surface coverage with impenetrable materials is reduced) and are situated on sloped terrain which further increases run-off, the influx of stormwater run off into the MR outcrop zone is likely to be significant.

7 Identification and location of key buildings

All buildings under consideration are located in the CBD of JHB comprising a total of 10 buildings (Fig. 7.1).

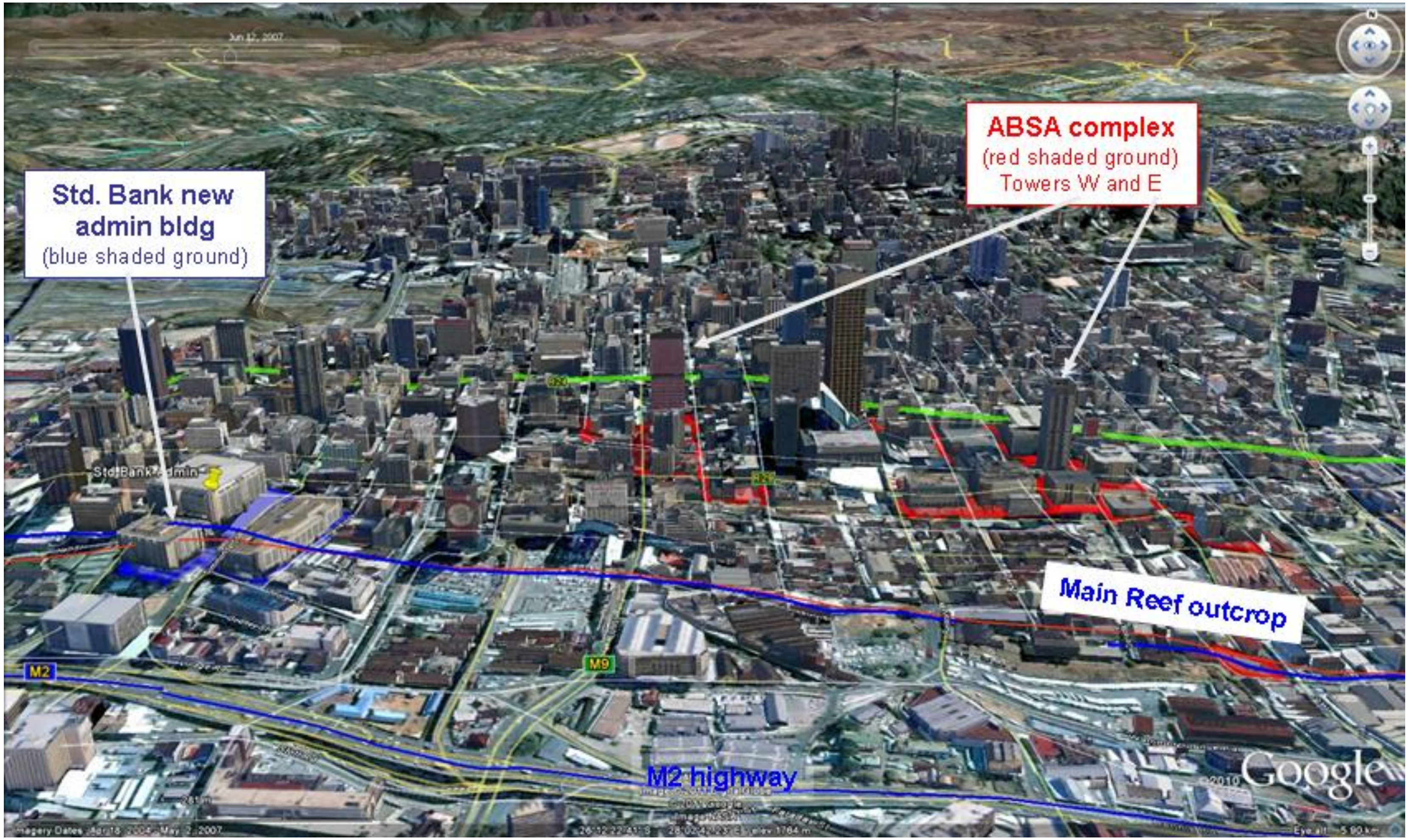

Fig. 7.1a: Location of bank buildings in the JHB CBD as shown in a 3D view of Google Earth (looking north)

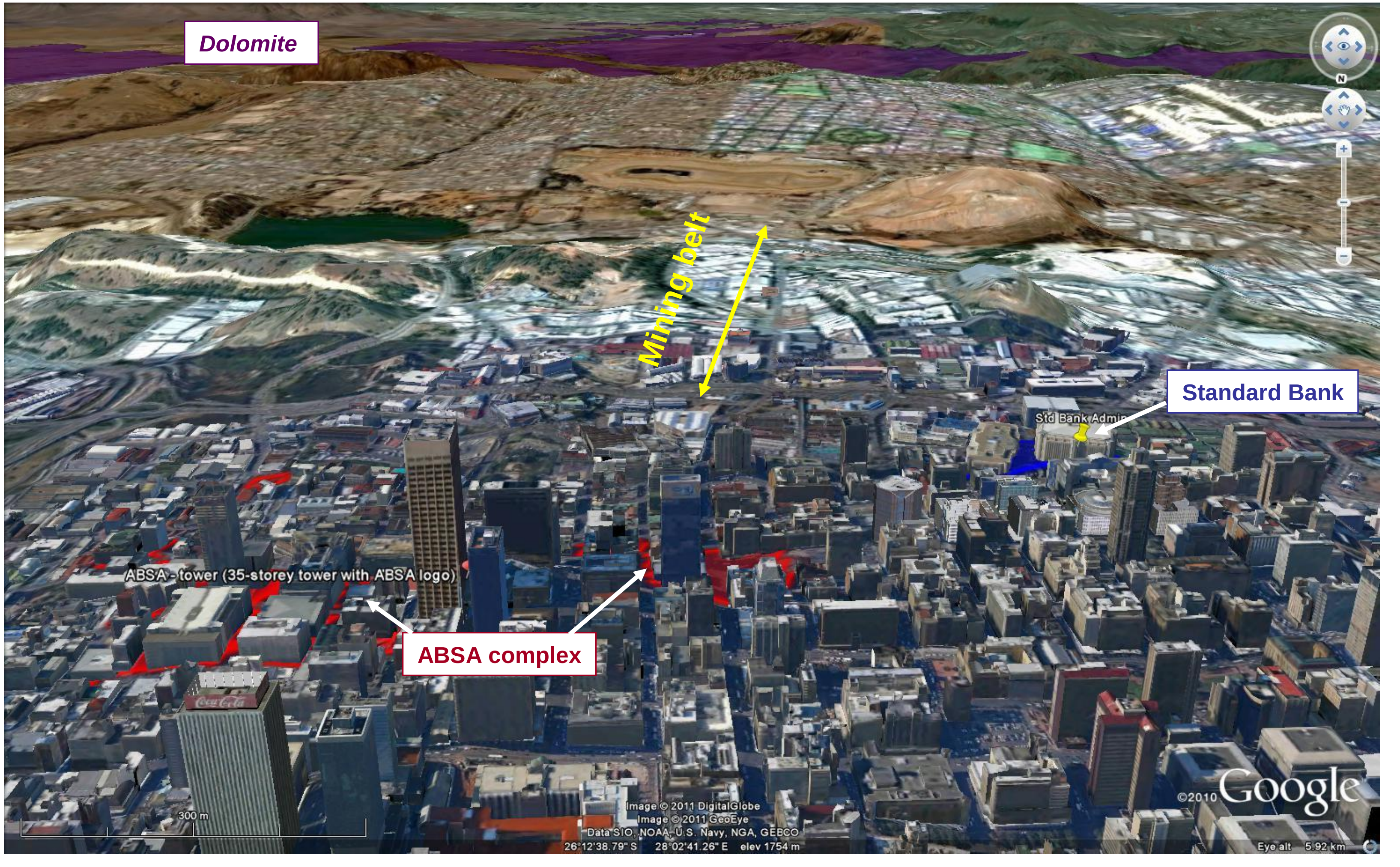

Fig. 7.1b: Location of bank buildings in the JHB CBD as shown in a 3D view of Google Earth (looking south)

The ABSA complex (internally termed ‘campus’) within the CBD comprises of high rise buildings such as the two ABSA towers East and West assumed to require comparatively deep piles anchoring the buildings in bed rock.

While Standard Bank has identified the new administration building as a structure of concern ABSA indicated a number of different buildings associated with the ABSA tower complex. As most of these buildings are close to each other at a comparable distance to the MR outcrop only the highest 35-storey tower was used for the risk assessment displaying the deepest pile structures. The ABSA property at the corner of School Street and Mooi Street, which – like the standard Bank building – is located directly on the projected outcrop of the Main Reef exposing it to a potentially increased flooding risk, was not included in the assessment as Google Earth satellite imagery dated 7 Feb. 2010 showed only a single relatively small building on that property.

Tab. 7.1: Location and depth below surface for the identified key buildings of Std. Bank and ABSA

Building	No of floors above surface	Coordinates	Street address
Standard Bank new admin building	10*	26°12'33.57"S 28° 2'21.78"E	Cnr Simmonds-Frederick Streets
ABSA tower W	25	26°12'22.47"S 28° 2'37.15"E	Cnr Fox-von Brandis Streets
ABSA tower E (with logo on top)	39	26°12'22.82"S 28° 2'59.63"E	Cnr Troye– Main Streets

* Information provided telephonically by security in the admin building of Standard Bank (Tel. 011/ 299 4701; 0860123000).

8 Determination of elevations

8.1. Sources and accuracy of elevation data

It is commonly assumed that Lidar data, owing to their high vertical resolution (up to 25 cm in this case) is the most accurate data set, followed by the digital 5 m-contour data from the official 1:50000 topographic map series of the CD-NGI and in last position the Google Earth elevations at 1m-resolution (or 1 feet) which are derived from a global SRTM data set.

In this regard it needs to be pointed out, however, that the digital 5m contour data is the product of a computerised interpolation process using trigonometrically surveyed beacons and as such prone to technique-related uncertainties.

8.1.1 5-m-contours of the CDNGI (1:50,000 topographic map)

While ground-based trigonometric surveys produce the most reliable elevation data this method is time and cost intensive and only applied to very few points such as trigonometric beacons. All collar elevations of monitoring shafts used in this project have been trigonometrically surveyed. Fig. 8.1 shows the 5-m-contour intervals provided by the CDNGI for the study area.

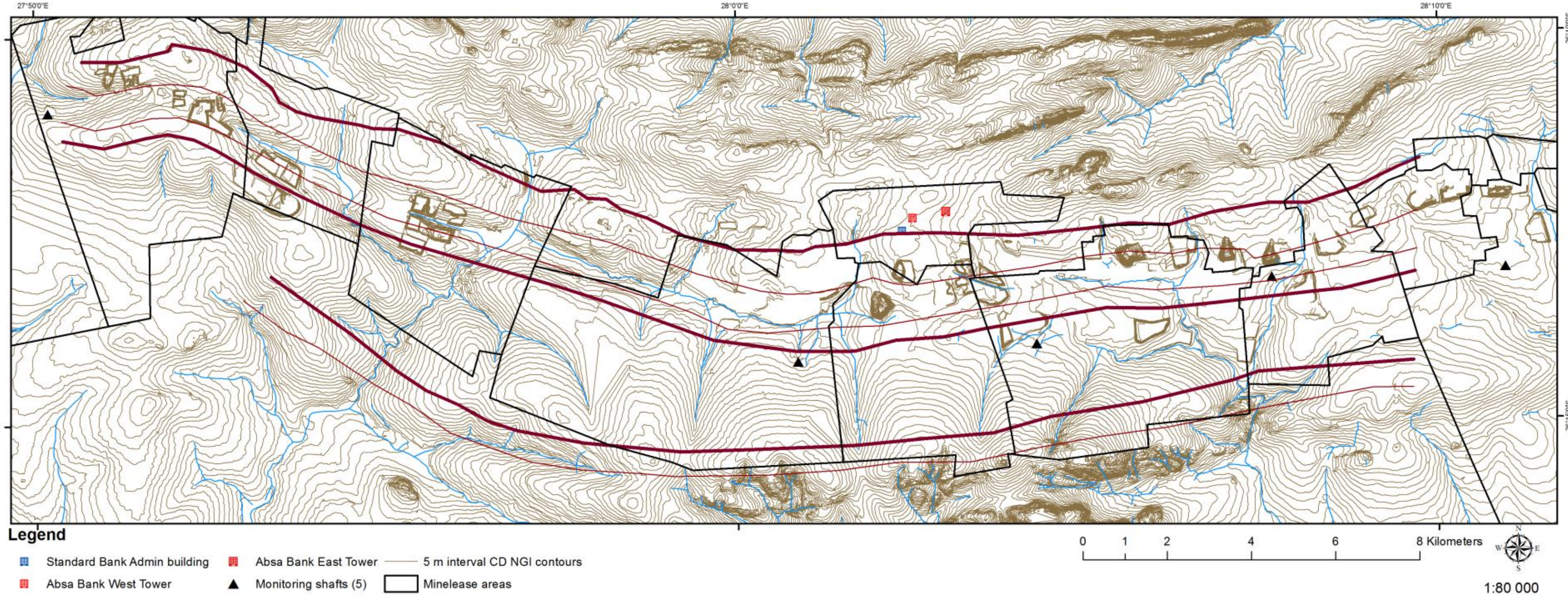

Fig. 8.1: Contours at 5 m interval provided by the Chief Directorate for National Geospatial Information (CDNGI) for use in 1:50,000 topographic map series (red lines: reef outcrops [from N-S] Main R., Johnstone R., Livingstone R., Bird R., Kimberley R., Elsburg; bold: mined reefs)

Apart from trigonometric beacons, the 1:50000 topographic maps also displays so-called ‘spot heights’ which are derived from aerial surveys using orthographic stereo imaging. The generated pairs of 1:10000 orthophotos (with 60% overlap) are subsequently digitized (since 2008 digital cameras are used providing direct digital data) and fed into a stereo digital 3D processing workstation which subsequently generates a digital elevation model (DEM) through applying stereographic and interpolation techniques (Duesimi, 2011; CDNGI 2011).

Both techniques are prone to introduce a certain error into the DEM. In order to reference the absolute altitude of the DEM, the elevations of a number of trigonometric beacons are used aiming for a vertical accuracy of 0.3 m. From the calibrated DEM the 5 m contours are derived used in 1:50,000 topographic maps (Duesimi, 2011). Spot heights, which are also derived from the DEM, are mainly added to provide additional information in (flat) areas where the 5m contours are too far apart, or to indicate the exact location of a summit of prominent landmarks such as hills and ridges. It should be noted that their accuracy is identical to the 5m contours as both are derived from the same DEM. According to the CD NGI the maximum vertical deviation of both elevation data sets is 2.5 m compared 0.3 m for ground-surveyed trigonometric beacons (Duesimi, 2011). The larger error margin of a spot height is mainly introduced by the various computerized data processing techniques. The DEM based on 5 m CDNGI contours used in this project is depicted in Fig. 8.2.

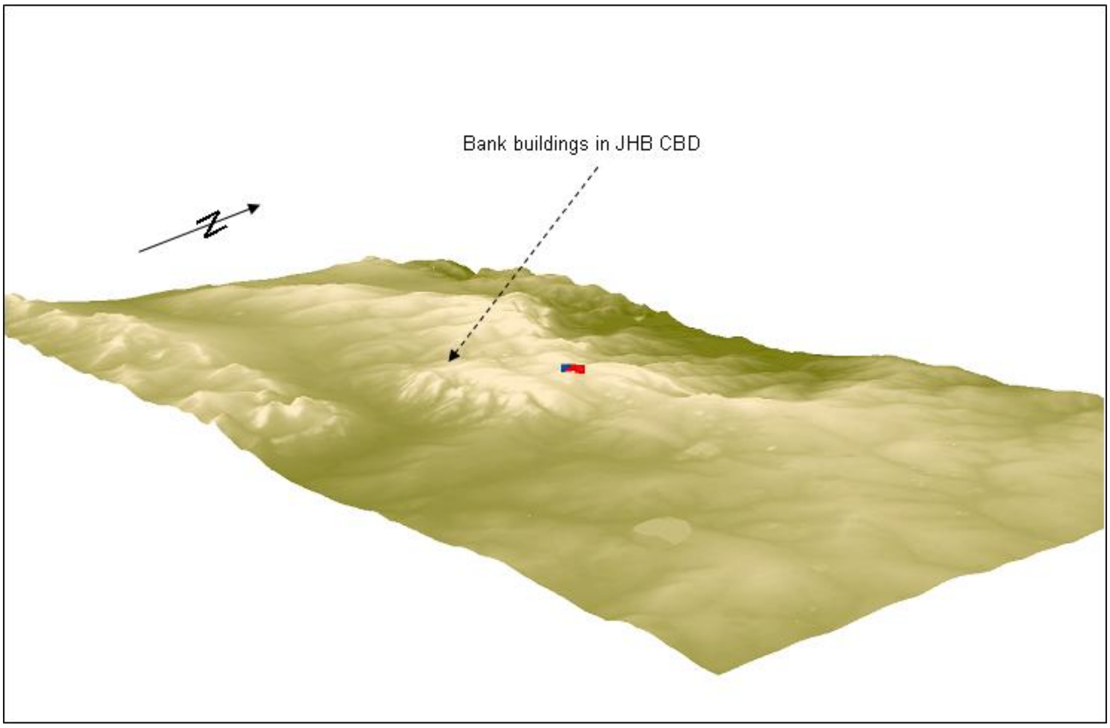

Fig. 8.2: DEM of the Central Rand based on 5 m contour intervals of the 5 m-CDNGI contours

8.1.2 Lidar data (25 cm/ 1 m vertical resolution)

Lidar is an acronym that stands for Light Detection And Ranging and is a remote sensing technique that uses light (often pulsating lasers) to measure distances e.g. from a flying object such as aircrafts to the earth surface (Wikipedia, 2011).

The data used in this study have been purchased/ obtained from municipal offices. Since the study area stretches over two municipalities (JHB and Ekhuruleni) which manage the distribution of lidar data of their respective areas independently from each other, the data set used in this report consist of two sub-sets obtained from the two different departments, both generated in 2006.

The lidar data for the western part covering the study area from Roodepoort to the Ekhuruleni district boundary in the east displays a vertical accuracy of 25 cm. The data was purchased in the form of ground elevation points from which subsequently contours were generated via GIS-based interpolation techniques (inverse distance weight – idw - type). The western part of the study area is covered by approximately 8 million points resulting in 1 point per 1.43 m². In order to remain accurate, the vertical resolution should not exceed a third of the horizontal resolution (1.43 m). For the lidar data in question, this amounts to 48 cm. Compared to this value, the 25 cm vertical resolution claimed by the data provider is approximately double the resolution it can possibly achieve.

For the eastern part covering the Ekhuruleni municipal area, the vertical resolution of the lidar data (delivered free of charge) was given at 1m. This data set was provided in the form of contours at 1m interval. For buildings falling between two contours the elevation is to be determined by interpolation. Together with the contours, so-called ‘town survey marks’ were provided (Fig. 8.3).

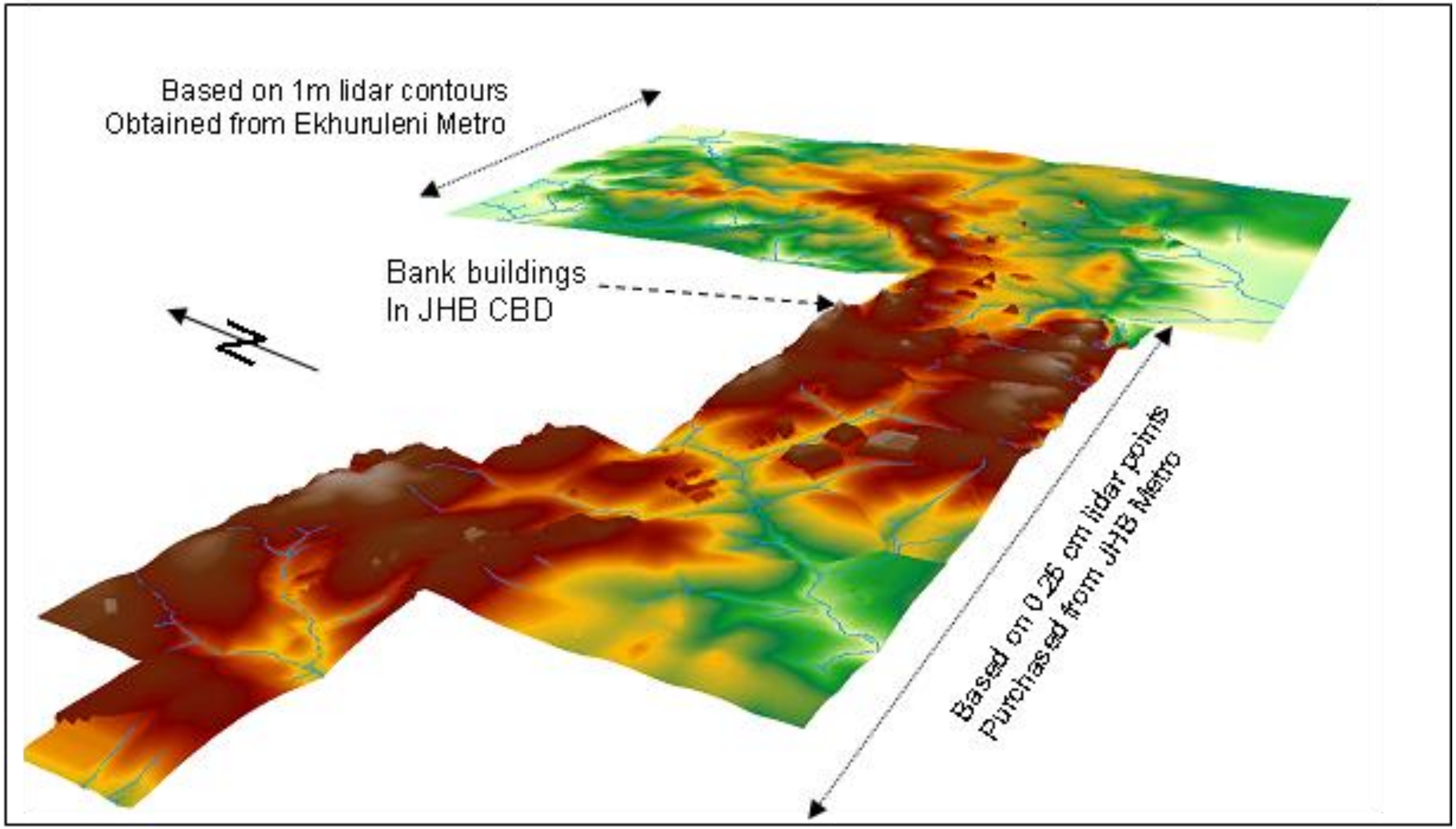

Fig. 8.3: DEM of the CR based on Lidar data with a vertical resolution of 25 cm and 1 m

8.1.3 Google Earth (GE) elevation data at 1m vertical resolution

Elevation data from Google Earth can be retrieved by hovering with the cursor over a certain point and noting the values displayed in the bottom task bar of the programme. The data is based on the SRTM data set, a global mission that generated elevation data for the entire globe (except the polar regions) using satellite-based synthetic aperture radar technology to determine surface elevations. This data seem to have been refined in Google Earth underlying an integrated DEM that allows 3D views on the area of interest. The highest vertical exaggeration of this DEM is 3 x times and can be set on a continuous scale starting from 1 (no exaggeration). An image of the GE DEM for the study area is depicted in Fig. 8.4.

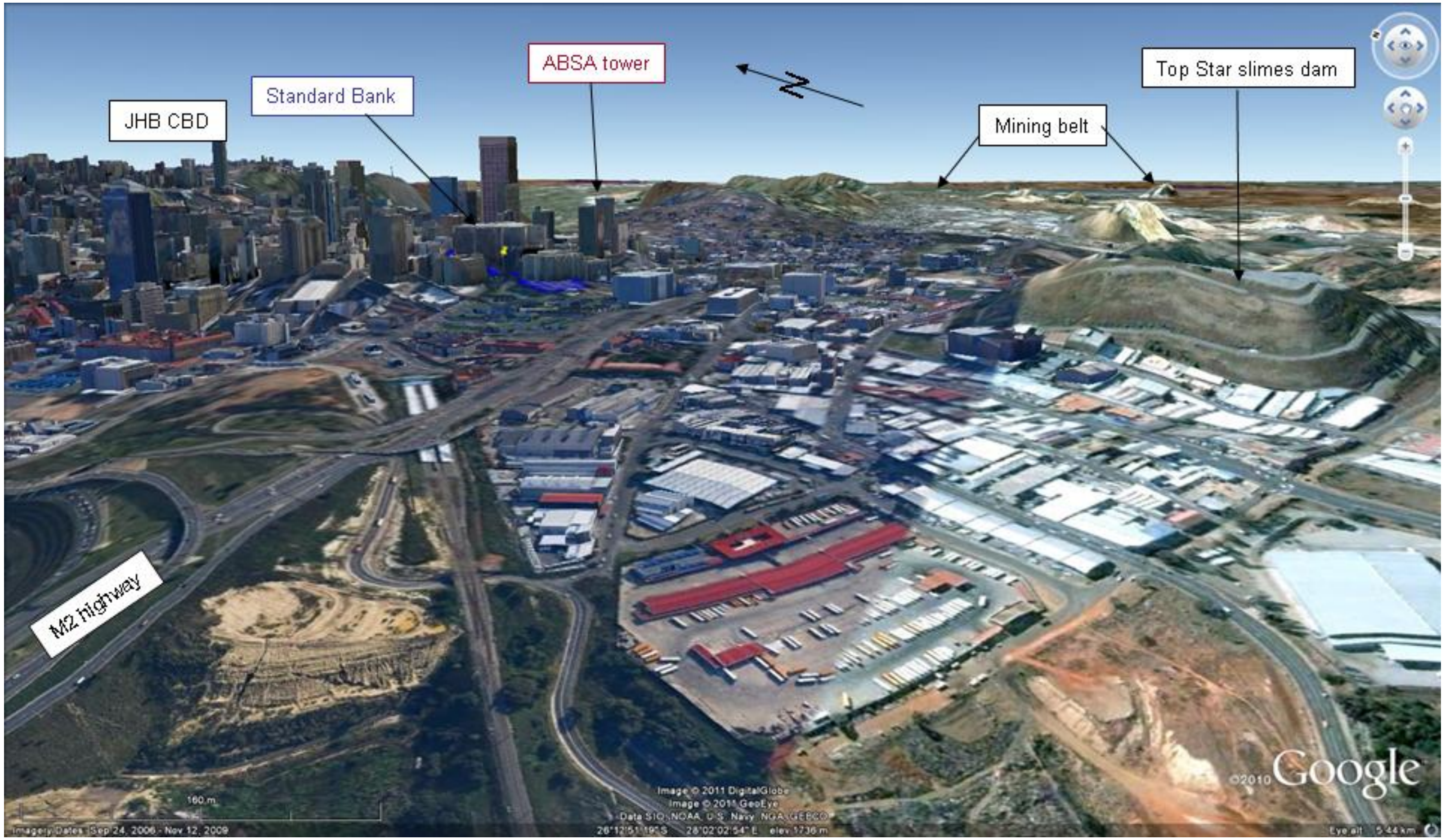

Fig. 8.4: Google Earth DEM image of the CBD next to the mining belt of the Central Rand goldfield

In interpreting GE elevation data, cognisance should be taken of the fact that radar technology frequently indicates higher surface elevations where the surface is covered by radar impenetrable layers such as water, vegetation (forests, reeds), buildings etc. In such areas the GE elevations tend to be too high. Furthermore, the relative error margin increases with the steepness of the relief and is smallest in topographically flat areas such as the Highveld. This generally favours the use of GE in this project. In addition to reliable elevation data, GE also provides frequently updated images of land use, which is often not the case with topographic maps that have much larger updating intervals (10 years). Since GE imagery is rapidly updated for areas of international interest, the coverage of JHB as one of the host cities of the 2010 Soccer World Cup is very recent.

8.2 Assessing the accuracy of the different elevation data sets

Since this risk assessment is essentially based on elevation differences, it is of crucial importance to ascertain the accuracy of the used elevation data. In order evaluate relative accuracies, high confidence elevation data for selected points in the study area were compared with elevations determined in Google Earth or by DEMs based on the 5 m contours of the CDNGI and the Lidar data sets.

Benchmark points with reliable elevation data include the following:

- Spot heights generated by the CDNGI as indicated in the 1:50,000 topographic map series with an associated vertical error of not larger than 30 cm. Two different sets of spot heights were used relating to the 2001/2002 edition and the 2007 edition of the associated topographic maps.

- Collar elations of shafts determined by ground-based trigonometric surveys conducted by registered surveyors on behalf of DRDGold. This dataset is the most accurate and high-confidence point elevations available for the study area.

8.2.1 Comparison to spot heights of the topographic map (CDNGI)

However, owing to their much higher density, spot heights are more readily available than trigonometric beacons. For the 1° x 1° area of the 1:50000 topographic map 2627 at total of 723 trigonometric beacons compares to 10501 spot heights. Selective comparisons between spot heights and trigonometric beacons in the study area indicate deviations of up to 2.5 m.

Moreover, a comparison of 20 selected spot heights in the study area used in the 2001 edition of the topo map to spot heights used in the 2007 edition revealed that only 1 of the 20 spot elevation used in 2001 was still used in 2007 while 8 x points completely disappeared and 11 x points shifted in horizontal position by 45 m to 312 m. Comparing the elevations of the 2001 spot heights to the available 2007 points showed a generally good agreement with 10 of the 12 points showing identical elevations while in 2 x cases a deviation of 2 m was found confirming the above stated error margin.

Comparing 20 x spot heights (from 2001) to 5m contours (from 2007) also showed a generally good agreement for the majority of points, but 3 x cases indicated differences of more than 20 m (max. 23 m) (we ascertained with satellite imagery of different dates – i.e. before and after 2001 - that this was not related to the removal of slimes dams that could have resulted in a change in surface elevation). The fact that these points were no longer displayed in the 2007 topo maps could perhaps indicate that their large inaccuracy has been detected and addressed. However, since Lidar data was generated in 2006 (flown in April/ May 2006), i.e. before the new topographic maps were released in 2007, it is likely that the lidar data was calibrated using the old (i.e. wrong) 2001 data points explaining the large deviation that Lidar data shows at these points.

In order to assess the accuracy of the different data sets elevations for selected points and contours are compared to various spot heights as indicated in the 1:50,000 topographic map. Comparing spot heights from the 2001 map edition to the 2007 edition indicated not only a significant increase in the total number of spot heights in 2007 but also some differences in the indicated elevations which needs to be considered when evaluating the differences found for the various data sets. The available spot heights of the 2011 and 2007 edition of the topographic 1:50,000 maps for the study area are depicted in Fig. 8.5

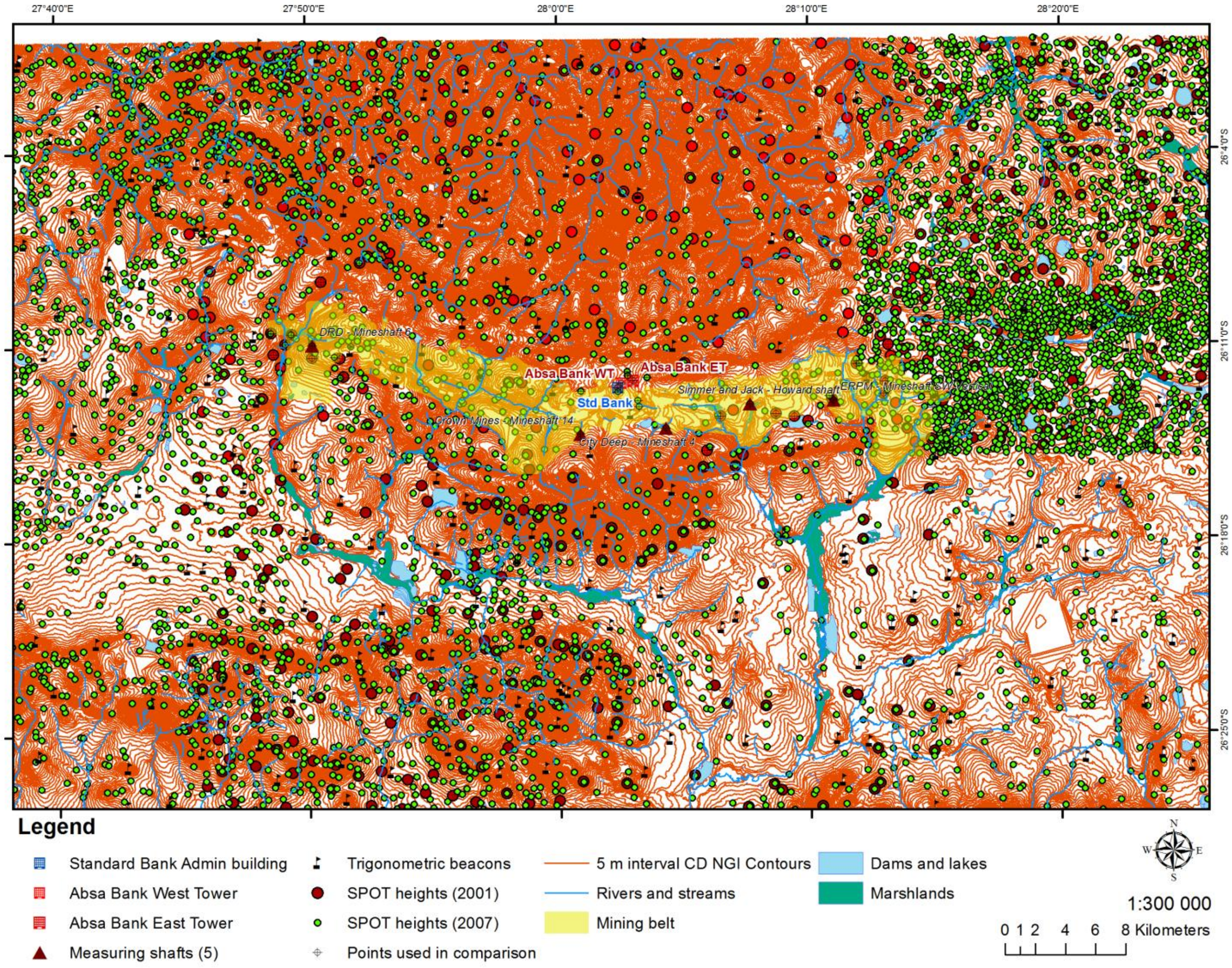

Fig. 8.5: Map of the study area depicting 5 m contours and spot heights of the 2001 and 2007 editions of the 1:50,000 CDNGI topographic map series.

Generally topographically relative flat areas were selected (i.e. not steep slopes or cliffs were possible elevation errors are commonly high) to compare the spot heights to the various elevation data sets. As most spot heights do not always exactly coincide with the available elevation data (contours and points) a certain error is introduced. However, the horizontal distance between the spot heights and the nearest elevation data ranged was kept to a minimum (0.3 m to a maximum of 4.4 m) which in topographically flat terrain results in relatively small elevation errors. The results of the comparisons are listed in the Tab. 8.1.

Tab. 8.1: Differences of elevation detected between the **2001/2** CDGI spot heights of the 1:50,000 topographic map series and the various elevation data sets

	Elevation [m amsl]				Deviation from spot height [m]		
CDNGI spot height (2001/2)	**SPOT Heights**	**Google Earth**	**5 m CDNGI-DEM**	**LIDAR-DEM**	**Google Earth**	**5 m CDNGI-DEM**	**LIDAR-DEM**
1	1706	1708	1707,98	1705,91	2	1,98	-0,09
2	1665	1670	1664,69	1664,21	5	-0,31	-0,79
3	1746	1747	1744,70	1744,15	1	-1,30	-1,85
4	1744	1734	1740,36	1740,30	-10	-3,64	-3,70
5	1723	1723	1725,09	1724,18	0	2,09	1,18
6	1691	1694	1688,95	1691,77	3	-2,05	0,77
7	1681	1681	1677,74	1679,72	0	-3,26	-1,28
8	1646	1653	1645,00	1647,49	7	-1,00	1,49
9	1722	1728	1723,83	1723,86	6	1,83	1,86
10	1684	1688	1684,71	1683,69	4	0,71	-0,31
11	1678	1688	1679,59	1680,27	10	1,59	2,27
12	1675	1649	1645,75	1646,17	-26	-29,25	-28,83
13	1789	1783	1782,13	1781,73	-6	-6,87	-7,27
14	1666	1668	1666,64	1664,59	2	0,64	-1,41
15	1689	1663	1656,87	1657,95	-26	-32,13	-31,05
16	1686	1654	1652,51	1652,02	-32	-33,49	-33,98
17	1644	1648	1643,29	1643,57	4	-0,71	-0,43
18	1647	1653	1645,34	1646,85	6	-1,66	-0,15
19	1619	1616	1613,53	1613,03	-3	-5,47	-5,97
20	1625	1632	1629,65	1628,49	7	4,65	3,49
21	1657	1661	1655,93	1657,00	4	-1,07	0,00
22	1678	1682	1676,83	1677,88	4	-1,17	-0,12
Number	22	22	22	22	22	22	22
Average					**-1,73**	**-5,00**	**-4,83**
Max	1789,00	1783,00	1782,13	1781,73	10,00	4,65	3,49
Min	1619,00	1616,00	1613,53	1613,03	-32,00	-33,49	-33,98
Largest deviation					-32,00	-33,49	-33,98
Smallest deviation					0	-0,31	0,00

Tab. 8.1 indicates that the average deviation from the 22 x selected spot heights is the smallest for Google Earth data (underestimating spot height elevation on average by 1.7 m) followed by Lidar and CDNGI data both with an average error of ca. 5 m underestimation (Tab. 8.1). As Lidar data are commonly referenced using official CDNGI elevation data, the similarity of the error margins of the two data sets is not surprising. What was however somewhat unexpected, is the fact that Google Earth data showed the best fit to the spot heights, while the CDGI DEM, which is based on these spot heights displayed the largest error margin. With 5 m the error is almost 3 x times larger than the error of the Google Earth rendering the GE data the most accurate.

Of particular concern is that the maximum deviations detected for all 3 x data sets are above 30 m. For elevation based risk assessments this is unacceptable high.

However, spot heights appear to have been updated in the 2007 edition of the 1:50,000 topographic maps with possible consequences for the accuracy of the different data sets. Tab. 8.2 lists the differences between the elevations indicated by the different DEMs and the updated spot heights from 2007.

Tab. 8.2: Differences of elevation detected between the **2007** CDGI spot heights of the 1:50,000 topographic map series and the various elevation data sets

	Elevation [m amsl]				Deviation from spot height [m]		
Spot heights (2006/7)	SPOT Heights	Google Earth	5 m CDNGI-DEM	LIDAR-DEM	Google Earth	5 m CDNGI-DEM	LIDAR-DEM
1	1746	1737	1745,23	1743,61	-9	-0,77	-2,39
2	1689	1695	1689,86	1691,23	6	0,86	2,23
3	1679	1681	1677,51	1679,21	2	-1,49	0,21
4	1644	1653	1644,88	1645,48	9	0,88	1,48
5	1722	1728	1723,74	1722,54	6	1,74	0,54
6	1683	1684	1682,71	1681,51	1	-0,29	-1,49
7	1678	1683	1676,49	1678,69	5	-1,51	0,69
8	1789	1782	1792,22	1787,38	-7	3,22	-1,62
9	1666	1668	1666,88	1664,74	2	0,88	-1,26
10	1649	1653	1646,76	1647,44	4	-2,24	-1,56
11	1618	1620	1618,66	1617,64	2	0,66	-0,36
12	1657	1663	1655,86	1657,00	6	-1,14	0,00
Number	12	12	12	12	12	12	12
Average					**2,25**	**0,07**	**-0,29**
Max	1789,00	1782,00	1792,22	1787,38	9,00	3,22	2,23
Min	1618,00	1620,00	1618,66	1617,64	-9,00	-2,24	-2,39
Largest deviation					9,00	3,22	-2,39
Smallest deviation					1	-0,29	0,00

The results for the 12 benchmark points displayed in the upper part of the table indicate that the 2007 spot heights introduced a significant reduction of the error margin associated with the CDNGI data and the lidar data which now match the spot heights very well showing average deviations of only +7 cm and -29 cm respectively. In contrast, the average error of the Google Earth data changed considerably from an underestimation by 1.7 m to an average overestimation of over 2 m (Tab. 8.2). Based on the updated spot heights the CDGI data clearly displays the highest accuracy followed closely by the Lidar data. For all 3 x data sets a significant reduction in the maximum deviations is noticeable from over 30 m down to +/-9 m for Google Earth, +3.2 m for CDNGI and -2.4 m for the Lidar set (Tab. 8.2).

Since densely build-up areas such as the inner city of JHB display specific problems for remote sensing techniques such a radar (used for determining the SRTM elevation data used in Google Earth) the overall accuracy of the 3 x data sets was specifically investigated for the CBD around the 3 x identified key buildings. For this purpose the elevation of a number of 2007-spot heights in the buildings' immediate vicinity was compared against the elevations indicated by the 3 x data sets (Tab. 8.3).

Tab. 8.3: Differences of elevation detected between the **2007** CDGI spot heights of the 1:50,000 topographic map series and the various elevation data sets for points near the key bank buildings

	Elevation [m amsl]				**Deviation from spot height [m]**		
2007 spot heights near banks	**SPOT Heights**	**Google Earth**	**5 m CDNGI-DEM**	**LIDAR-DEM**	**Google Earth**	**5 m CDNGI-DEM**	**LIDAR-DEM**
1	1703	1712	1702,95	1702,62	9	-0,05	-0,38
2	1739	1763	1739,65	1739,98	24	0,65	0,98
3	1741	1768	1740,58	1743,26	27	-0,42	2,26
4	1709	1717	1709,84	1709,34	8	0,84	0,34
5	1742	1752	1741,23	1744,24	10	-0,77	2,24
6	1742	1746	1741,63	1742,64	4	-0,37	0,64
7	1753	1748	1754,98	1754,86	-5	1,98	1,86
8	1758	1753	1762,30	1757,67	-5	4,30	-0,33
Number	8	8	8	8	8	8	8
Average					**9,00**	**0,77**	**0,95**
Max	1758,00	1768,00	1762,30	1757,67	27,00	4,30	2,26
Min	1703,00	1712,00	1702,95	1702,62	-5,00	-0,77	-0,38
Largest deviation					27,00	4,30	2,26
Smallest deviation					4	-0,05	-0,33

Based on 8 x available 2007 spot heights (Tab. 8.2), the average deviations for the near-bank area somewhat differs from the ones based on 12 spot heights discussed earlier (Tab. 8.1). For all 3 x sets the accuracy was found to be lower than that of the spot heights. The largest deviations are again displayed by Google Earth indicating an average overestimation of 9 m, compared to 80 cm and 95 cm to CDNGI and Lidar respectively (Tab. 8.3). Of concern is the finding that the highest deviation also increased for all 3 x data sets with a maximum of +27m detected for Google Earth more than 10 x times larger than the error for the Lidar set (+2.3 m). The fact that the smallest deviation for GE data is still +4 m, supports the suspicion that the radar based techniques tend to overestimate elevations in densely build up areas or any other area covered by layers impenetrable for radar (forests, reeds etc.). Although comparatively low in relation to the Google Earth deviations, the overestimation of maximal 4.3 m by the CDNGI set is significantly larger than the 2.5 m given by the CDNGI as a maximal error.

The resulting differences in elevations of the 3 x identified key bank buildings are shown in Tab. 8.4

Tab. 8.4: Differences between Google Earth elevations and 5 m CDNGI-DEM and the 1m Lidar DEM for the 3 x identified key bank buildings

		Elevation [m amsl]			Deviation from Google Earth [m	
	Bank Buildings	Google Earth	5 m CDNGI-DEM	LIDAR-DEM	5 m CDNGI-	LIDAR-DEM
Standard Bank new admin		1747	1739,02	1741,49	-7,98	-5,51
	ABSA tower W	1760	1734,90	1736,43	-25,10	-23,57
	ABSA tower E	1744	1725,89	1725,95	-18,11	-18,05
	Number	3	3	3	3	3
	Average				**-17,06**	**-15,71**
	Max	1760,00	1739,02	1741,49	-7,98	-5,51
	Min	1744,00	1725,89	1725,95	-25,10	-23,57
	Largest deviation				-25,10	-23,57
	Smallest deviation				-7,98	-5,51

Using Google Earth as reference for the surface elevations of the bank buildings it appears that the CDNGI data and the Lidar data are considerably lower by a margin of 17 m and 15.7 m respectively (Tab. 8.4). Since Google Earth data is radar based and thus presumably less reliable for covered surfaces all elevations of key buildings are used from the other 2 data sets.

8.2.2 Comparison to surveyed shaft collar elevations

(i) Determination of shaft collar elevations

In addition to spot heights, elevations of 4 x DRD monitoring shafts were used as benchmarks as their collar elevation were determined by trigonometric surveys and are therefore highly accurate. In fact, following a request by the Department for Mineral Resources, DRD recently had all monitoring shafts re-surveyed by a contracted, registered company adding additional confidence to the reliability of the data (Labuschagne, 2011b). As errors may have legal implications, these elevations are treated as accurate on the highest possible confidence levels relative to all other types of elevation data used in this report. The elevations for the 5 x shafts are given in Tab. 8.5.

Tab. 8.5: Coordinates and collar elevations of monitoring shafts [mamsl] according to the DRD survey and Scott (1995)

Data source	Name of monitoring shaft	Latitude (S) (DRD coordinates acc. to Google)	Longitude (E)	Altitude (elevation) [mamsl]
DRD*	DRD no. 6 shaft	26 10' 57.01"	27 50' 12.08"	**1705.00**
Scott (1995)	DRD: 6	26°10'46.95"S	27°50'10.94"E	1698**
deviation		350 m N		-7.0 m
DRD*	Crown Mines: no. 14 shaft (Gold Reef City)	26 14' 14.09"	28 00' 52.20"	**1706.88***
Scott (1995)	Crown mines no 14 shaft	26°14'13.04"S	28°0'53.93"E	1698**
deviation		61 m NE		-8.9 m
DRD*	City Deep: no. 4 shaft (market)	26 14' 01.33'	28 04' 16.11"	**1691.80**
Scott (1995)	City Deep: 4	26°14'0.25"S	28° 4'14.88"E	1682**
deviation		52.3 m NW		-9.8 m
DRD*	Simmer & Jack: Howard shaft	26 13' 11.33"	28 07' 41.38"	**1652.93**
Scott (1995)	only shaft for S&J	26°12'15.48"S	28° 8'43.67"E	1687**
deviation		2460 m NE		+34 m
DRD*	ERPM: SW Vertical shaft	26 13' 03.04"	28 10' 57.59"	**1653.24**
Scott (1995)	ERPM: South West	26°13'9.04"S	28°10'51.52"E	1646**
deviation		245 m SW		-7 m

* Labuschagne (2010 and 2011a/b)

** Data is given at 1 m precision, i.e. no decimals are shown

It appears that collar elevations determined by Scott (1995) using 1:10,000 orthophotos deviate considerably from the surveyed data ranging from an underestimation of almost 9m to an overestimation of 34 m (Tab. 8.5). To some extent this can perhaps be explained by the fact that the Scott (1995) positions of the shafts differ somewhat from the correct location indicated by the survey (Labuschagne, 2011b). Being based on manually determined elevations using 1:10,000 orthophotos shaft elevations given in Scott (1995) are to be regarded as less reliable as indicated by Scott (1995) himself stressing that more accurate determinations may become necessary.

(ii) Data comparison

Since the accuracy of Google Earth data may be different in open terrain such as the mining belt and because some of the results of the above accuracy tests are partly inconsistent, all 3 x data sets were additionally compared to surveyed elevations of the 5 x monitoring shafts currently used by DRDGold to monitor the water levels in the Central Basin. The results of the comparison are depicted in Tab. 8.6.

Tab. 8.6: Differences between surveyed collar elevations of the 5 x monitoring shafts used to measure mine water levels in the CR mine void system (Labuschagne, 2010) and the various elevations data sets (Google Earth, 5 m CDNGI-DEM and the 1m Lidar DEM)

	Elevation [m amsl]				Deviation from surveyed height [m]		
Monitoring shafts	**surveyed height**	**Google Earth**	**5 m CDNGI-DEM**	**LIDAR-DEM**	**Google Earth**	**5 m CDNGI-DEM**	**LIDAR-DEM**
DRD 6#	1705,00	1705	1699,86	1697,74	0,00	-5,14	-7,26
GRC (CM 14#)	1706,90	1706	1695,75	1699,55	-0,90	-11,15	-7,35
City Deep 4#	1691,80	1692	1681,56	1682,87	0,20	-10,24	-8,93
S&J Howard #	1652,90	1654	1645,38	1648,98	1,10	-7,52	-3,92
ERPM SWV #	1653,20	1652	1645,37	1646,56	-1,20	-7,83	-6,64
Number		4	4	4	4	4	4
Average					**-0,20**	**-9,18**	**-6,71**
Max		1706,00	1695,75	1699,55	1,10	-7,52	-3,92
Min		1652,00	1645,37	1646,56	-1,20	-11,15	-8,93
Largest deviation					-1,20	-11,15	-8,93
Smallest deviation					0,00	-5,14	-3,92

In contrast to spot heights, with the discussed uncertainty attached to them, the comparison with the surveyed collar elevations of the 5 x monitoring shafts which are more or less evenly distributed across the entire mining belt from DRD in the west and ERPM in the east suggests that elevations determined in Google Earth are significantly more accurate than those derived from DEM based on CDNGI 5 m-contours or the 1 m-lidar data (Tab. 8.6). Since elevations are given in Google Earth for any point on which the cursor is placed on with 1 m precision, direct comparison between Google elevations and surveyed ground heights are possible. For interpreting the detected deviations it needs to be noted that collar elevations of shafts refer to the upper level of the concrete slab that is placed directly on surface to stabilise the shaft. Thus the collar elevation may be between 10 cm and 30 cm above the true surface level indicated by Google Earth.

Tab. 8.6 indicates that the shaft elevations retrieved from DEMs based on the 5-m-contours from the official 1:50000 topographic map and on 0.25 m/1-m lidar data underestimate the true (i.e. ground surveyed) collar elevations of the 5 x monitoring shafts on average by ca. 8 m and 7 m with the largest deviations being around 10 m and 9 m respectively. This is a considerable margin of vertical error given that the 5 m contours are retrieved from the official topographic maps which, according toe the CDSM (now CDNGI), have a maximum vertical error of 2.5 m, i.e. 4 x smaller than the detected error. The same is unfortunately true for the rather expensive lidar data set which we hoped would be the most accurate of all elevation data. However, despite a vertical resolution of 25 cm suggesting a high degree of accuracy deviations of up to 35 times the vertical resolution were found. The similarity in the extent of the deviations regarding average, smallest and largest error between the 5 m topographic contours and the lidar data may result from the fact that lidar data are commonly referenced using official topographic maps.

Tab. 8.6 indicates that the Google Earth based elevation data are for all 5 x shafts are very close to the surveyed collar elevations, showing a slight average underestimation of just 20 cm. This could be explained by the fact that the surveyed collar elevation refers not directly to the surface but to the upper level of a concrete slabs covering the shaft collars. With the thickness of these concrete covers ranging from 15 cm to 30 cm this results in collar elevations being on average some 20 cm higher than the surrounding surface detected in Google Earth. Taking into account that no decimals are shown in Google Earth, the match with surveyed data is convincing. This was also confirmed by the chief surveyor of DRD at ERPM Mr. V. Labuschagne stating that Google derived elevations are commonly accurate within 25 cm in flat areas and 50 cm on steeper terrains (Labuschagne, 2011).

The general good fit of Google Earth data compared to CDGI elevations is supported by Hoffmann & Winde (2010) pointing out that their generated Google Earth-based 1-m-interval DEM showed less deviations from surveyed benchmark elevations than found between the 5-m and 20-m-contours used in standard topographic maps despite the latter being generated by the same source (i.e. the CDSM).

8.3 Determination of the critical pile base level (CPBL)

As indicated in connection with the definition of the risk concepts the determination of the CPBL involves the following aspects:

(1) Determination of PBLs for the identified key buildings
(2) Determination of the critical PBL as the pile base level of the most exposed building.

Exposure to mine water is regarded as a function of:

- the distance of the building to the MR outcrop (m) as the closest potential decant area of a flooded mine void;
- -the hydraulic connectivity of the rock between the key buildings and the mine void.

8.3.1 Determination of pile base levels (PBL) for key buildings

The PBL is defined as the elevation of the deepest underground structure associated with the 2 x identified key buildings. It is determined by the elevation of the topographic surface on which the buildings are constructed and the depth to which the basement and the supporting piles extend into the bed rock. The basement depth is calculated as the sum of the height of the different basement levels as retrieved from construction plans or indicated by competent persons while the maximal pile depth is a generic values that was provided by a construction company specialising in pile construction with a long track record of projects in central JHB (Esor Franki).

According to the construction company Esor Franki Ltd. the ABSA towers as most other high rise buildings in central JHB were built on solid unweathered bedrock minimising the depth to which concrete support pillars (termed ‘piles’) penetrate into the geological underground (Olivier, 2011). According to Olivier (2011) the maximum pile length of other – non-problematic buildings in the CBD is 12 m.

An exception where deeper piles had to be used is described by Brink (1978) for the old Std. Bank Centre (cnr. Fox- and Simmonds Street, meanwhile sold by Std. Bank) where weathering at the contact zone of quartzite and diabase resulted in soft rock that required deeper anchoring. Below the 5 x basement levels extending to 18.3m below surface, concrete piles of different length penetrate into the bedrock with a maximum length of some 38 m reaching a total depth of 56.3 m below the original surface (estimated from the drawing in Fig. 8.6).

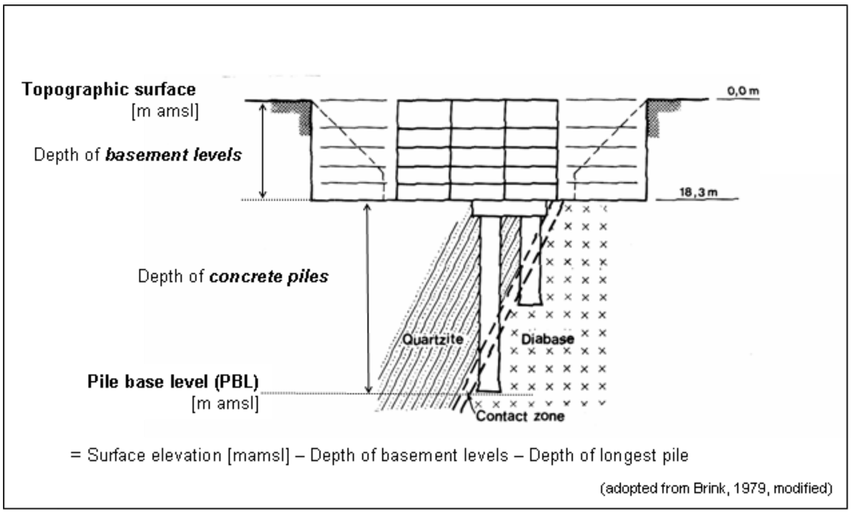

Fig. 8.6: Cross section through the excavation pit for the old Std. Bank centre (cnr. Fox- and Simmonds Street, meanwhile sold by Std. Bank) illustrating how the pile base level (PBL) is calculated

Protruding through the original water table found at 23 m below surface the completed basement structures were subsequently water proved by a installing a 75 mm-wide drainage cavity between the concrete wall and the brick skin. Rows of perforated concrete filter blocks in the concrete wall prevent the built up of external water pressure. A drainage layers is incorporated below the surface bed.

For the ABSA towers the construction company Asakheni Consult. Engineers (2011) reported 4 x basement levels of which the top 3 ones have an average height of 2.7 m while the lowest basement level displays a height of 3.0 m resulting in the bottom of the basement being at 11.1 m below surface. Since solid unweathered bedrock was encountered during excavation piles only extent a maximum of 10 m below basement reaching a total depth of 24 m below the original surface. All piles were reported to be constructed in quartzite that was dry at the time. While during excavation and pile construction no groundwater was encountered sumps have been included in the basement design to allow for later installation of pumps should this become necessary.

For Standard Bank the focus is on their new administration building. Wile the building may not be considered a high-rise structure with deep basements it is located furthest to the south of all 10 objects and thus the closest to the mined MR outcrop zone and the underlying mine void.

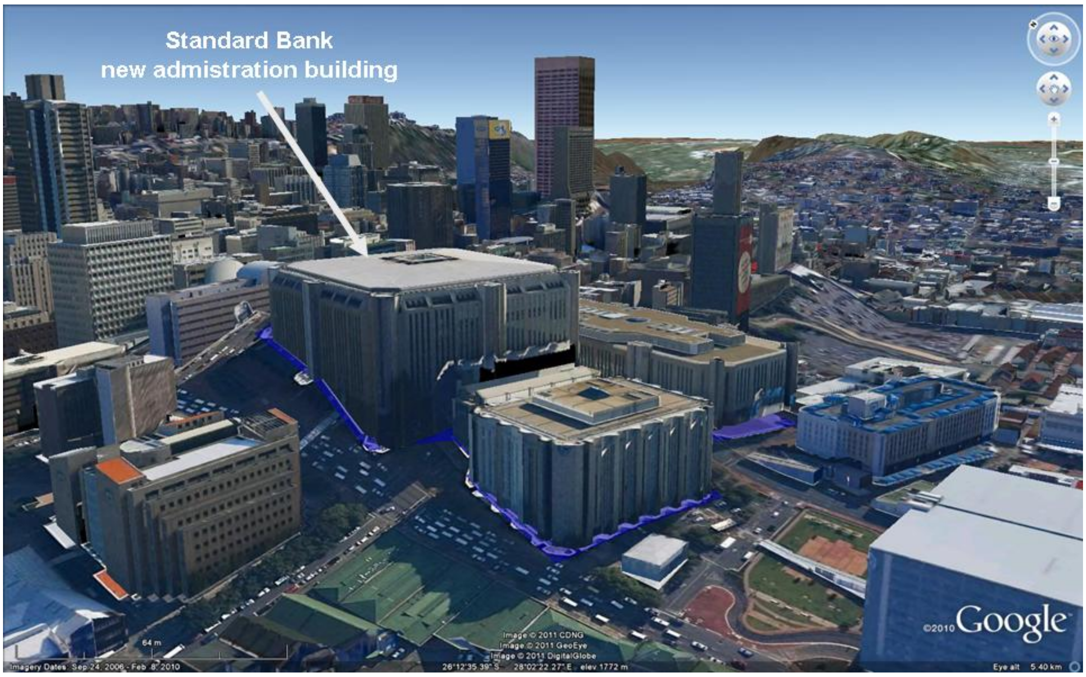

Fig. 8.7: Google 3D view of the Standard Bank new administration building

The relevant data for three selected bank buildings are given in Tab. 8.7.

Tab. 8.7: Surface elevation and preliminary PBLs for three selected bank buildings according to different elevation data sources [mamsl]

Building	Street address	Surface level **[mamsl]** (lowest) **5-m CDSM DEM**	**Google Earth (GE)** (dev. from 5m DEM) *	**Lidar - 25cm points** (dev. from 5m DEM)	Lowest surface level [mamsl] (source)	Total depth of basement + piles **[mbs]**	PBL [mamsl] (data source)
Standard Bank new admin building	Simmonds-Frederick Streets	**1740.0****	1755 (-15,3m)	1741.5 (+1.5m)	1740 (CDSM)	**20**	**1720.0** (CDSM)
ABSA tower W*	Fox-von Brandis Streets	**1734.9**	1755 (-20,1m)	1739.4 (m)	1739.4 (lidar)	**21.1**	**1718,9** (lidar)
ABSA tower E (logo)	Troy Str – Main Street	**1726.1**	1744 (-17.8m)	1726.0 (m)	1726.0 **(lidar)**	**21.1**	**1704,9** (lidar)
	Av. deviat.		-21.1				

* Elevation derived from DEM based interpolation between 5m-contours

** Confirmed exactly by elevation data on building plans provided by MacLeod (2010)

Since Google Earth elevation data are ultimately derived from radar based SRTM data, it is possible that ground elevations of areas covered by vegetation (reeds, forests etc.) or buildings (settlements, cities) are prone to overestimation. Since the mining belt is largely open and free from vegetation or buildings, shaft elevation derived from Google Earth are very accurate. However, the generally higher levels found in the CBD (15-20m above CDSM and Lidar data) may reflect the overestimation of ground surface levels in covered areas. For this reason, and in order to apply the most risk-conservative approach as mentioned earlier, the lidar data (as lowest of all three data sets) is used to determine the ground surface elevation of the three bank buildings (Tab. 8.7).

At the time of writing no information as to the presence of water during excavation could be obtained. According to MacLeod (2010) the deepest level of the 3 x storey basement of the new admin building is at 1743 mamsl some 10 m below surface (1740 mamsl). For the length of the concrete piles extending into the bedrock below the basement no data could be found in the construction plans provided. A conservative estimate by MacLeod (2010) suggested a maximal pile depth of 20 m below the lowest basement level. However, information provided by Eso Franki (2011), a pile construction company routinely operating in the CBD of JHB, indicated that the deepest level is 12 m while 10m are much more common as all high-rise buildings are situated on solid bedrock. The only exception is the old

Standard Bank Tower (Centre) where piles extended some 30 m through a highly weathered contact diabas-quartzite contact zone before reaching solid bedrock (Brink 1979) as mentioned earlier.

Tab 8.8 summarises the pertinent data for the Std. Bank new admin building as the one closest to the MR outcrop zone and the ABSA tower at the cnr Troy and Main Street as the highest building with the deepest piles installed.

Tab. 8.8: Depth of basements and piles for key buildings of Std. Bank and ABSA

Building	Street address	Nr. of basement levels	Nr. and av. height of basement levels	Total depth of basement bottom [mbs]	length of bottom of deepest pile [mbs]	Total depth basement + pile **[mbs]**
Std. Bank new admin building	Simmonds-Frederick Streets	3***	3 x 3.33	10**	10	**20**
ABSA tower West	Fox-von Brandis Street	4*	3 x 2.7m* 1 x 3m	11.1*	10*	**21.1***
ABSA tower East (logo)	Troy Str – Main Street	4	3 x 2.7m 1 x 3m	11.1	10	**21.1**

* As no specific data were supplied for this (older) building the data provided for the new building was used.
** MacLeod (2010): basement level at 1430 mamsl compared to 1740 mamsl surface level: basement depth 10 m
*** Information provided telephonically by security in the admin building of Standard Bank (Tel. 011/ 299 4701; 0860123000).

8.3.2 Selection of the critical PBL (CPBL)

(i) Distance of key buildings from MR outcrop zone

The location of the outcrop in relation to the key building was derived from various sources since the exact location of the outcrop zone differs between different sources. The largest scale depiction of the outcropping MR in central Johannesburg was found in Mendelsohn and Potgieter (1986) presumably displaying the highest accuracy (Fig. 8.8).

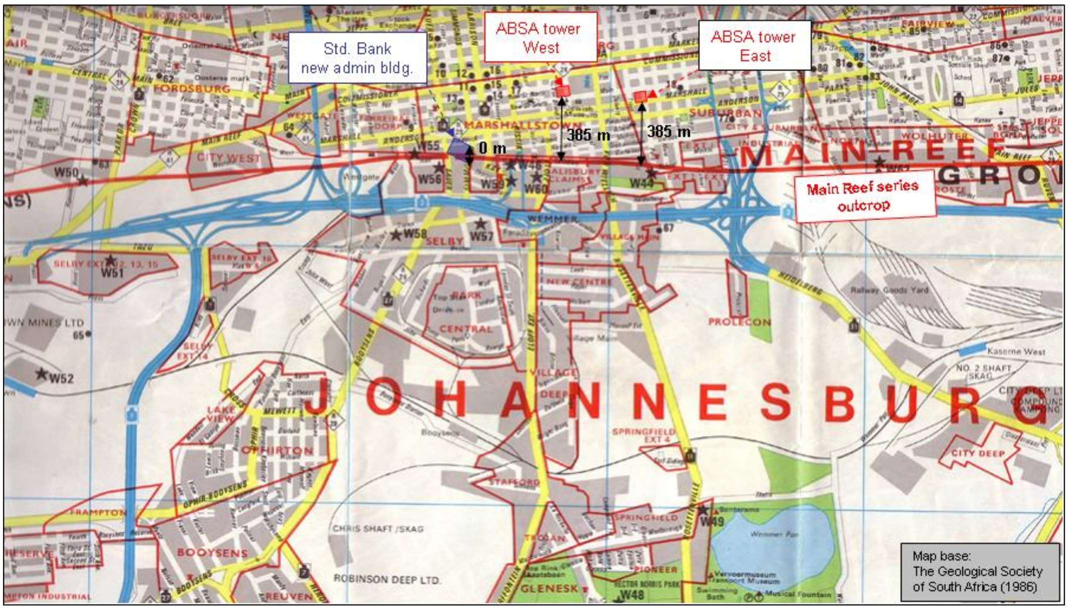

Fig. 8.8: Outcrop of MR in relation to bank buildings

Importing the location of the new admin building into this map indicates that it is placed right on top of the outcropping reef bands of the MR series, while the 2 ABSA towers (East and West) are both located almost 400 m away to the north of the outcrop. This is largely confirmed by using 2 x additional maps depicting the course of the MR outcrop which had been imported into Google Earth (Fig. 8.9).

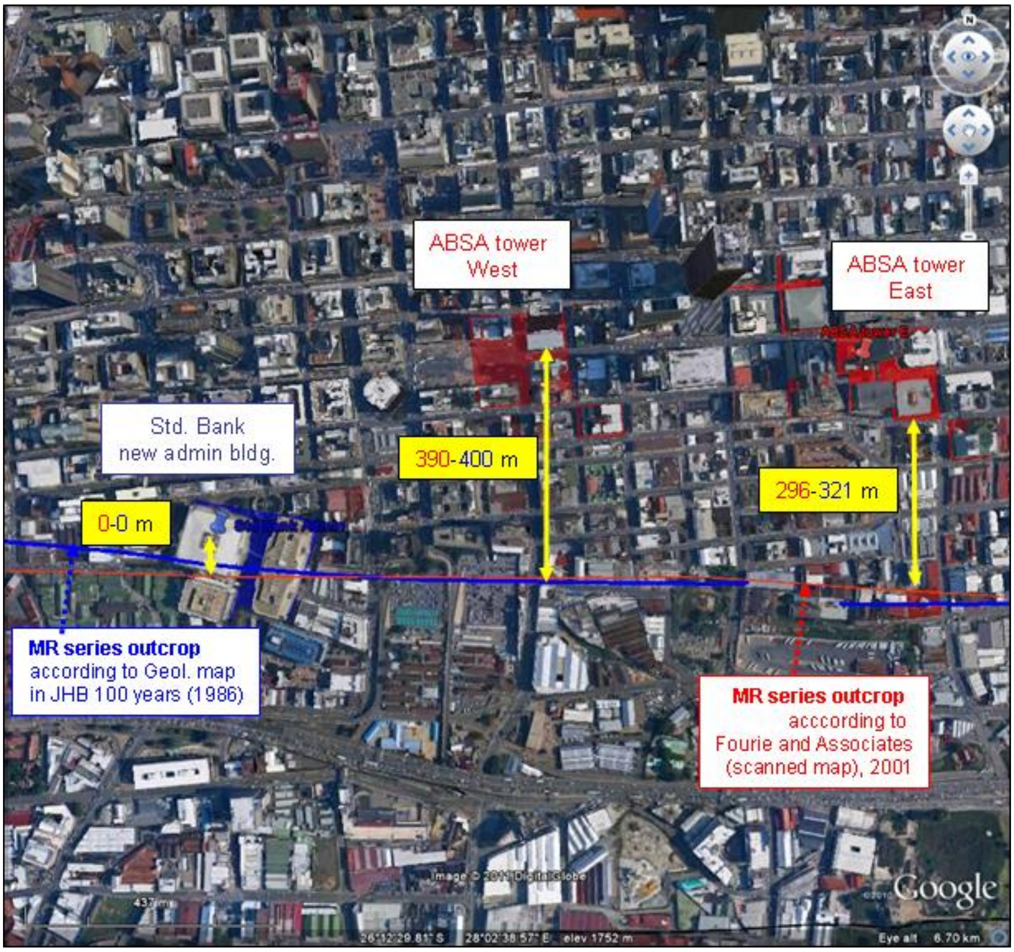

Fig. 8.9: The identified key buildings in relations to the MR outcrop (based on Google Earth satellite imagery)

The satellite image in which all new bank buildings are already depicted confirms that the new administration building of Standard Bank is indeed situated directly on the outcrop while the 2 x ABSA towers are located between 300 m (tower E) and 400 m (tower W) to the north. The distances of the key buildings from the outcropping MR series are listed in Tab. 8.9.

Tab. 8.9: Horizontal distance of key bank buildings to the outcropping Main Reef series

Building	Coordinates	Street address (cnr. Of)	PBL [mamsl] (data source)	Distance from MR outcrop [m] acc. to below sources:			Potential mine water exposure
				1:21,400 street map JHB	GE – JHB 100 years (1986)	GE - Fourie (2001)	
Standard Bank	26°12'33.57"S 28° 2'21.78"E	Simmonds-Frederick Street	1720.0 (CDSM)	**0**	**0**	**0**	highest
ABSA tower W	26°12'22.47"S 28° 2'37.15"E	Fox-von Brandis Street	1718,9 (lidar)	**385**	**400**	**390**	second highest
ABSA tower E	26°12'22.82"S 28° 2'59.63"E	Troye– Main Street	1704,9 (lidar)	**385**	**321**	**296**	third highest

With Std. Bank being the closest of all 3 buildings to the MR outcrop it is clearly the most critical building for assessing a possible flooding risk associated with possibly decanting mine water

(ii) Hydraulic connectivity of rocks between key buildings and the mine void

Being located right on top of the outcropping Main Reef Leader and perhaps the South Reef which are the most extensively mined reefs in the entire CR the basement of Std. Bank coincides with the entrance of the underground mine void which – provided the mine water level will rise to this elevation – would be the decant level (Fig. 8.10).

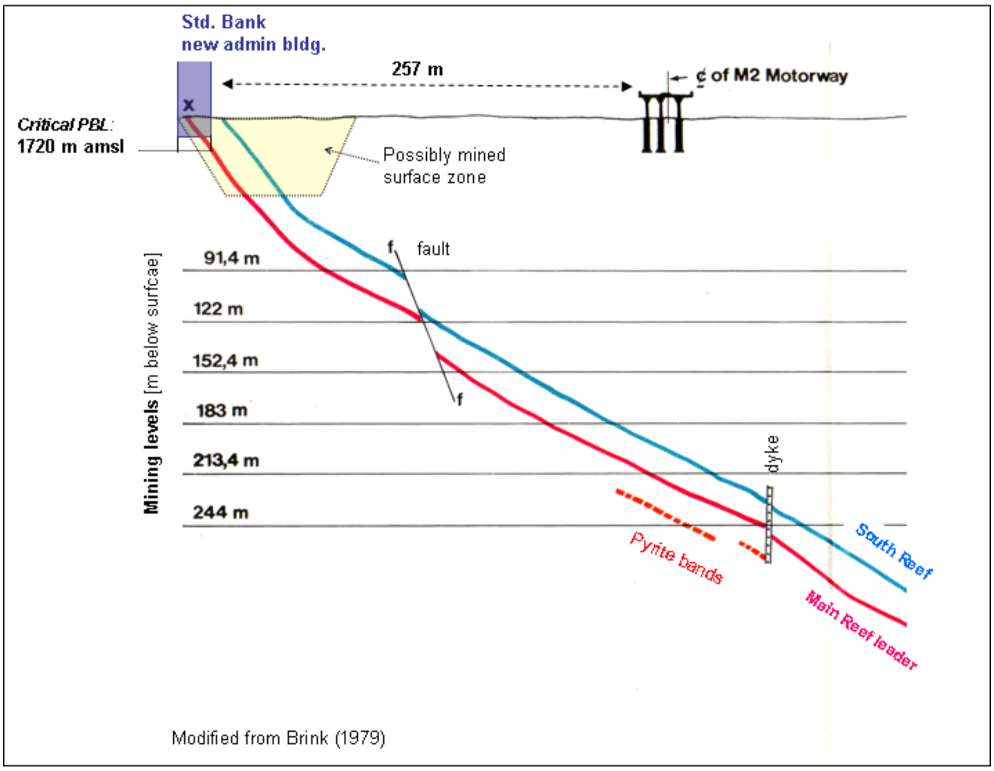

Fig. 8.10**:** Cross section of outcrop of MR series in relation to the STD. Bank new admin building situated right on top of the outcropping reef bands (modified from Brink, 1979; originally shown for a profile at the Kazerne goods office)

The cross section schematically depicts the location of the outcrop zone where the two reef bands were originally mined from surface. This zone disturbed to some 40m below surface was often filled with unconsolidated material and generally shows a high infiltration potential and elevated lateral permeability. Complete or partial water saturation caused by rising mine water may therefore result in reduced shear stress and liquefy porous fill material such as sand and tailings. This, in turn, could compromise geotechnical stability. Although no record was found mentioning the encounter of unconsolidated material while excavating the basement for the building the possibility of flooding induced instability should be investigated – even if only the lower part of the 40-m-deep zone maybe affected.

Being situated right on top of the mine void entrance also means that no rock as possible buffer between the void and the basements exists as is the case with the 2 x ABSA towers. I.e there is no possible barrier function of the rock for decanting mine water.

Furthermore, during excavations of the basement pit, the number 1 level of the old mine workings of the Ferreira Gold Mine (which by the way was a client of Std. Bank at the time and was a particular rich gold mine owing to its location on top of the MR leader and the SR which both yielded exceptionally high Au grades. The mine closed in the late 1920s, Editing Committee, 1986) were discovered at a depth of 30 m below surface connected to the up to 1000-m-deep mine void further south (Standard Bank, undated).

An example of shallow workings of the same mine (Ferreira Deep) being opened up in the early 1980s during excavations for the headquarters of the Priceforbes Federal Volkskas (PFV) at the corner of Sauer and Hall streets in the CBD is shown in Fig. 8.11.

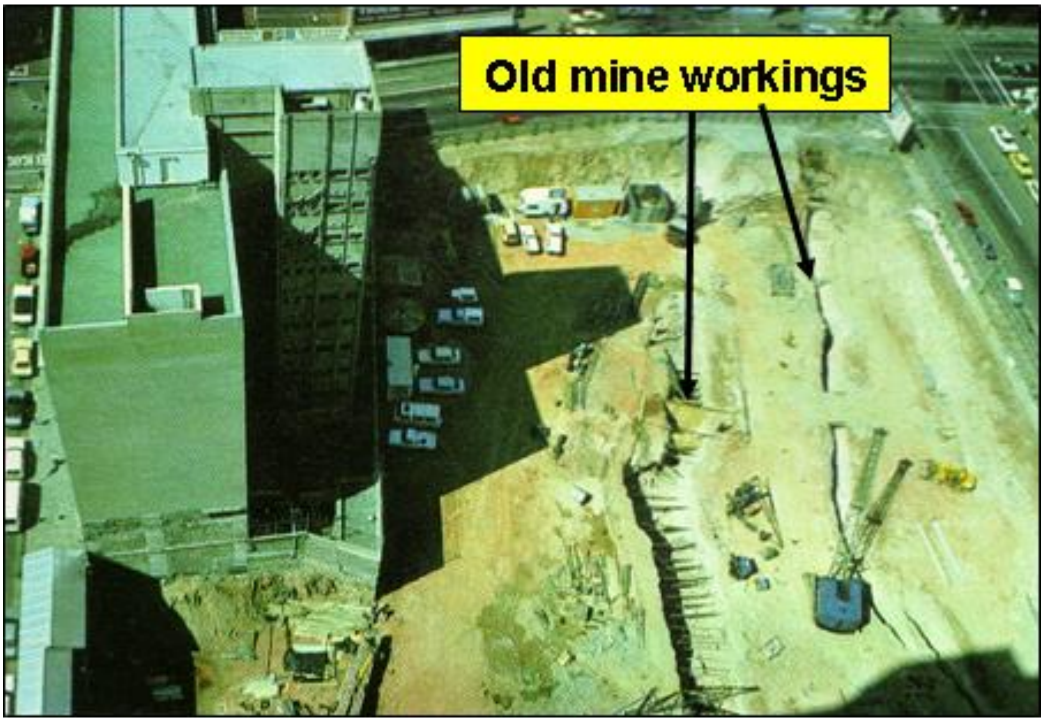

Fig. 8.11: Shallow underground workings of the Ferreira Deep gold mine opened up during excavations for the headquarters of the Priceforbes Federal Volkskas (PFV) at the corner of Sauer and Hall streets in the JHB CBD (early 1980s) (Photo: Editorial Committee, 1986)

To preserve the historical evidence efforts were made to convert the shaft into a publicly accessible site with an associated museum (Fig. 8.12).

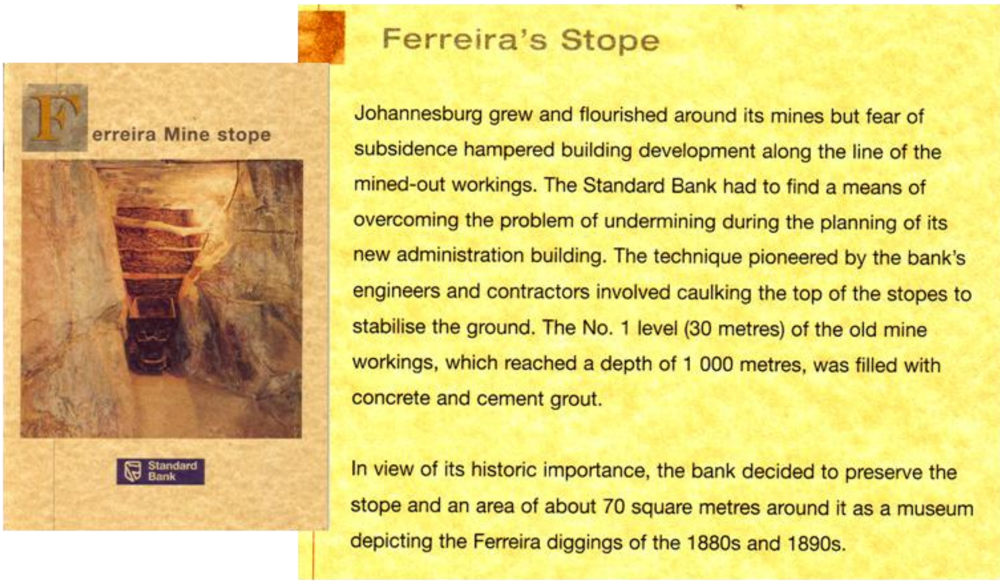

Fig. 8.12: Booklet of Standard Bank for the Ferreira stope which was uncovered during excavations for the new administration building of Standard Bank and is now preserved at the basement of the building as a museum open to the general public

In order to overcome problems possibly associated with shallow underground mine working such as subsidence the tops of the stopes were filled with a waterproof sealant (caulking) to stabilise the ground.

Furthermore, the shallowest workings at no 1 level (30 mbs) of the mine void which reaches a depth of 1000 m were completely filled with concrete and cement grout.

Even though the workings have been plugged, the new Std. Bank admin building is generally more exposed to possible flooding as it is located right on the mine void entrance, suggesting that the PBL of Std. Bank should be used as critical PBL.

On the other hand however, it appears that the PBL of the ABSA tower East is considerably lower than that of Std. Bank resulting in a higher possible flooding risk even though the exposure pathway to mine water may be less direct. A conceptual model of the resulting situation is schematically depicted in Fig. 8.13.

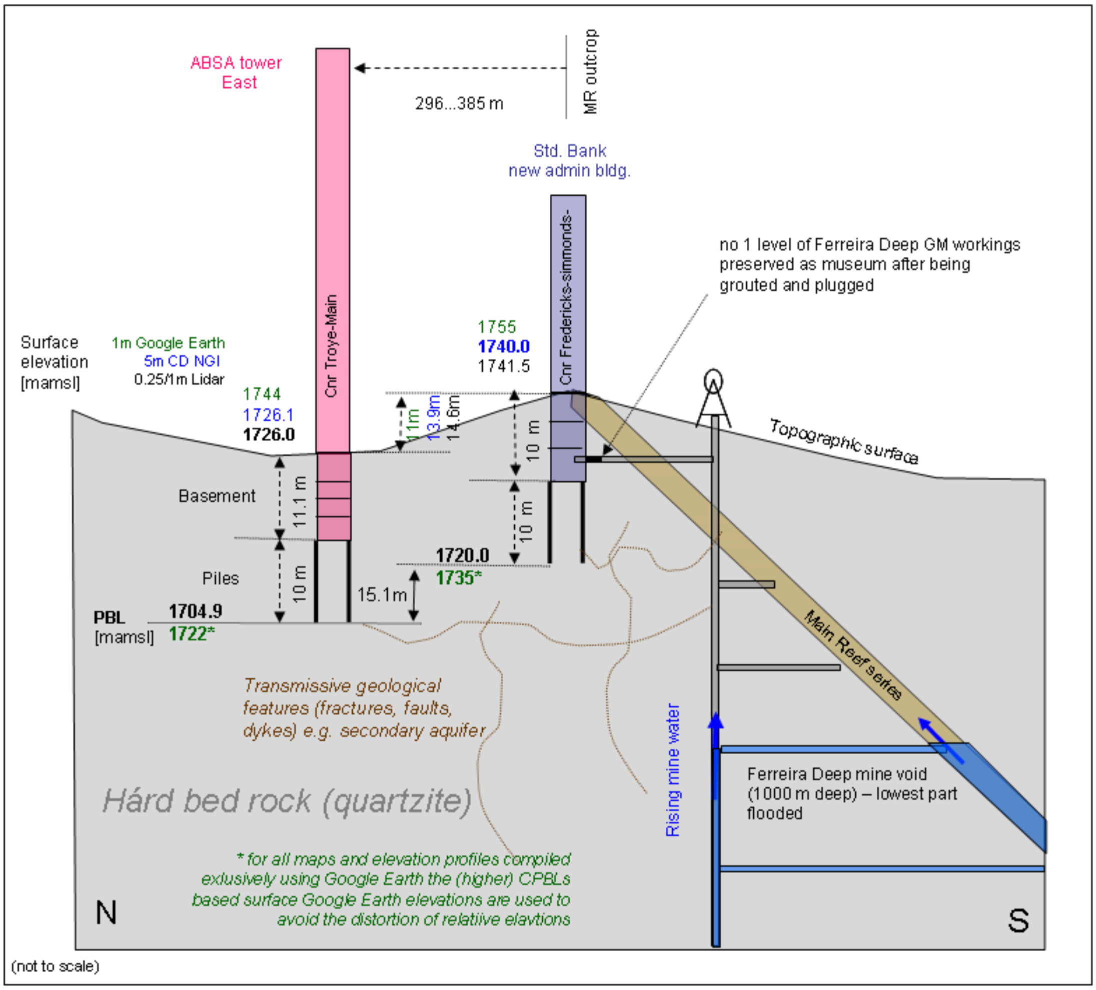

Fig. 8.13: Schematic depiction of the surface elevation [mamsl], the basement depths and the pile depths for the ABSA tower East and the new admin building of Standard Bank as the two key buildings most exposed to the risk of being flooded by rising mine water.

In order to be conservative in assessing the flooding risk the lowest value (lidar) for the lowest PBL of all key buildings (ABSA tower East) is used as critical PBL additionally rounded to the lower full meter for adding an extra safety margin. Thus the critical pile base level (CPBL) is set at **1704 mamsl** even though the PBL of the most exposed building (Standard Bank) is located more than 15 m above this level.

8.4 Relief-related surface elevations

In order to assess the flooding risk the final mine water level will pose to the underground structures of the identified bank buildings, their position in the topographic relief of the area in general and in relation to the entrance/ outflow points of the flooded mine void more specifically, needs to be analysed.

Of particular interest with respect to the risk assessment are the elevations of the mine void entrance across the 44-km long E-W running strike of the outcropping Main Reef bands, as the latter are closest to the key buildings in question. Furthermore, low lying areas in the natural topography (relief) such as stream valleys, drainage lines, wetland and floodplain areas etc. which may act as natural outflow points are also of interest.

Since a high-confidence relief analysis cannot be achieved through using spot heights only larger data sets covering the whole area of interest had to be used. Based on the findings discussed under data accuracy Google Earth provide reliable data especially for the largely open (i.e. not built-up) mining strip running parallel to the MR outcrop in the south.

In contrast to GIS based topographic contours and lidar data elevation data from Google Earth (other than the original SRTM set) cannot directly be imported into GIS somewhat limiting the analytical possibilities of data interpretation. However, inbuilt functions into GE, such as the generation of elevation profiles along selected transects and 3D relief visualisation, provided sufficient means to conduct meaningful data analyses.

8.4.1 Mine void entrance level

As initial mining closely followed the outcropping reef bands of the Main Reef series across the entire Central Rand it is reasonable to assume that the surface elevation of the Main Reef outcrop represents the entrance level to the mine void. However, in places the actual surface mining may have been somewhat deviating from this line generally affecting a 0.1–1 km wide area to the south of the MR outcrop line. Since the topography is generally N-S dipping much of the actual mine void entrance may indeed be lower than indicated by this profile.

The outcrop of the Main Reef was imported into Google Earth from scanned and GIS-referenced maps of various scales and sources as were the mine lease area boundaries of the different mines. Apart from inherent inaccuracies of the various original maps (between which significant deviations occur) the scanning and subsequent import into GIS and later Google Earth has the potential to further add additional inaccuracies. In order to prevent large deviations the MR outcrop was also located by following the Main Reef Road through JHB.

The E-W surface elevation profile along the MR outcrop is shown in Fig. 8.14.

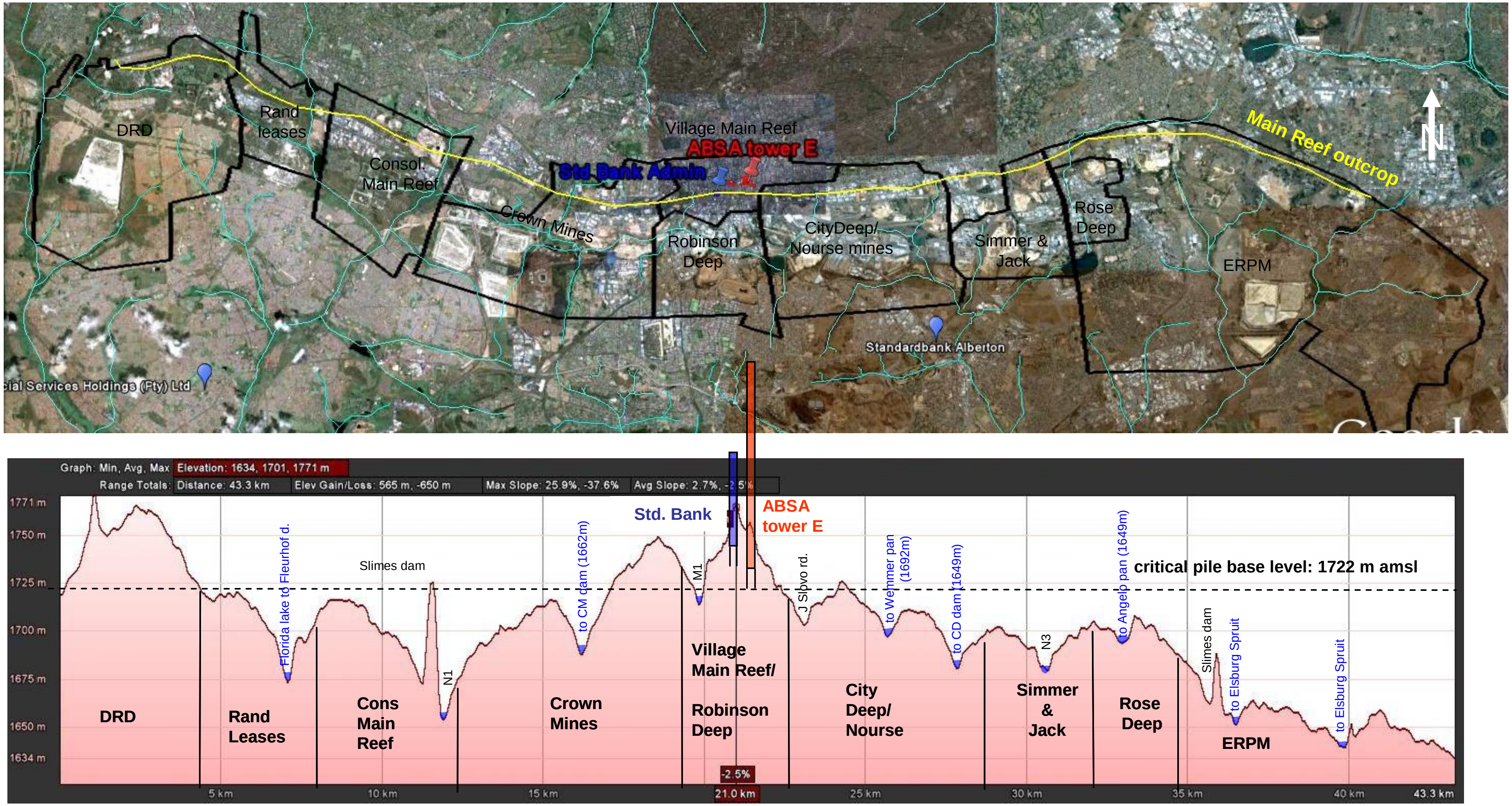

Fig. 8.14: Lower part: E-W elevation profile for the outcrop of the MR with Std. Bank and ABSA tower E projected onto the outcrop line (in reality only Std. Bank falls onto the outcrop while ABSA tower E is some 300 m to the N). The PBL is projected across the entire Central Rand indicating that the overwhelming majority of the Central Rand is lying below the critical level of 1722* mamsl (PBL). Blue fillings indicate river valleys potentially acting as outflow points. The upper part of the figure depicts a plane view of the study area indicating the different mine lease areas, the two bank buildings and the outcrop of the Main Reef. (Pile length acc. to scale, building's intersection with surface and building heights approximated) * see text below for reasons not using 1704 mamsl

As indicated earlier for non-covered areas outside the CBD, Google Earth provided the most reliable elevation data fitting best to the surveyed shaft elevations. However, owing to possible effects of buildings on the radar based determination of elevation in the build up area of JHB, the CPBL was determined using lidar data, resulting in a CPBL of 1704 mamsl. Since the Google profile largely displays the open, non-built up area along the mining belt, the elevation profile is regarded as correct not only of all elevations relative to each other but also in terms of the absolute altitude. However, the surface elevations for the two key building of Std. Bank and Absa tower E determined by lidar data (1740 and 1726 mamsl respectively) are in Google Earth some 27 m and 8 m respectively higher. If the lidar based CPBL (1704 mamsl) would now be applied to the Google Earth profile, it would result in unrealistically deep basement structures protruding into the bedrock by >30m. Since it is known that the basement and its supporting piles extent only ca. 20m below surface, an application of the Lidar based PBL to the Google Earth based profile would result in unrealistic relative positions between surface and PBLs. Since the main purpose of the profile is to assess the relative position between the natural relief and the pile base level this would skew the picture. Therefore the basement depths of the ABSA tower E and the STD. Bank (21.1 mbs and 20 mbs respectively) are subtracted from the surface elevation as displayed in Google Earth, allowing for a realistic assessment of the resulting height difference between the lowest piles and the different parts of the natural relief. This results in a CPBL, applicable to this profile only, of 1722 mamsl.

The E-W profile in Google Earth indicates that the surface elevation along the main reef outcrop varies by more than 130 m ranging from 1620 mamsl in the east to 1755 mamsl in the west. For the 5m CDSM data this range is even larger (150 m: 1615 – 1765 mamsl). This variation across the Central Rand needs to be considered when interpreting mine water table levels given for the entire Central Basin in m ‘below surface’ commonly without stipulating to what part of the some 44-km-long surface it refers to.

In order to circumnavigate this problem relatively early in mining the concept of the ‘datum’ was introduced as a common benchmark for all elevation references which are of crucial importance in deep level mining. ‘Datum’ refers to a virtual line located somewhat above the topographic surface across the whole CR which was originally set at an elevation of 6000 ft above mean sea level (= 1828.82 mamsl). Compared to the use of elevation data given in metres above mean sea level (mamsl) the reference to a common datum line has the advantage that no negative values occur, as is the case for deep level gold mines operating *below* – and no longer *above* - the ’mean sea level’.

The E-W profile illustrates that the CBD of JHB is located at the second most prominent relief position across the entire Central Rand (just 12 m lower than the highest lying area near Roodepoort). This generally reduces the risk of flooding as most of the reminder of the relief and thus the mine void entrance level that defines the FMWL is at a lower altitude. Of the

43.3-km-long profile only 7.1 km displays elevations above 1722 mamsl. This implies that close to 84% of the mine void entrance lie below the critical pile base level.

Furthermore, the profile illustrates that the MR runs across a number of low lying relief areas which may act as natural decant points. Where the MR outcrop crosses such valleys one needs to consider a commonly overlooked cartographic effect which results in the over-/underestimation of map-based determinations of the elevation outcrops depending on the direction of their underground dip. The elevation error associated with this effect in the study area and a method to correct for it are discussed below.

8.4.2 Correcting elevation data for the S-dipping MR outcrop (valley effect)

While the E-W profile indicated the surface elevation of the outcropping MR reef series it is not entirely accurate. This applies to the location of the reef outcrop in valleys where erosion lowered the land surface. Since the reefs dip steeply at an average angle of 45° to the south surface erosion e.g. by streams will cut into the reefs and remove the upper in plan projection further north lying part of it. With this part being no longer present in the resulting valleys the cut reef is now intersecting the surface further south compared to the un-eroded areas on both sides of the valley. Since the topography in the study area also dips southward the elevation of the reef is now lower compared to the non-eroded reef outcrop. Since none of the consulted maps takes this effect into account, the degree of the southward shift needs to be known in order to correct the course and the associated elevation of the outcrop accordingly (Fig. 8.15).

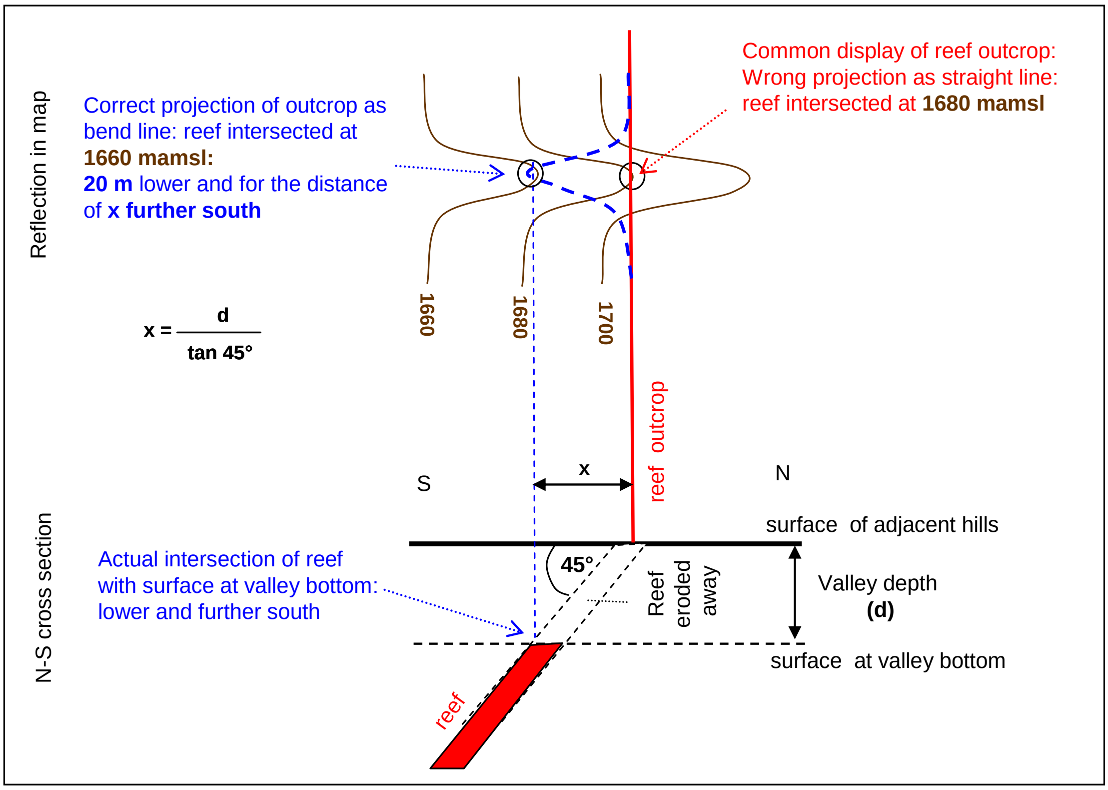

Fig. 8.15: Sketch illustrating the distortion of the actual elevation and location of dipping outcropping reefs in most maps by ignoring the fact that reefs in valley are intersecting the surface at lower elevations and thus further south than in adjacent – non-valley areas

Applying triangle geometry the southward shift can be calculated using the tangents of the dipping angle of 45° and the depth of the valley.

For the N1 valley in the east of the profile this effect resulted in a 105 m lower elevation of the reef outcrop (mine void entrance level) assuming a 45° reef dip adding a further safety margin for the underground bank structures.

8.4.3 Natural decant points of the relief

Particular low lying areas are valleys of N-S running streams and other natural drainage lines which, according to a number of sources are important sources of water recharging the underlying mine void. If water from these valleys can flow into the mine void it is principally possible that it can flow in the inverse direction too, i.e. from the filled mine void into the stream valley. This would render topographically low-lying areas which are hydraulically connected to the mine void future outflow points where mine water would flow to surface as soon as the mine water table rises above the stream level in the valleys. If the hydraulic conductivity of the zone/features connecting the mine void with the stream (e.g. fractures, dykes, faults, fill material etc.) is high enough to accommodate the total decant volume (which needs to be estimated to assess this possibility) then any risk of higher-lying areas being flooded by mine water can be excluded. In the above E-W profile this applies to a number of stream valleys including the following:

Tab. 8.10: Elevations of stream valleys as potential decant points

Stream valley (elevation of receiving water body)	**Distance from Std. Bank** [km]	**Elevation** [mamsl]	**m below PBL***
N-S stream from Florida lake to Fleurhof dam	13.94	1673	49
N-S stream on Cons- Main Reef (N1)	9.1	1653	69
Stream to Crown Mines dam (1662 mamsl)	4.8	1688	34
N-S stream along M1	1.1	1713	9
N-S stream to Wemmer pan (1692 mamsl)	4.7	1698	24
N-S stream to City Deep dam (1649 mamsl)	6.8	1681	41
N-S stream along N3	9.6	1679	43
N-S stream to Angelo pan (1649 mamsl)	12.0	1694	28
W-tributary to Elsburg spruit	15.5	1651	71
E-tributary to Elsburg spruit	18.8	1639	83

* for this Google profile only: 1722 mamsl

With elevations ranging from 28 m to 83 m below the lowest piles of ABSA Bank all of the water filled drainage lines listed in Tab. 8.10 are low enough to safely prevent any risk of basement flooding provided they are hydraulically connected to the mine void and that the cumulative capacity of all hydraulic connections is large enough to accommodate the decant volume. Should the decant volume be larger than what can be accommodated by the natural outflow points hydraulic heads in higher lying entrance areas of the mine void which still receive ingress water could form raising the final mine water level above these points.

Since N-S running streams crossing the disturbed outcrop area were frequently identified as major ingress sources (i.e. losing water to the underlying mine void) it can be assumed that these streams will in principle also be able to act as decant points.

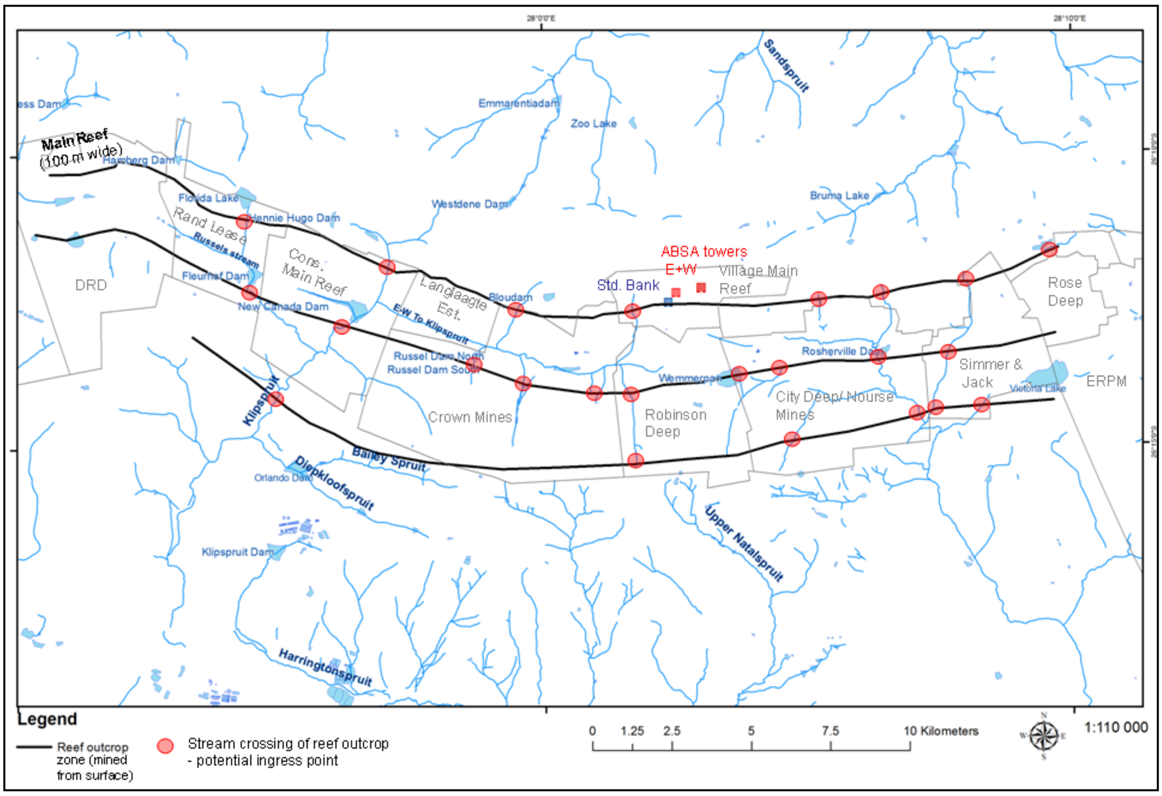

Fig. 8.16: Stream crossings of reef outcrop zones as potential ingress points

Recharging the empty mine through hydraulic connections such as weathered dykes, fissures, fractures, faults, unconsolidated sediments etc. will render these streams decant points as soon as the water table in the flooding mine void rises above the stream level. With stream loss being estimated of contributing up to 8.7 Ml/d to the mine ingress (Van Biljon, 1999; cited in Boer et. al., 2004; other estimates are lower), streams could accommodate approximately a third of the total decant volume. This flow rate may however rise significantly once artesian pressure builds up below the streams due to the formation of hydraulic heads in higher lying parts of the mine void. Since the 8.7 Ml/d were only driven by a relatively small water column (stream depth, rarely exceeding 0.5 m) it is likely that much more water could flow through the hydraulic links connecting the void to the stream if this water is under (artesian) pressure. Thus it is not unreasonable to assume that all stream valleys below the critical PBL together, are sufficient to act as a safety valve (i.e. decant points) preventing the final mine water level from reaching the CPBL. Since artesian pressure would affect the lowest-lying streams first, the preceding rise in mine water level (an hydraulic head of an est. 2–5 m, i.e. a water pressure 5 to 10 larger than exerted by normal streams, should be sufficient to significantly increase flow rates into streams) would not have to be added to the elevation of the highest lying stream valley, as outflow in lower lying streams are likely to increase well before the higher-lying streams are reached. Thus the lowest elevation difference between the critical PBL and the highest lying stream valley of 9 m would not have to be reduced (Tab. 8.10).

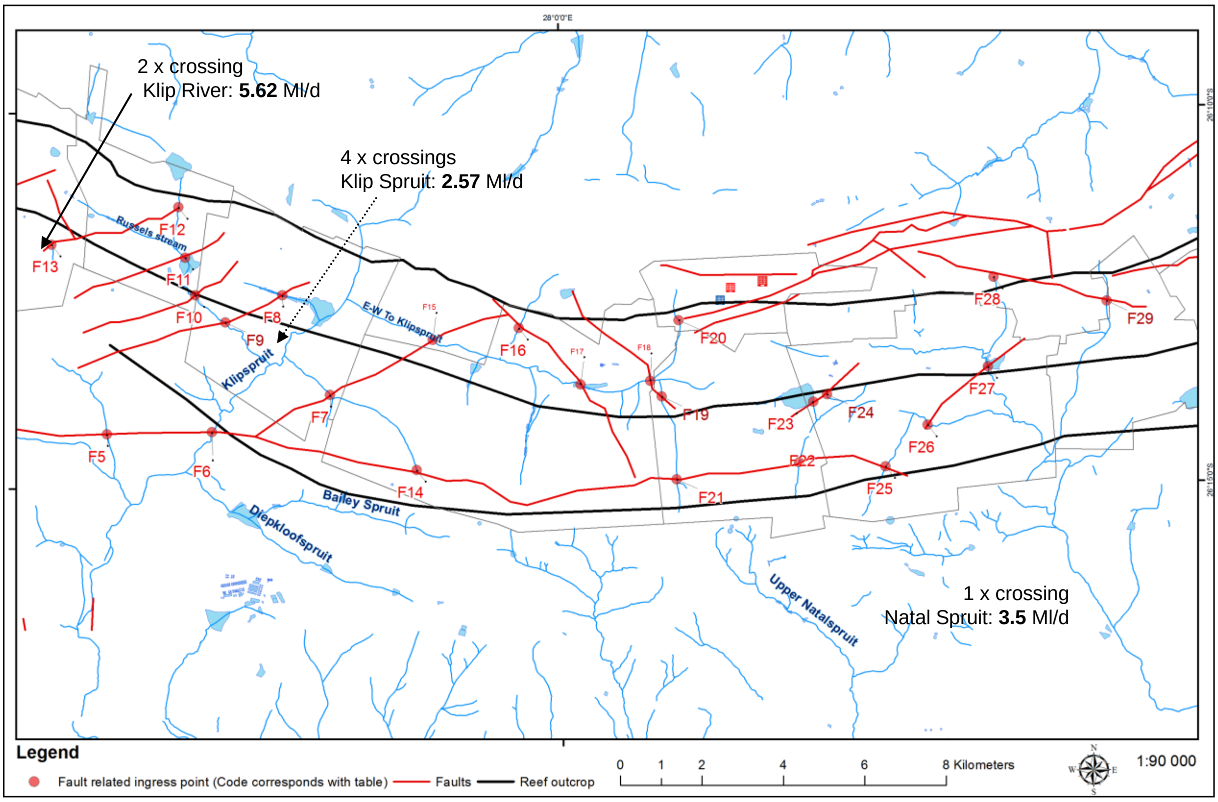

Fig. 8.17: Streams crossing fault lines in the mining belt where water may be lost to the underlying mine void (location of crossings and dykes based on Van Biljon & Walker 2001)

According to van Biljon and Walker (2001) the total ingress derived from streams losing water to crossed faults is 11.96 Ml/d of which more than two thirds (68%) occur in the western part of the CR at DRD and RL mines.

Ingress areas associated with streams losing water into the mine void while crossing dykes are depicted in Fig. 8.18.

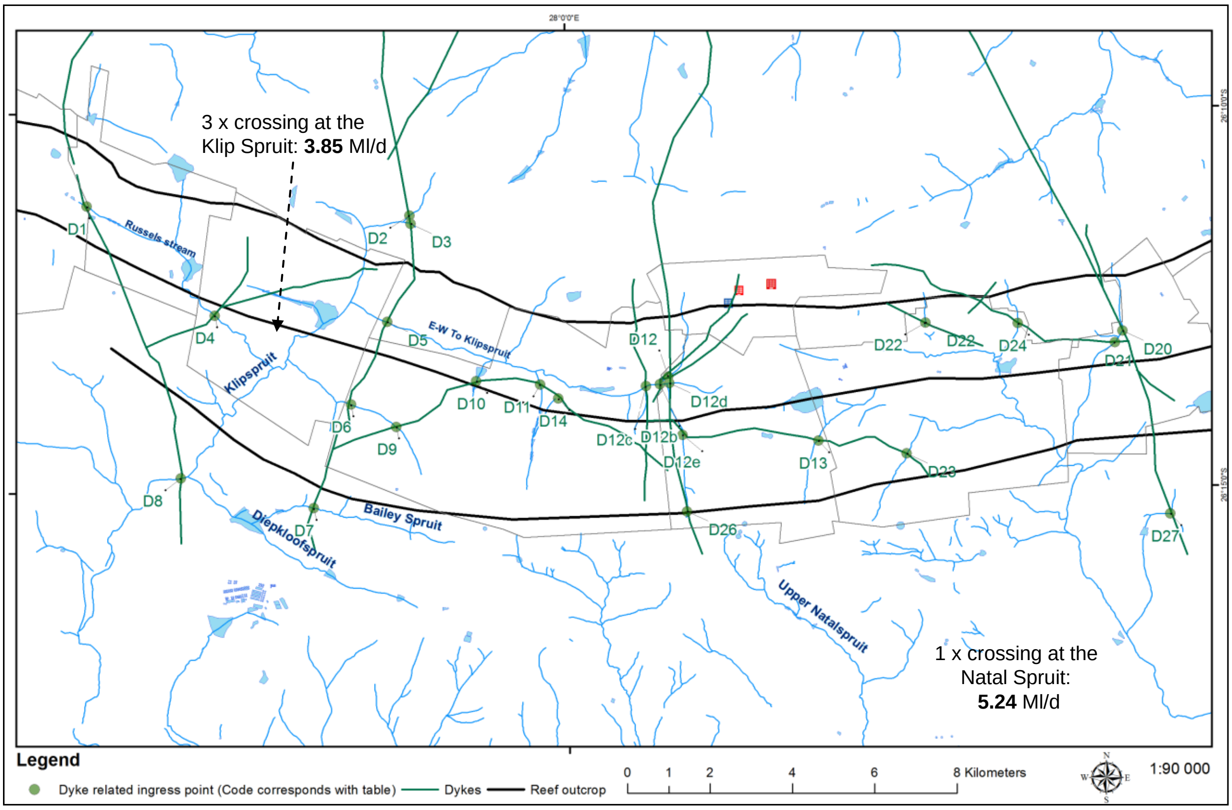

Fig**.** 8.18: Sites where streams lose water into the mine void via dykes (based on maps in van Biljon & Walker 2001)

In total some 9.09 Ml/d of ingress is estimated by van Biljon and Walker (2001) to originate from streams crossing transmissive weathered dykes that are connected to the underlying mine void.

Stream loss via faults and dykes as estimated by van Biljon and Walker (2001) would account for 21.05 Ml/d of which more than half (57%; 12.04 Ml/d) is generated in the western section of the CR at DRD-CMR.

As discussed earlier all streams which are currently recharging the mine void can principally also act as future decant points. To identify streams which may be affected by mine water diffusely seeping out of the flooded void along dykes and faults into adjacent streams the elevation of all sites where streams cross dykes and faults are listed in Tab. 8.11 and compared against the probable decant level of 1614 mamsl (Cinderella W) (the determination of the decant level this explained later in the report).

Tab. 8.11: Elevations for 62 ingress points [mamsl] determined by 3 different data sets where streams cross the 3 x mined outcrop zones associated with the Main, Bird and Kimberley Reef series losing water to the mine void via faults (F) and dykes (D). The data are sorted in ascending order of the difference between the critical pile base level (CPBL) determined in Google (1722 mamsl) and the Google Earth elevation of the ingress point. Negative values indicate that the ingress points lies below the CPBL (blue shaded: streams which may receive mine water seeping from the flooded mine void)

Code	Ingress pathway	Coordinates		Mine lease area	Elevation according to:			Diff. stream - decant level [m] (decant level: Cinderalla West #: 1614 mamsl) (0 & negative: possible decant)		
	F - fault *D - dyke*	latitude	longitude		GE [mamsl]	LIDAR [mamsl]	CDNGI [mamsl]	GE	Lidar	CDNGI
D1	D	26° 11' 13.21" S	27° 53' 01.35" E	DRD	1703	1694,49	1695,00	89	80,49	81,00
D10	D	26° 13' 33.85" S	27° 58' 40.64" E	CM	1672	1665,35	1664,92	58	51,35	50,92
D11	D	26° 13' 36.84" S	27° 59' 36.35" E	CM	1675	1669,43	1671,34	61	55,43	57,34
D12	D	26° 13' 31.34" S	28° 01' 28.4" E	Rob D	1686	1681,24	1683,20	72	67,24	69,20
D12b	D	26° 13' 37.42" S	28° 01' 21.16" E	CM	1686	1682,44	1680,98	72	68,44	66,98
D12c	D	26° 13' 38.68" S	28° 01' 08.92" E	CM	1695	1689,77	1683,35	81	75,77	69,35
D12d	D	26° 13' 36.51" S	28° 01' 29.6" E	Rob D	1688	1682,48	1683,02	74	68,48	69,02
D12e	D	26° 14' 17.51" S	28° 01' 40.7" E	Rob D	1701	1691,88	1694,25	87	77,88	80,25
D13	D	26° 14' 22.74" S	28° 03' 39.37" E	Rob D	1706	1698,46	1697,81	92	84,46	83,81
D14	D	26° 13' 48.08" S	27° 59' 52.1" E	CM	1689	1677,09	1680,86	75	63,09	66,86
D2	D	26° 11' 22.48" S	27° 57' 43.29" E	Outside Minelease areas in sources	1666	1660,93	1662,88	52	46,93	48,88
D20	D	26° 12' 57.52" S	28° 08' 06.07" E	S&J	1657	1649,16	1649,70	43	35,16	35,70
D21	D	26° 13' 06.63" S	28° 07' 59.33" E	S&J	1650	1646,13	1642,42	36	32,13	28,42
D22	D	26° 12' 50.01" S	28° 05' 13.78" E	CD	1683	1675,73	1669,88	69	61,73	55,88
D23	D	26° 14' 33.52" S	28° 04' 56.55" E	CD	1684	1673,74	1674,27	70	59,74	60,27
D24	D	26° 12' 51.03" S	28° 06' 34.62" E	CD	1667	1656,42	1652,97	53	42,42	38,97
D26	D	26° 15' 18.07" S	28° 01' 44.19" E	Rob D	1735	1724,84	1724,09	121	110,84	110,09
D27	D	26° 15' 22.33" S	28° 08' 46.91" E	Outside Minelease areas in sources	1611	1602,27	1603,87	-3	-11,73	-10,13
D3	D	26° 11' 28.84" S	27° 57' 44.37" E	Outside Minelease areas in sources	1663	1654,50	1658,90	49	40,50	44,90
D4	D	26° 12' 40.31" S	27° 54' 52.32" E	CMR	1647	1638,13	1639,89	33	24,13	25,89
D5	D	26° 12' 46.35" S	27° 57' 23.08" E	Outside Minelease areas in sources	1650	1645,05	1643,29	36	31,05	29,29
D6	D	26° 13' 51.48" S	27° 56' 50.87" E	CM	1657	1652,07	1653,11	43	38,07	39,11
D7	D	26° 15' 13.22" S	27° 56' 17.67" E	Outside Minelease areas in sources	1632	1624,14	1624,40	18	10,14	10,40
D8	D	26° 14' 48.88" S	27° 54' 21.69" E	Outside Minelease areas in sources	1590	1587,06	1583,12	-24	-26,94	-30,88
D9	D	26° 14' 09.48" S	27° 57' 30.49" E	CM	1678	1673,02	1674,05	64	59,02	60,05
F1	F	26° 08' 04.43" S	27° 49' 26.39" E	Outside Minelease areas in sources	1734	1725,33	1724,72	120	111,33	110,72
F10	F	26° 12' 26.37" S	27° 54' 37.34" E	RLC	1652	1643,68	1644,54	38	29,68	30,54
F11	F	26° 11' 56.23" S	27° 54' 28.62" E	RLC	1664	1660,18	1660,94	50	46,18	46,94
F12	F	26° 11' 15.99" S	27° 54' 23.14" E	RLC	1671	1664,95	1665,53	57	50,95	51,53
F13	F	26° 11' 44.9" S	27° 52' 30.56" E	DRD	1707	1702,00	1703,18	93	88,00	89,18
F14	F	26° 14' 47.52" S	27° 57' 52.03" E	CM	1703	1697,49	1694,08	89	83,49	80,08
F15	F	26° 13' 03.72" S	27° 58' 07.13" E	Outside Minelease areas in sources	1652	1648,37	1644,80	38	34,37	30,80
F16	F	26° 12' 54.71" S	27° 59' 23.35" E	CM	1684	1671,71	1672,98	70	57,71	58,98
F17	F	26° 13' 40.08" S	28° 00' 17.46" E	CM	1679	1672,64	1690,03	65	58,64	76,03
F18	F	26° 13' 37.48" S	28° 01' 19.19" E	CM	1687	1682,05	1680,30	73	68,05	66,30
F19	F	26° 13' 50.26" S	28° 01' 29.28" E	Rob D	1691	1683,38	1684,85	77	69,38	70,85
F2	F	26° 10' 30.1" S	27° 49' 06.11" E	Outside Minelease areas in sources	1665	1658,93	1659,14	51	44,93	45,14
F20	F	26° 12' 49.28" S	28° 01' 44.82" E	VMR	1721	1721,32	1717,34	107	107,32	103,34
F21	F	26° 14' 56.57" S	28° 01' 42.12" E	Rob D	1717	1709,58	1710,00	103	95,58	96,00
F22	F	26° 14' 43.46" S	28° 03' 30.58" E	Rob D	1716	1707,56	1709,47	102	93,56	95,47
F23	F	26° 13' 55.29" S	28° 03' 43.38" E	CD	1703	1690,84	1690,39	89	76,84	76,39
F24	F	26° 13' 49.39" S	28° 03' 55.53" E	CD	1695	1685,51	1702,21	81	71,51	88,21
F25	F	26° 14' 47.29" S	28° 04' 46.98" E	CD	1708	1685,64	1690,14	94	71,64	76,14
F26	F	26° 14' 14.22" S	28° 05' 24.22" E	CD	1660	1648,38	1649,75	46	34,38	35,75
F27	F	26° 13' 28.02" S	28° 06' 18.11" E	CD	1648	1646,73	1645,36	34	32,73	31,36
F28	F	26° 12' 16.33" S	28° 06' 23.49" E	Outside Minelease areas in sources	1694	1688,00	1688,18	80	74,00	74,18
F29	F	26° 12' 36.01" S	28° 08' 03.38" E	unknown	1671	1661,31	1660,98	57	47,31	46,98
F2b	F	26° 10' 14.38" S	27° 49' 03.97" E	Outside Minelease areas in sources	1668	1663,23	1661,86	54	49,23	47,86
F2c	F	26° 10' 47.86" S	27° 49' 28.19" E	DRD	1661	1652,49	1654,55	47	38,49	40,55
F2d	F	26° 10' 39.48" S	27° 49' 38.44" E	DRD	1666	1655,53	1657,57	52	41,53	43,57
F2e	F	26° 11' 01.31" S	27° 49' 06.59" E	Outside Minelease areas in sources	1652	1645,57	1644,91	38	31,57	30,91
F3	F	26° 13' 14.76" S	27° 48' 41.31" E	Outside Minelease areas in sources	1614	1608,83	1608,89	0	-5,17	-5,11
F37	F	26° 08' 01.75" S	27° 51' 02.6" E	Outside Minelease areas in sources	1738	1729,16	1729,58	124	115,16	115,58
F38	F	26° 09' 05.26" S	27° 54' 00.92" E	Outside Minelease areas in sources	1735	1728,69	1729,68	121	114,69	115,68
F39	F	26° 09' 36.03" S	27° 54' 24.18" E	Outside Minelease areas in sources	1705	1696,01	1699,21	91	82,01	85,21
F4	F	26° 13' 33.91" S	27° 50' 05.16" E	DRD	1635	1628,53	1629,49	21	14,53	15,49
F47	F	26° 11' 04.97" S	27° 57' 57" E	Outside Minelease areas in sources	1684	0,00	1675,62	70		61,62
F5	F	26° 14' 16.87" S	27° 53' 18.26" E	Outside Minelease areas in sources	1601	1595,71	1595,50	-13	-18,29	-18,50
F6	F	26° 14' 15.94" S	27° 54' 51.17" E	Outside Minelease areas in sources	1594	1616,00	1589,54	-20	2,00	-24,46
F7	F	26° 13' 47.02" S	27° 56' 35.86" E	CMR	1648	1643,31	1643,45	34	29,31	29,45
F8	F	26° 12' 26.93" S	27° 55' 54.37" E	CMR	1659	1643,79	1641,41	45	29,79	27,41
F9	F	26° 12' 48.25" S	27° 55' 03.78" E	CMR	1642	1634,55	1634,84	28	20,55	20,84
				n	62	62	62	62	61	62
				av.	**1674,11**	**1640,21**	**1667,36**	**60,11**	**53,10**	**53,36**
				median	***1673,50***	***1665,15***	***1667,70***	***59,50***	***51,35***	***53,70***
				min.	1590,00	0,00	1583,12	-24,00	-26,94	-30,88
				max.	1738,00	1729,16	1729,68	124,00	115,16	115,68

The blue shaded rows in Tab. 8.11 indicate stream crossing with faults and dykes which are located below the decant level rendering them prone to be affected by water diffusely seeping from the flooded mine void along the transmissive dykes and faults. It appears that only 5 x of the 62 x stream crossings are potentially affected. The geographical location of these points is depicted in Fig. 8.19.

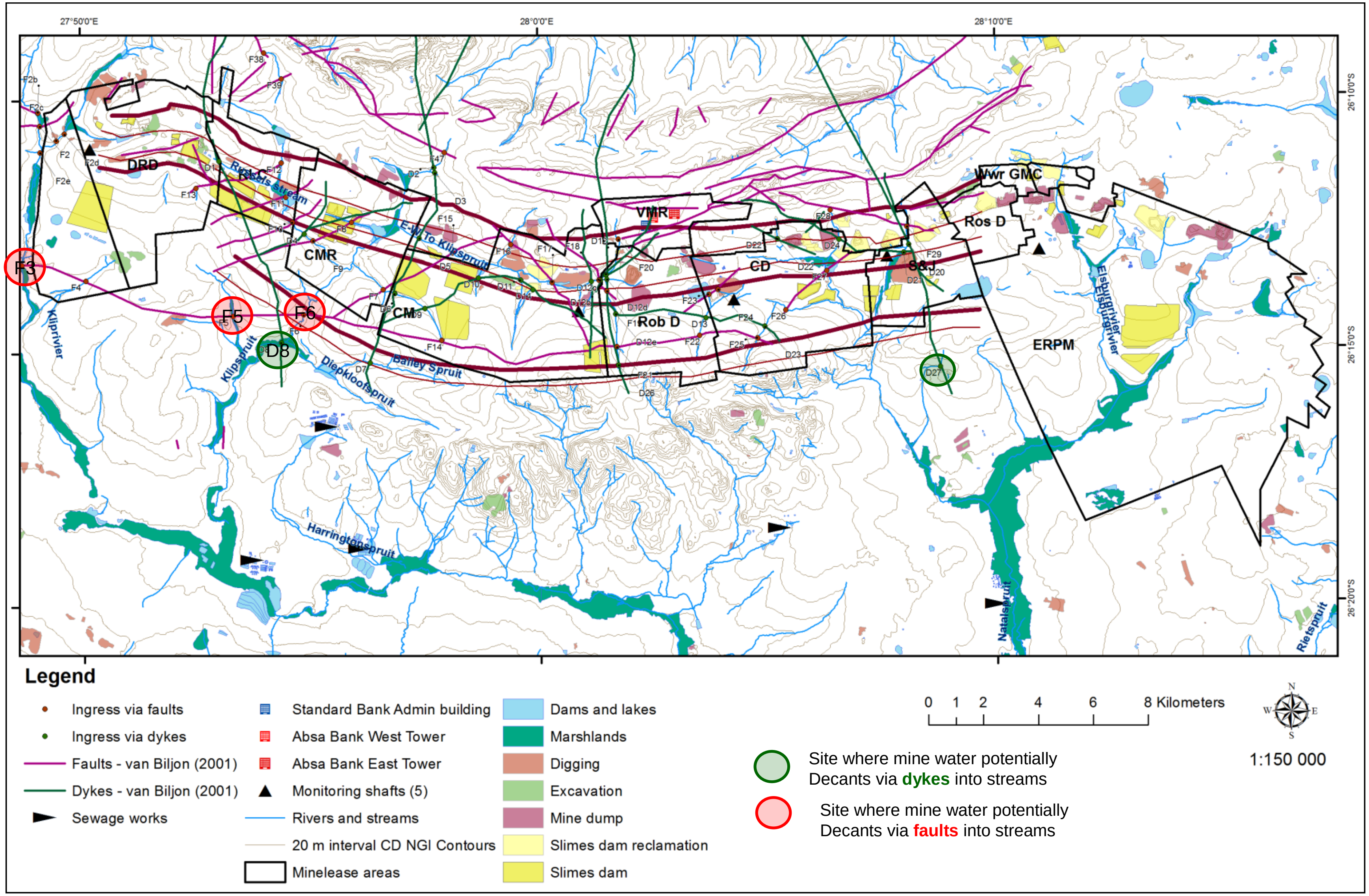

Fig. 8.19: Stream crossings of dykes and faults located below the decant level of 1614 mamsl as possible additional decant points for mine water from the Central Basin once it is completely flooded

Generally it appears all streams potentially affected by mine water seeping through transmissive dykes and faults are located south of the mining belt (Fig. 8.19). Since these streams partly run through densely populated urban areas measures should be taken to prevent possibly polluted mine water impacting on downstream users.

8.4.4 Role of hydraulic heads owing to topographic gradients

Water levels in aquifers are usually not entirely flat but often reflect to some degree the topographic surface. This could principally result in somewhat elevated mine water levels in higher lying areas.

However the ability of groundwater to mirror the overlying surface relief is limited. For relatively flat dolomitic karst aquifers in the Far West Rand Wolmarans (1984) found an average gradient of the groundwater table of 1:1250. In many respects these karst aquifers are comparable to the mine void system since both consists essentially of a network of pipes and conduits of different diameter and hydraulic conductivity ranging from free flowing in open karst channels (for the mine: shafts and tunnels) to very slow flow through collapsed channels or conduits filled with äolian sands, rubble, wad and chert (the equivalent to collapsed underground workings and the back-filled outcrop area of the mine void).

Applying a somewhat steeper gradient of 1:1000 to be conservative results in a maximum hydraulic head forming between the banks in the CBD and the stream valley near the N1 (some 9.1 km away) of 9.1 m. This would raise the FMWL to 1662.1 mamsl (from 1653 mamsl; Tab. 8.11) which still is close to 60 m below the critical basement flooding level (Tab. 8.11.).

For the lowest lying stream valley in the east (E-tributary to Elsburg spruit) the results are similar. Located some 18.8 km to the east of the bank buildings the hydraulic head would be some 19 m above the decant level of 1639 mamsl remaining some 64 m below the critical PBL (Tab. 8.11).

It can therefore be concluded that the natural decant points provided by the natural topography of the area, will most likely be sufficient to accommodate the expected flow rate of overflowing mine water and thus keep the final MWL well below even the deepest basement structures of the two banks.

In addition to the above discussed Google Earth profiles the elevation for some potential natural decant points on ERPM property, as lowest lying mine lease area of the entire Central Rand, have been surveyed by DRD. This includes mainly (although not exclusively) water bodies such as dams, lakes, pans and streams (Tab. 8.12).

Tab. 8.12: Surveyed elevations of potential decant point at the ERPM lease area (data: DRD, 2001)

The following Table 1 contains the GPS co-ordinates of all water monitoring points and the attached map (Figure 1) shows the spatial positioning of these points.

Table 1: ERPM - Water sampling points GPS co-ordinates

SITE CODE	SITE NAME	2.1.1.1.1.1 POSITION		ALTITUDE (m)
S1	ELSBURG PUMP STATION	S 26° 15.606'	E 28° 13.298'	1587
S2	WALSCHESSPRUIT	S 26° 13.155'	E 28° 12.535'	1629
S5	VICTORIA LAKE OVERFLOW	S 26° 13.949'	E 28° 11.802'	1629
S6	DELMORE DAM OVERFLOW	S 26° 12.831'	E 28° 11.699'	1644
S7	BOKSBURG LAKE INFLOW	S 26° 13.100'	E 28° 15.174'	1625
S8	ANGELO PAN OVERFLOW	S 26° 13.077'	E 28° 13.193'	1641
S14	SAR DRAIN	S 26° 12.325'	E 28° 12.640'	1657
S15	HDS PLANT	S 26° 12.881'	E 28° 11.270'	1670
S15A	HDS OUTLET (ELSBURG DAM)	S 26° 12.909'	E 28° 11.732'	1640
S16	BOKSBURG LAKE OVERFLOW	S 26° 13.289'	E 28° 14.591'	1615
S17	CINDERELLA DAM OVERFLOW	S 26° 14.603'	E 28° 14.302'	1606
S18	DIXIE SPRUIT (HEIDELBURG ROAD)	S 26° 15.681'	E 28° 13.326'	1584
S19	ELSBURGSPRUIT	S 26° 14.584'	E 28° 12.395'	1596
S20	ELSBURGSPRUIT (OSBORN ROAD)	S 26° 16.565'	E 28° 12.109'	1569
S21	DIXIE SPRUIT (N17)	S 26° 15.224'	E 28° 13.751'	1586
S22	SUBWAY (N17)	S 26° 15.123'	E 28° 12.751'	1622
S23	SILT TRAP (DAM 2 N\E CORNER)	S 26° 14.205'	E 28° 14.035'	1622
S25	ELSBURG DAM OVERFLOW	S 26° 13.593'	E 28° 12.054'	1623
E1	BOKSBURG LAKE O/FLOW (MIDDEL ST)	S 26° 13.711'	E 28° 14.460'	1613
E2	CINDERELLA DAM (EAST INFLOW)	S 26° 14.401'	E 28° 14.806'	1611
E3	ELSBURG SPRUIT,(SWART STREET)	S 26° 14.177'	E 28° 12.310'	1609
E4	ELSBURG SPRUIT (N17)	S 26° 15.179'	E 28° 12.324'	1580
E5	F.E.V.SHAFT (PIPIT ROAD)	S 26° 16.475'	E 28° 13.151'	1579
E6	ELSBURG SPRUIT (join)	S 26° 16.138'	E 28° 12.583'	1569
E7	ELSBURG SPRUIT (HDS join spriut)	S 26° 12.982'	E 28° 11.749'	1625
E8	ELSBURG SPRUIT (railway)	S 26° 12.333'	E 28° 12.625'	1641
E9	ELSBURG SPRUIT (Delmore gardens)	S 26° 12.581'	E 28° 12.713'	1635

The above table indicates a total of 27 x potential outflow points of which 21 x can be classified as natural or semi-natural areas. The latter areas are at between 135 m (Elsburg spruit join and Elsburg spruit at Osborn Road both at 1569 mamsl) and 63 m (Angelo pan overflow t 1641 mamsl) below the CPBL (now the lidar level of 1704 mamsl has to be used as the above data are not derived from Google Earth but from a survey) adding a considerable safety margin in support of the above argument that decant via natural low lying areas is very probably sufficient to keep the final mine water level well below the critical elevation.

This will, of course, also apply to shafts as structures most directly connected to the mine void as long as they are still open or can be re-opened (where they have only been capped but not filled with impervious material to depth). This is discussed in a separate section below.

Apart from the E-W profile it is also important to understand the relative position of the banks to the mine void entrance especially as the Std. Bank building is located on top of the main reef outcrop and thus very close to the southward lying mining belt. This profile is again derived from Google Earth and discussed in the following section.

8.4.5 North-south profile (Google Earth)

Fig.8.20. Displays the N-S elevation profile from Linksfield ridge via ABSA tower E and Std Bank admin building to South Crest indicating some of the outcropping reefs and the critical PBL.

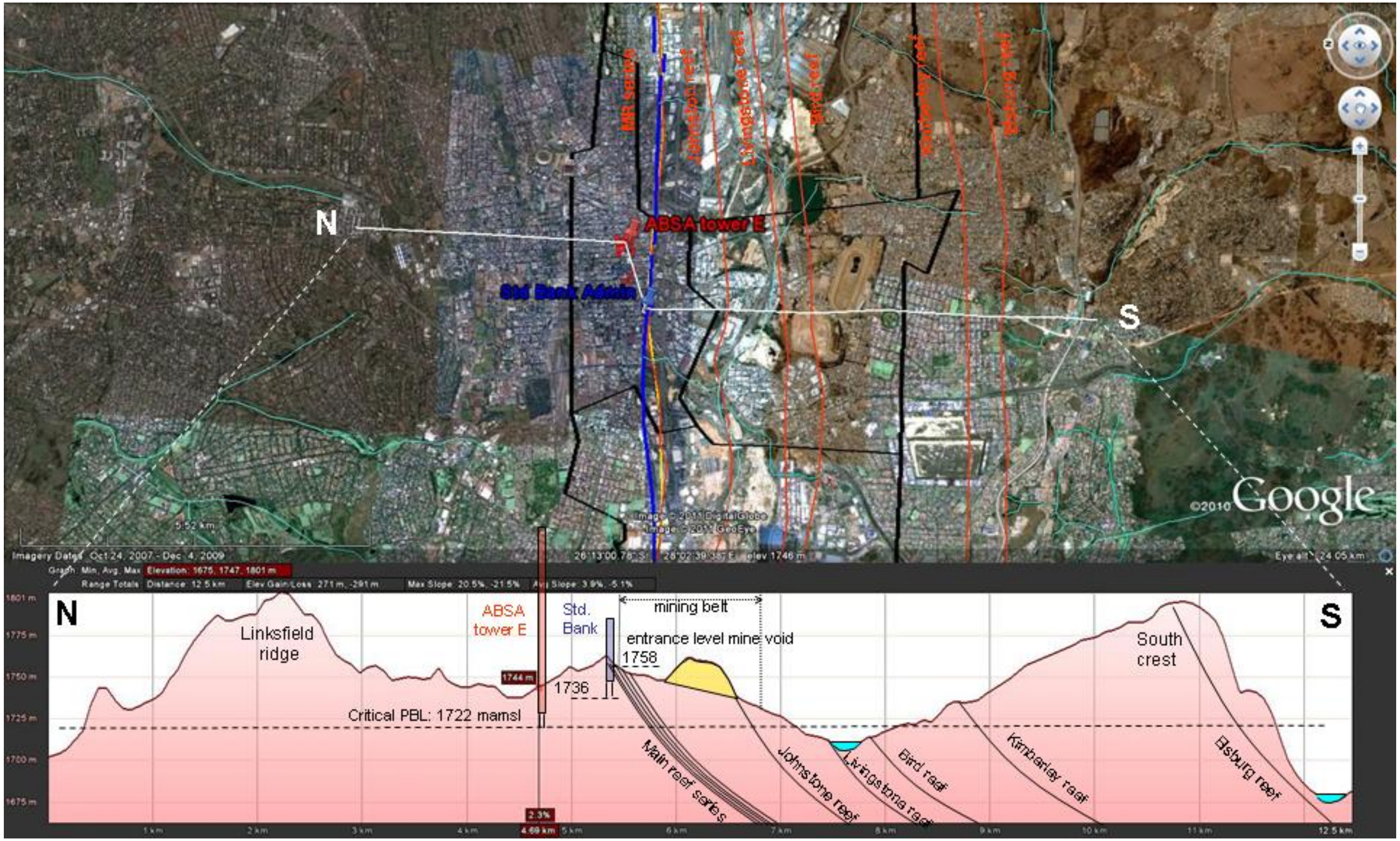

Fig. 8.20: N-S profile through the critical bank buildings based on GE elevations indicating the critical PBL in relation to the natural relief as well as underground features such as outcropping reefs, shafts and the MR outcrop zone mined from surface.

The N-S cross section indicates that the critical PBL (ABSA tower E) as well as the PBL for Std. Bank are both located below the outcropping main reef series. Having historically being mined from surface the outcrop is taken as a proxy to mark the entrance level of the underlying mine void. For this particular profile the void entrance level is 36 m above the critical PBL and still 22 m above the Std. Bank PBL suggesting a potential flooding risk. However, as shown in the previous section where the E-W profile of the mine entrance along the MR outcrop was analysed this risk can be excluded as low lying outflow areas in the natural relief are most probably able to keep the mine water level from reaching the surface in one of the highest lying parts of the Central Rand.

8.4.6 Impacts of slimes dams on groundwater flow and elevation

While this is true for the mine water level in the completely flooded void it was explored how far meso- and microscale man-made and natural relief features may pose a flooding risk after the mine void is completely filled. The underlying worst-case assumption is that prominent relief structures in the immediate vicinity to Std. Bank such as slimes dams and quartzite ridges may result in localised inversions of the general N-S direction of groundwater flow that could pose a basement flooding risks. The latter is assumed to result from the fact that a filled mine void may no longer accept N-flowing water such as tailings seepage, surface run off or natural groundwater allowing the former to recharge the fractured aquifer on which Std. Bank is build. In order to assess the probability of such scenario a large scale N-S profile was analysed (Fig. 8.21).

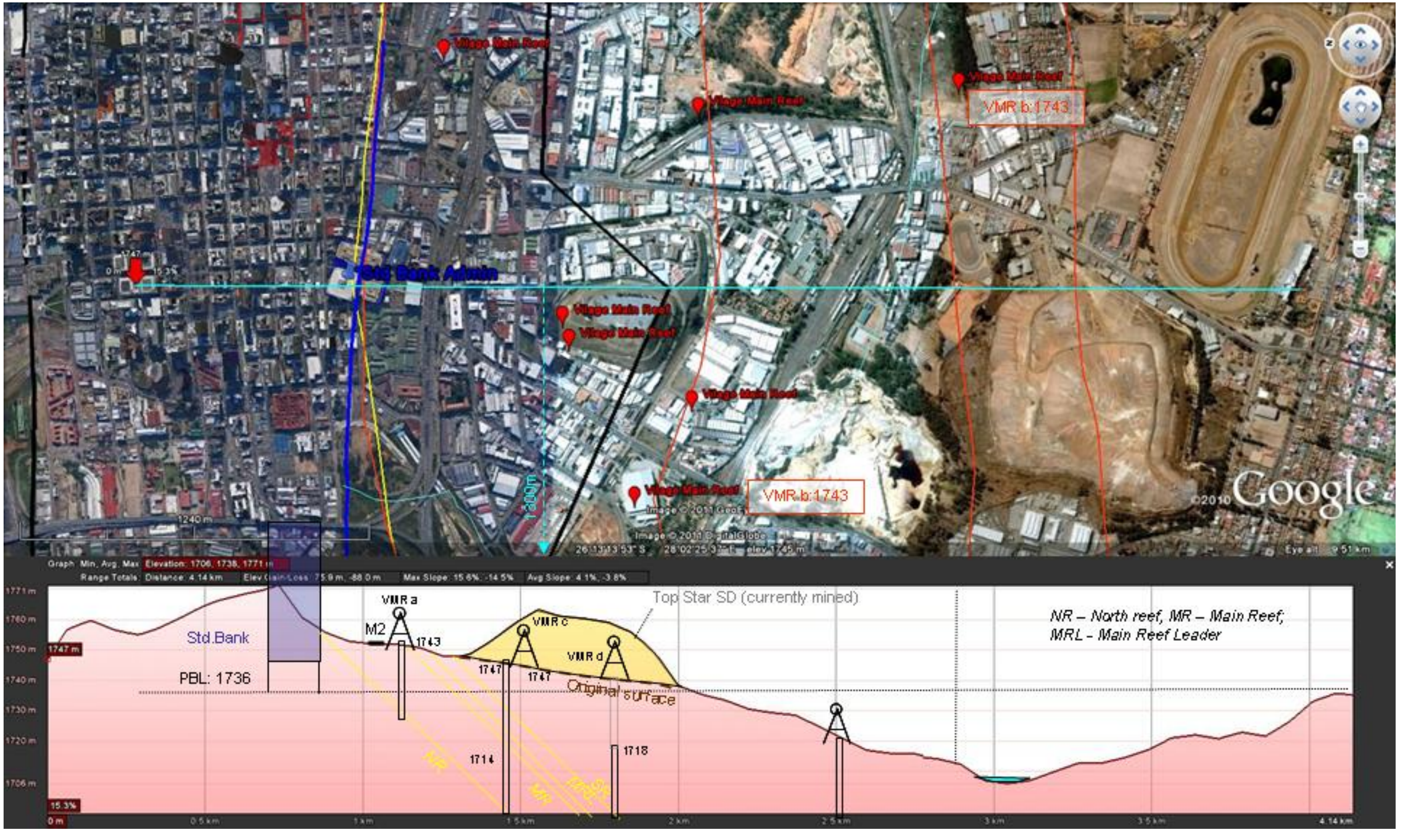

Fig. 8.21: N-S profile indicating the shafts and the Top Star SD near the Standard Bank new admin building

Some 600 m to the south of Std. Bank a large slimes dam (known as ‘Top Star’ that used to have a drive-in cinema on its top) is located which changed the natural relief and resulted in a localised inversion of the otherwise S-sloping topography (Fig. 8.21).

Since porewater trapped in the tailings deposits forms a water table a few metres below the surface of the slimes dams an elevated piezometric surface is generated that could possibly result in localised elevations of the water table or an inversion of the N-S groundwater flow direction. Given the additional elevation of some 15m above surface it is assumed that at least some of the SD seepage drains towards the MR outcrop (i.e. flows from S to N). However, currently it cannot reach the underground structures of the bank as the mine void is in between the tailings dam and the bank intercepting the seepage. The latter could be facilitated by two old shafts of the Village Main Reef gold mine that were later covered by the slimes dam (Fig. 8.21).

Located close to the MR outcrop where deep level mining started it is likely that these shafts are rather old and may not have been lined. In that case they would act like a French drain directing collected groundwater including tailings seepage to the deeper void. However, once the mine void is flooded this interception of seepage flow may no longer take place possibly posing a threat to underground building structures in the north.

As a rule of thumb for estimating the groundwater lift caused by slimes dams a geometric method can be applied that estimates the lift as a function of the horizontal extent of the SD (not the height though). From the point where the piezometric surface of the porewater intersects the edge of the SD at bottom level a straight diagonal line to the normal groundwater table is drawn. This line should intersect the groundwater table in a distance from the edge of the slimes dams that is equal to about 25% of the total lateral extent of the SD (i.e. length and width). Applying this method to the Top Star SD indicated that even under conditions of a completely flooded mine void the general N-S flow direction of the Groundwater would be maintained (Fig. 8.22) preventing a possible basement flooding by tailings seepage.

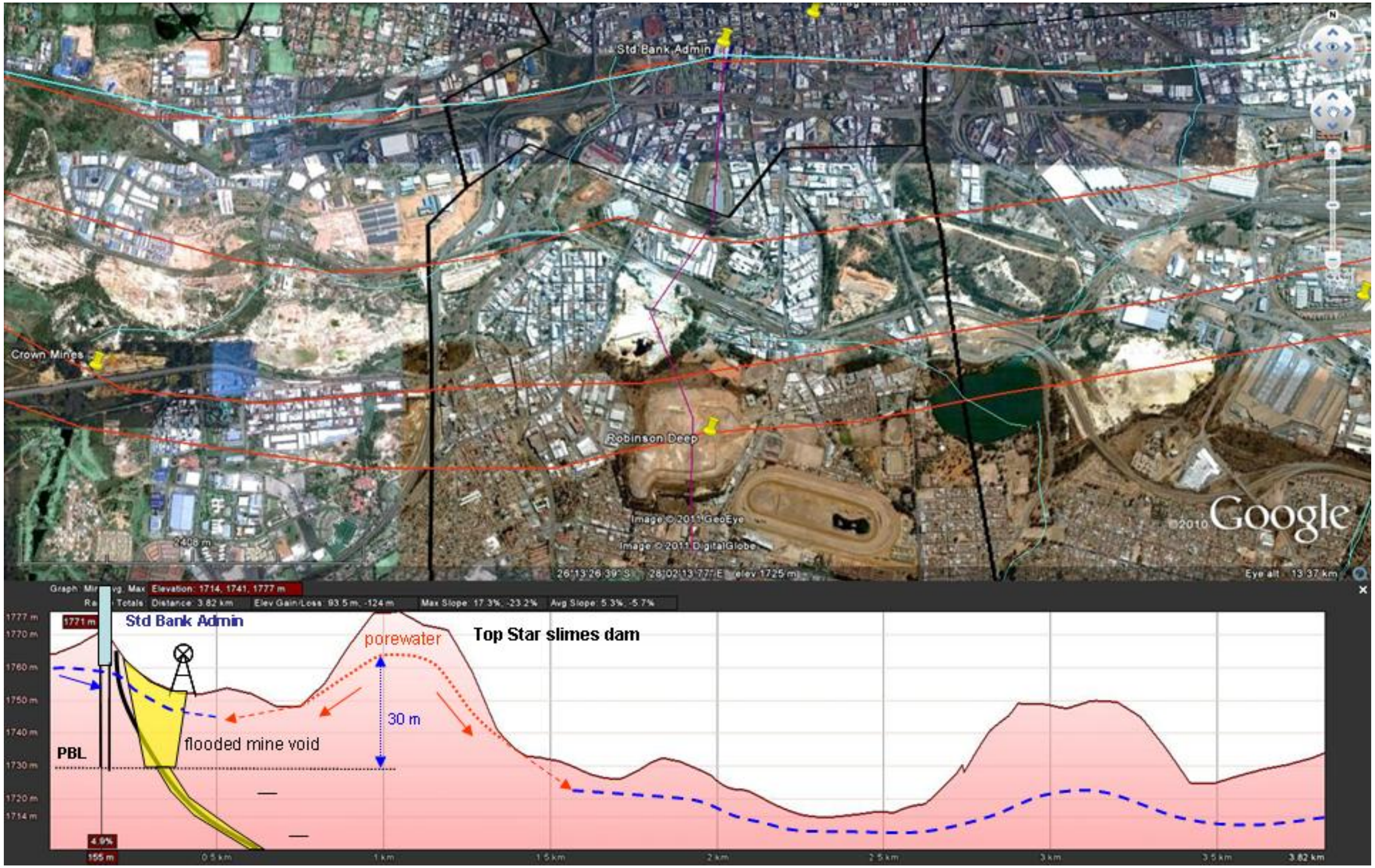

Fig. 8.22: N-S profile indicating the estimated groundwater level in the vicinity of the Top Star slimes dams under conditions of a hypothetical worst-scenario where the mine void is flooded close to surface.

8.5 Shaft collar elevations

Since shafts are the longest living structures of the mine void connecting it to the surface they are of particular significance for determining the final water level in the flooded mine void.

In this project shafts and associated elevations have been compiled from different sources. A list of 66 shafts including collar elevations has been compiled in Scott (1995). However, these shafts could not yet be georeferenced owing to technical difficulties. Fourie (2001) also indicated 66 shafts on a map which are, however not entirely identical with the ones listed in Scott (1995). The elevations of these shafts have been determined after importing the map into Google Earth. Since GE elevation were found to be in good agreement with surveyed elevations this method is regarded as reliable. However, the import of maps into GIS is associated with some degree of (lateral) inaccuracy which may result in locations of the shafts being not entirely correct. The possible error margin associated is estimated to be in the region of maximal 5-100 m. However, for 4 of the 5 x monitoring shafts used by DRD to measure the mine water level a very good fit regarding location and elevation was found.

Tab. 8.13: Surveyed shaft collar elevations of DRD monitoring shafts [mamsl] (Labuschagne 2010)

Name of monitoring shaft	Latitude	Longitude	Collar elevation [mamsl]
DRD no. 6 shaft	26°10'58.24"S	27°50'11.92"E	**1705.00**
Crown Mines: no. 14 shaft (Gold Reef City)	26°14'14.06"S	28°0'52.02"E	**1706.88**
City Deep: no. 4 shaft (market)	26°14'1.28"S	28° 4'16.37"E	**1691.80**
Simmer & Jack: Howard shaft	26°13'10.13"S	28° 7'37.67"E	**1652.93**
ERPM: SW Vertical shaft	26°13'3.24"S	28°10'57.76"E	**1653.24**

Tab. 8.14: Examples for surveyed shaft collar elevations provided by DRD

COLLAR ELEVATIONS OF THE LOWEST SHAFTS IN CENTRAL BASIN AT ERPM

DATE 27th September 2010

Description	Datum line elevation	Collar elev at MSL	Measured distance below collar	Water level below datum	Water level at MSL	Remarks
South West Vertical Shaft	175,75	1653,05	545,00	720,75	1108,05	OPEN SHAFT NOT CAPPED
Hercules Shaft	202,40	1626,40				FILLED AND CAPPED
Central Shaft	191,70	1637,10	529,05	720,75	1108,05	OPEN SHAFT NOT CAPPED
South East Vertical Shaft	214,39	1614,41				NOT PART OF CENTRAL BASIN (PLUGS INSTALLED -FEV DRY)
Far East Vertical Shaft	212,81	1615,99				NOT PART OF CENTRAL BASIN (PLUGS INSTALLED -FEV DRY)
Cinderella East Shaft	199,11	1.629,69	521,64	720,75	1.108,05	OPEN SHAFT 2nd LOWEST POINT IN CENTRAL BASIN
Cinderella West Shaft	203,57	1.625,23	517,18	720,75	1.108,05	OPEN SHAFT LOWEST POINT IN CENTRAL BASIN
Comet Vertical Shaft	181,70	1647,10				FILLED -NOT CAPPED
Angelo Vertical Shaft	174,86	1653,94				FILLED -NOT CAPPED
Cason Incline Shaft	182,65	1646,15	538,10	720,75	1108,05	OPEN SHAFT NOT CAPPED
Angelo Main incline Shaft	176,18	1652,62	544,57	720,75	1108,05	OPEN SHAFT NOT CAPPED
Datum line (6000 Ft)	1828,80					

Cinderella west shaft is situated south of Rondebult road between Boksburg lake and Cinderella dam (north of Hypermarket)

From 17 Aug 2009 to 22 Sept 2010 water raised by 218.1m in (Central basin DRD / ERPM)

Daily rise in this period was 0.545mm

674 days @ current rate to 150m below surface

949 days @ current rate to surface

Vivian Labuschagne
Survey dept.
Date 27September 2010

Finally a combined list based on both data sets was compiled comprising a total of 110 x shafts across the CR including their collar elevation (Fig. 8.23).

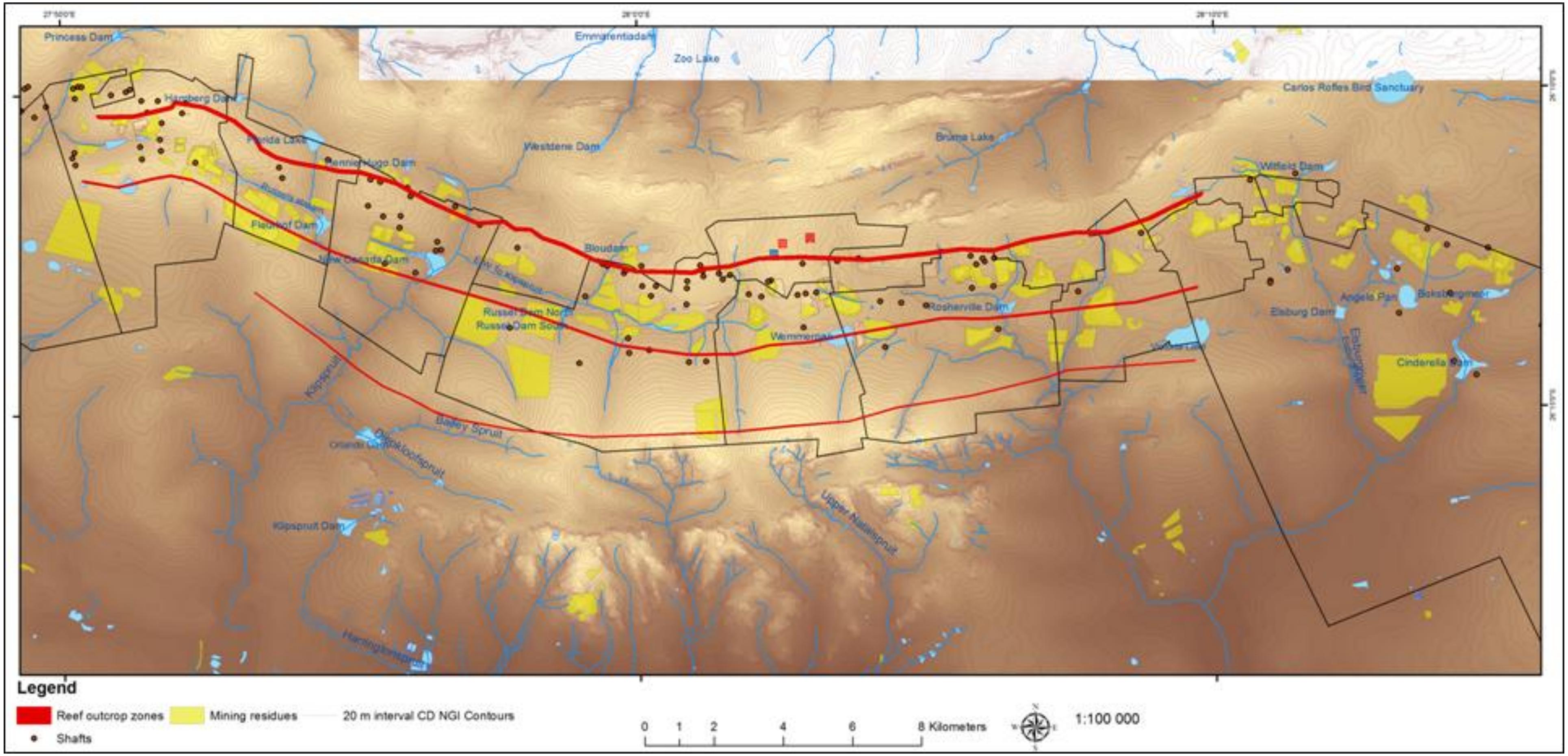

Fig. 8.23: Distribution of shafts in the Central Rand (based on various sources)

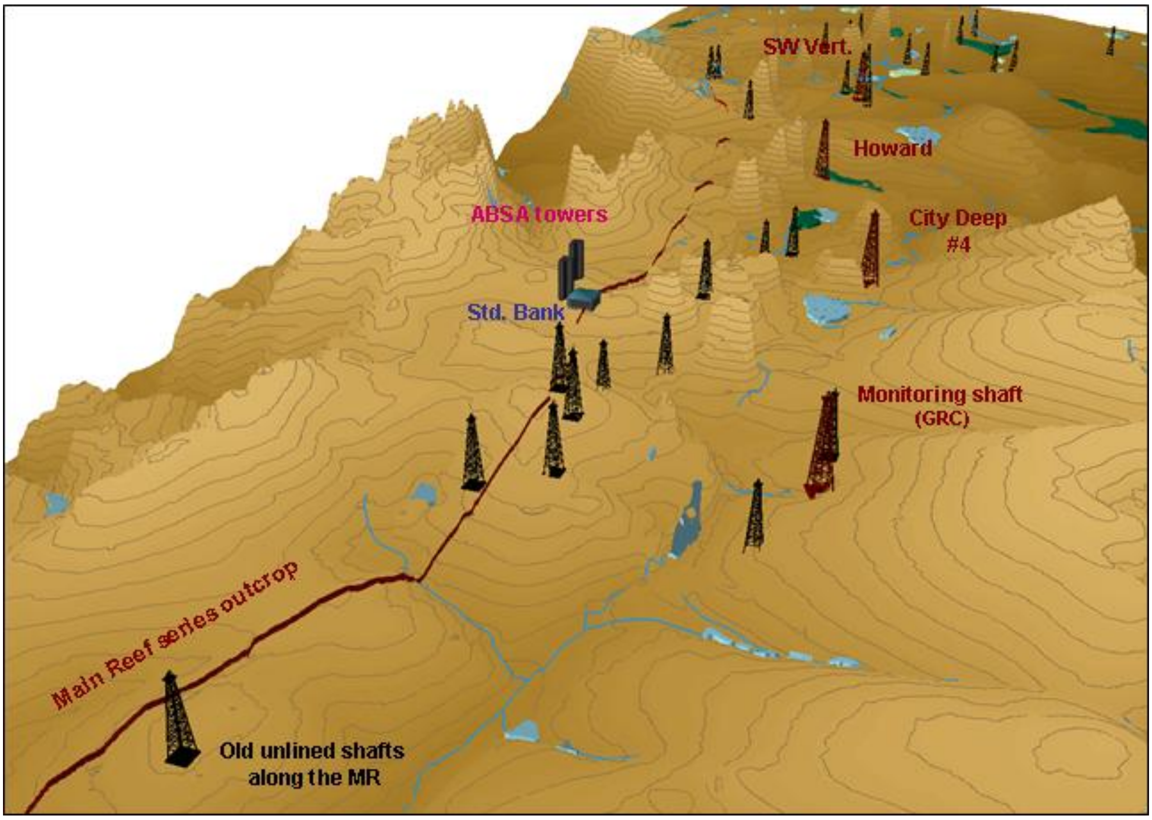

Fig. 8.24: A 3D-view of central part of Central Rand depicting the positions of shafts relative to the bank buildings (red large shafts symbols indicate monitoring shafts)

Elevations for shafts are commonly given as ‘Collar elevations’ referring the upper edge of a 10-50cm thick concrete slab directly placed on surface providing stability for the shafts structure above. Therefore, highly accurate surveyed collar elevations are somewhat higher (10-50 cm) than the actual surface measured for example by Google Earth or Lidar (Labuschagne, 2011). The collar elevations for the 111 x shafts in the Central Rand are listed in Tab. 8.15.

Tab. 8.15: Collar elevations of shafts in the CR compared against the CPBL/PBL

Nr in map	Shaft name	Coordinate Latitude	Longitude	Collar elevation [mamsl] DRD survey	Google Earth	LIDAR	5m CDNGI	Old shaft, unlined (all shafts within 520 m of MR outcrop)	Remarks (empty cells: capping/ filling unknown)	Diff. shaft collar (GE/ *survey*) - CPBL (1704 mamsl) [m]	PBL Std. Bank (1720 mamsl) [m]
50	#7	26d11'03.25"S	27d52'20.05"E		1769	1775,18	1771,74			-65	-49
102		26d10'16.85"S	27d52'06.54"E		1766	1759,06	1759,18	1		-62	-46
103		26d12'58.25"S	28d02'18.06"E		1761	1761,80	1761,61			-57	-41
46	#1	26d10'07.29"S	27d52'10.51"E		1760	1755,72	1756,44	1		-56	-40
45	#2	26d10'05.11"S	27d51'41.37"E		1757	1752,87	1754,87	1		-53	-37
47	Circular	26d10'25.94"S	27d51'45.49"E		1757	1744,90	1743,65	1		-53	-37
101		26d10'15.94"S	27d51'30.35"E		1754	1748,03	1749,66	1		-50	-34
44	#3	26d10'04.98"S	27d51'24.45"E		1751	1746,89	1749,78	1		-47	-31
104		26d12'59.03"S	28d02'15.13"E		1750	1735,91	1735,50			-46	-30
7	#2 - S	26d12'53.83"S	28d01'07.96"E		1745	1735,79	1736,23	1		-41	-25
15	#3	26d12'58.13"S	28d00'51.14"E		1743	1735,07	1735,04	1		-39	-23
8	#2 - N	26d12'43.64"S	28d01'04.51"E		1743	1740,59	1739,98	1		-39	-23
6	#2	26d12'48.65"S	28d01'07.06"E		1742	1739,26	1738,54	1		-38	-22
87		26d12'42.49"S	28d02'51.07"E		1742	1732,91	1731,99	1		-38	-22
77		26d10'59.69"S	27d51'24.7"E		1739	1733,80	1734,04			-35	-19
76		26d10'52.19"S	27d51'44.46"E		1738	1732,37	1733,94			-34	-18
63		26d13'12.05"S	28d02'45.88"E		1736	1728,57	1729,82			-32	-16
90		26d13'04.93"S	28d00'50.43"E		1734	1729,42	1731,82	1		-30	-14
109		26d09'56.76"S	27d51'08.2"E		1734	1730,37	1732,25			-30	-14
110		26d09'54.47"S	27d51'12.58"E		1734	1729,98	1731,10			-30	-14
20	Village Deep	26d13'10.65"S	28d02'54.22"E		1734	1727,79	1730,05			-30	-14
89		26d12'51.07"S	28d01'23.88"E		1732	1728,52	1725,54	1		-28	-12
16	#6	26d13'02.78"S	28d00'18.32"E		1731	1724,77	1726,07	1		-27	-11
62		26d13'09.64"S	28d03'05.16"E		1731	1724,89	1724,87			-27	-11
48	#5	26d10'41.52"S	27d51'43.53"E		1729	1727,95	1726,46			-25	-9
65		26d13'13.46"S	28d02'08.13"E		1729	1722,50	1721,80			-25	-9
91		26d13'02.61"S	28d00'04.26"E		1728	1723,81	1725,30	1		-24	-8
111	#1	26d12'56.75"S	28d01'27.89"E		1723	1717,21	1719,39	1		-19	-3
98		26d11'20.41"S	27d55'22.33"E		1722	1715,57	1715,49	1		-18	-2
10	Aurora	26d11'22.99"S	27d55'32.42"E		1721	1712,19	1714,29	1		-17	-1
17	#7	26d12'43.83"S	28d00'03.36"E		1720	1714,59	1716,22	1		-16	0
105		26d13'20.07"S	28d00'50.14"E		1720	1714,04	1718,57			-16	0
106		26d13'12.01"S	28d00'13.15"E		1720	1715,22	1719,57			-16	0
										shafts **below** the PBL of **Std. Bank** (**1720** mamsl)	
12	Robinson Deep	26d13'10.91"S	28d01'54.79"E		1717	1713,67	1709,07			-13	3
88		26d12'52.5"S	28d01'35.64"E		1715	1707,07	1716,51	1		-11	5
49	#8	26d10'48.34"S	27d51'22.95"E		1713	1709,61	1711,67			-9	7
78		26d11'04.92"S	27d50'15.21"E		1713	1710,49	1712,23			-9	7
100		26d11'08.71"S	27d53'47.23"E		1713	1708,89	1710,05	1		-9	7
64		26d13'42.29"S	28d02'51.6"E		1713	1704,77	1705,13			-9	7
84		26d12'36.32"S	28d05'46.02"E		1712	1703,64	1705,51	1		-8	8
69		26d14'14.22"S	27d58'58.15"E		1712	1705,83	1705,13			-8	8
93		26d12'47.59"S	27d59'46.93"E		1712	1703,37	1703,98	1		-8	8
53	#8	26d11'45.24"S	27d55'19.35"E		1712	1710,58	1710,91			-8	8
85		26d12'37.9"S	28d03'49.34"E		1712	1706,78	1707,99	1		-8	8
18	#16a	26d14'13.64"S	28d01'10.19"E		1710	1703,81	1704,35			-6	10
86		26d12'40.34"S	28d03'27.13"E		1710	1704,46	1705,09	1		-6	10
75		26d11'55.14"S	27d55'35.19"E		1709	1702,63	1702,41			-5	11
92		26d12'50.25"S	27d59'45.27"E		1708	1700,48	1703,60	1		-4	12
9	#9	26d11'55.03"S	27d55'53.23"E		1708	1702,23	1703,13			-4	12
56	Mooifontein	26d12'25.69"S	27d57'54.57"E		1707	1702,40	1701,09			-3	13
5	#14 (Gold Reef City)	26d14'14.06"S	28d00'52.02"E	1706,88	1706	1699,38	1696,60		monitoring shaft	-3	*14*
3	#6	26d10'58.13"S	27d50'12.01"E	1705,00	1705	1697,74	1699,86		monitoring shaft	-1	*15*
82		26d12'38.77"S	28d05'58.63"E		1704	1698,02	1699,62	1		0	16
										Shafts **below** the **CPBL (1704** mamsl)	
81		26d12'41.88"S	28d06'00.54"E		1703	1695,53	1695,26	1	possibly diffuse outflow	1	17
19	#15	26d14'02.68"S	28d00'11.01"E		1703	1698,87	1699,75			1	17
40	Outcrop Inclines "4"	26d09'52.95"S	27d49'22.88"E		1703	1698,38	1699,42	*1*	possibly diffuse outflow	1	17
41	Outcrop Inclines "5"	26d09'51.03"S	27d49'26.86"E		1703	1698,82	1700,00	*1*	possibly diffuse outflow	1	17
51		26d11'18.57"S	27d53'50.29"E		1703	1699,79	1700,26			1	17
83		26d12'44.5"S	28d05'51.76"E		1702	1695,51	1696,56	1	possibly diffuse outflow	2	18
97		26d11'36.79"S	27d56'03.57"E		1702	1698,28	1699,99	1	possibly diffuse outflow	2	18
70		26d13'40.64"S	27d57'45.69"E		1701	1694,19	1694,76			3	19
80		26d12'38.68"S	28d06'10.58"E		1700	1688,64	1688,25	1	possibly diffuse outflow	4	20
112	#9	26d10'53.48"S	27d50'14.48"E		1698	1687,62	1698,08			6	22
52	New Unified	26d11'28.65"S	27d56'00.56"E		1696	1689,62	1694,57	1	possibly diffuse outflow	8	24
74		26d12'06.01"S	27d55'52.06"E		1696	1690,61	1689,95			8	24
21		26d12'15.99"S	28d08'43.67"E		1696	1688,00	1689,46	1	possibly diffuse outflow	8	24
66		26d14'05.15"S	27d59'50.1"E		1695	1691,51	1691,58			9	25
95		26d12'41.32"S	27d59'22.96"E		1693	1687,70	1689,32	1	possibly diffuse outflow	11	27
99		26d11'01.76"S	27d54'38.14"E		1692	1686,43	1686,69	1	possibly diffuse outflow	12	28
1	#4 (market)	26d14'01.28"S	28d04'16.37"E	1691,80	1692	1682,87	1681,56		monitoring shaft	12	*28*
13	#2	26d13'18.31"S	28d04'11.61"E		1691	1687,26	1685,23			13	29
94		26d12'43.55"S	27d59'28.01"E		1689	1690,30	1679,32	1	possibly diffuse outflow	15	31
39	Outcrop Inclines "3"	26d09'51.81"S	27d50'22.29"E		1688	1686,00	1688,98	*1*	possibly diffuse outflow	16	32
107		26d10'02.47"S	27d50'15.39"E		1688	1684,77	1687,45			16	32
67		26d13'51.16"S	27d59'49.15"E		1687	1679,61	1679,86			17	33
38	Outcrop Inclines "2"	26d09'51.45"S	27d50'18.7"E		1684	1671,95	1682,75	*1*	possibly diffuse outflow	20	36
71		26d12'39.42"S	27d55'36.38"E		1680	1673,23	1664,92			24	40
22	Waverely Deep	26d11'26.87"S	28d10'37.36"E		1679	1675,00	1672,03			25	41
58		26d13'19.98"S	28d04'33.76"E		1677	1672,66	1673,58			27	43
68		26d13'11.72"S	27d59'04.75"E		1676	1666,75	1668,83			28	44
37	Outcrop Inclines "1"	26d09'53.04"S	27d50'13.64"E		1676	1672,76	1676,95	*1*	possibly diffuse outflow	28	44
42	Princesses	26d10'10.02"S	27d49'45.03"E		1675	1669,08	1670,23			29	45
79		26d10'19.74"S	27d49'33.05"E		1675	1670,56	1670,36			29	45
55	#3	26d12'19.56"S	27d56'31.07"E		1675	1669,15	1668,18			29	45
54	Central	26d11'46.56"S	27d56'50.09"E		1674	1655,54	1661,98	1	possibly diffuse outflow	30	46
43	#4	26d10'13.71"S	27d49'19.43"E		1672	1666,14	1667,52			32	48
108		26d09'59.41"S	27d50'53.12"E		1672	1722,58	1723,19			32	48
59		26d13'06.4"S	28d05'47.24"E		1671	1668,51	1667,45			33	49
96		26d12'03.95"S	27d57'15.53"E		1670	1654,09	1664,88	1	possibly diffuse outflow	34	50
14	#1	26d13'22.51"S	28d04'59.12"E		1668	1663,88	1665,90			36	52
60		26d13'04.81"S	28d06'09.57"E		1668	1661,26	1661,13			36	52
73		26d12'26.93"S	27d56'35.02"E		1663	1653,58	1654,76			41	57
61		26d13'45.36"S	28d06'14.36"E		1661	1647,21	1649,82			43	59
72		26d12'27.72"S	27d56'29.57"E		1661	1651,05	1654,87			43	59
23		26d11'21.35"S	28d11'24.68"E		1659	1649,79	1650,83			45	61
11	#6	26d12'48.3"S	27d56'08.34"E		1655	1649,91	1650,00			49	65
27	Angelo West (Vertical)	26d12'50.97"S	28d13'09.96"E	1653,94	1620	1617,09	1620,26		filled not capped	50	*66*
24	South West Vertical (SWV)	26d13'01.28"S	28d10'58.05"E	1653,05	1652	1646,54	1645,48		monitoring shaft	51	*67*
4	Howard shaft	26d13'10.49"S	28d07'37.85"E	1652,93	1653	1648,98	1645,38		monitoring shaft	51	*67*
26	Angelo Main Incline	26d12'13.77"S	28d13'41.54"E	1652,62	1650	1643,00	1645,00		open, not capped	51	*67*
25	South West Vertic. Ventilation	26d12'51.31"S	28d11'16.13"E		1649	1643,97	1643,93			55	71
30	Comet Vertical	26d12'28.83"S	28d14'01.98"E	1647,10	1642	1639,76	1639,90		filled, not capped	57	*73*
34	Cason Incline	26d12'31.65"S	28d14'44.82"E	1646,15	1640	1635,00	1636,56		open, not capped	58	*74*
28	Central	26d13'32.57"S	28d13'11.85"E	1637,10	1631	1624,66	1625,06		open, not capped	67	*83*
35	Cinderella East	26d13'44.73"S	28d15'40.19"E	1629,69	1627	1618,84	1619,58		open, 2nd lowest shaft in central basin	74	*90*
29	Hercules	26d13'14.01"S	28d14'04.92"E	1626,40	1621	1614,49	1616,42		filled and capped	78	*94*
36	Leeupoort	26d13'53.98"S	28d16'10.39"E		1624	1621,47	1622,59			80	96
33	*Far East Vertical*	*26d15'40.18"S*	*28d15'59.38"E*	*1615,99*	*1604*	*1603,29*	*1597,00*		*plugged, not linked to central sub-basin*	88	*104*
32	*South East Vertical (SEV)*	*26d14'30.84"S*	*28d14'32.16"E*	*1614,41*	*1608*	*1596,86*	*1602,85*		*plugged, not linked to central sub-basin*	90	*106*
113	Cinderella West (Ventilation)	26d13'33.60"S	28d14'45.60"E	1613,72	1621	1613,47	1615,28		open, lowest elev. = decant shaft	90	*106*
31	Hercules South	26d14'17.71"S	28d14'08.47"E		1603	1598,62	1601,20			101	117
			n	15	111	111	111	43			
Colour code key:			nr. of shafts below CPBL/PBL							58	78
open shaft below CPBL/PBL: unrestricted outflow			average	1649,79	1697,86	1692,47	1693,40			5,46	21,47
old, unlined shafts below CPBL: diffuse outflow			minimum	1613,72	1603,00	1596,86	1597,00			-62,00	-46,00
not hydraul. connected to central sub-basin			maximum	1706,88	1769,00	1775,18	1771,74			101,00	117,00

As vertical structures, not exposed to the pressure of an overlying rock column and well supported by massive concrete lining, anchored in hard rock, shafts are amongst the most lasting elements of the mine void. This renders all shafts potential pathways for rising mine water to reach the surface. Where the opening of shafts is not closed-off by some form of capping, coverage or filling, shafts can act as decant points where the collar elevation is below the piezometric surface (which in turn is determined by the elevation of the main ingress source feeding the mine void). In how far the capping or covering of shafts will indeed prevent the rising mine water to reach the surface is uncertain as artesian pressure may built up inside the shaft, driving increasingly more water through cracks in the lining and fractures in the host rock into lower lying surface areas. Capping commonly consists of placing a concrete slab across the opening and anchoring it into competent surrounding bedrock. In order to assess the competency of the bedrock for shaft capping, regulations prescribe that the shaft lining has to be cut in order to test the rock strength. It is therefore likely that water under artesian pressure will still be able to daylight through these cuts even in capped shafts. Moreover, to allow for barometric pressure difference to equalize, air pipes penetrate through the concrete cap providing an additional outflow pathway (Marais 2002).

Shafts sunk before 1912 were not grouted or lined with concrete at all, which – during the mining and flooding phase – renders them French drains which collect and direct groundwater into the mine void[1]. With mine water levels rising close to surface, the inverse process is likely to occur i.e. the migration of mine water from the unlined shafts through fractures, faults and dykes to surface. Since nearly half of all shafts below the CPBL are old unlined shafts (30 x) it seems not impossible that these shafts alone would be able to control the final mine water level. Moreover, diffuse outflow of artesian mine water via geological pathways may also occur at shafts with collar elevations above the CPBL. This is especially true for old shafts of which 10 were found to be above the CPBL. Thus the total number of outflow shafts may in fact be closer to 77 (i.e. nearly 90% of all shafts).

While the decant capacity of closed shafts, unlined or lined is subject to some degree of speculation, this is not the case for shafts (below the CPBL) which are still open. In addition to the 5 x monitoring shafts across the entire CR this applies to another 6 x shafts mainly located at ERPM. With shafts diameters of 4 m x 4 m and bigger each of the open shafts alone could accommodate the full decant volume (whose determination will be discussed later in the report).

[1] Acc. to Scott, 1995: 1130 m from S-Reef, a sketch however indicates that a number of shafts are located 2-3 km south of the MR outcrop) to access the S-dipping reefs at greater depths. According to Scott (1995) the sinking of deep vertical shafts (in contrast to incline shafts used before) happened in two phases. The first shafts (i.e. the oldest ones) were sunk from the early 1890s onwards mainly in the vicinity of the MR outcrop. Where the outcrop was mined on surface (the overwhelming majority of the strike length) shafts were placed to the south of surface diggings some 300 to 520 m from the outcrop of the Southern Reef (Scott, 1995).

The most important conclusion from Tab. 8.15 is that the lowest lying shaft which is still open (Cinderella West shaft at ERPM near Boksburg at **1614 mamsl**) and thus likely to be the decant point for the Central basin is 90 m below the critical PBL and 106 m below the lowest basement structure of Std. Bank. Considering a maximum error of 7 m (the amount of which the Google Earth elevation for this particular shaft deviates from the surveyed) this clearly indicates that there is no flooding risk for the 2 x key buildings or any other building in the CBD of JHB. As the shaft is still open and able to accommodate even improbable high decant volumes of 60 to 90 Ml/d this shaft can control the final mine water level in the flooded basin.

Should, for some or other reason, the Cinderella W shaft not be able to act as decant point (e.g. because it is insufficiently connected to the mine void, collapses in the meantime or will be closed for safety reasons etc.) another 6 x open shafts exist located between 51 m and 74 m below the CPBL (60 m and 90 m below Std. Bank PBL) which could take over this function without compromising the safety of the bank buildings. The fact that 2 x of these shafts are monitoring shafts for which monthly measurements confirm their hydraulic connection to the central basin further adds confidence in the finding that no flooding risk associated with the filling of the mine void exists for the entire CBD. Fig. 8.25 depicts the elevation of 7 x open shafts along an E-W profile in relation to the decant level and the CPBL as well as the PBL of Std. Bank.

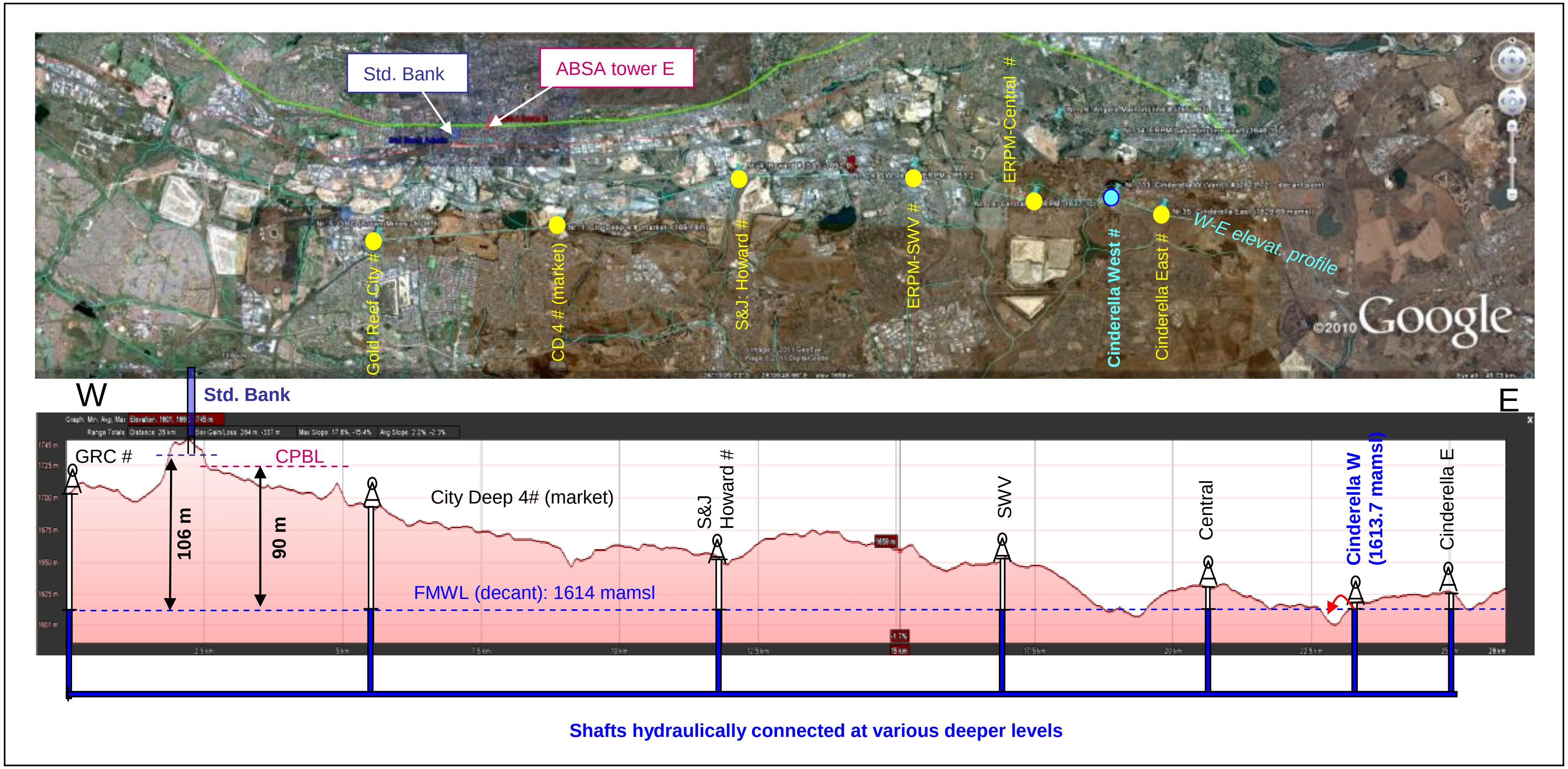

Fig. 8.25**:** Elevation of selected open shafts along an E-W profile in relation to the decant level and the CPBL as well as the PBL of Std. Bank indicating a safety margin of 90 m and 106 m respectively.

The situation resulting from this decant level along the N-S profile from the banks across the mining belt is depicted in Fig. 8.26.

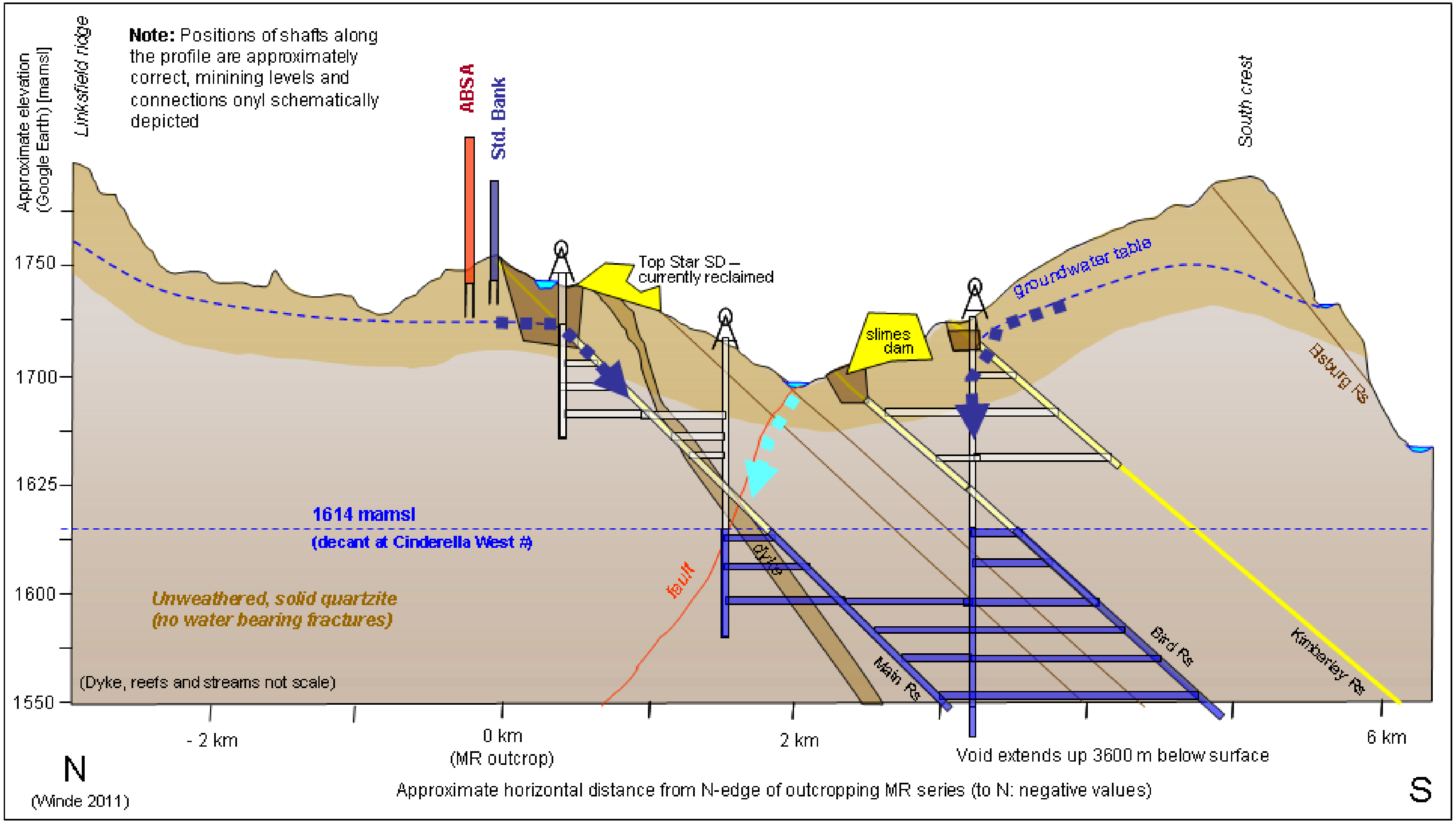

Fig. 8.26: N-S profile from the banks to the mining belt indicating the final level of flooding in the mine void in relation to the PBLs of the two bank buildings.

The sketch in Fig. 8.26 indicates that water will neither reach the basement structures of the two key buildings (and no other building in the CBD for that matter) nor the disturbed outcrop zone (at least in this part of the mining belt) limiting the earlier discussed geotechnical risks possibly associated with mine water saturating unconsolidated material. It also alleviates fears that seepage from an adjacent SD may drive water towards the banks as such seepage will continue to flow into the deeper mine void.

As the FMWL will remain over 100 m below the surface most ingress sources in this part of the Central Rand will not be cut off and thus continue to recharge the mine void (Fig. 8.27).

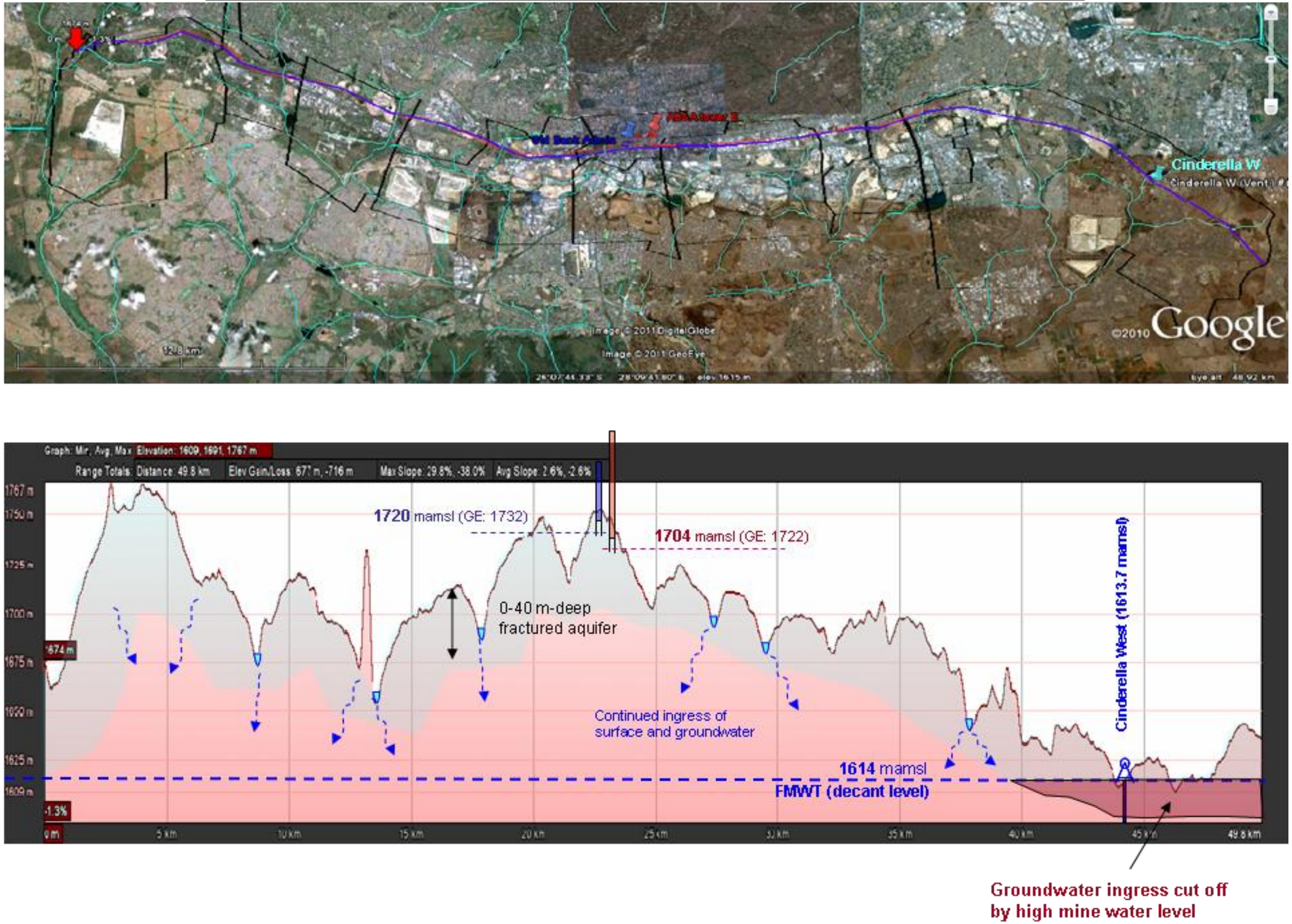

Fig. 8.27: E-W cross section based on GE elevations depicting where mine water rising to the decant level of 1614 mamsl will cut off diffuse inflow of groundwater from the fractured aquifer and thus reduce post-flooding ingress.

Fig. 8.27 suggests for the expected decant level of 1614 mamsl that nearly the entire MR outcrop zone along the strike length of some 44 km will continue to act as a recharge (ingress) area for the underlying mine void. A possible exception is the low lying area at ERPM where the FMWT may cut off some of the diffuse groundwater inflow. Assuming that the groundwater inflow is homogenous over the entire strike length of the MR outcrop the associated reduction in groundwater inflow is approximately 17% (assuming that over a distance of 5 km in the eastern part groundwater inflow is completely cut off). Based on Scott (1995) who estimated a groundwater derived ingress of 24 Ml/d this would mean a reduction in decant of some 4 Ml/d. However, the actual reduction my be somewhat larger as the 2 x other reef outcrop areas lie somewhat lower possibly increasing the proportion of groundwater inflow that may be cut off by the rising mine water level.

All in all 58 of the 111 shafts (= 52%) display collar elevations below the CPBL. Using the PBL of Std. Bank as reference 78 x shafts are below this level. However, for most of these shafts it is unknown whether or not they are filled, capped or otherwise closed potentially reducing there ability to act as decant points.

Apart from the 8 x open shafts (of which 3 x are used as monitoring shafts for which sufficient hydraulic connection to the mine void system is confirmed) a total of 16 old unlined shafts exists whose collar elevations are also below the CPBL. Since these shafts are unlined, chances are that mine water will migrate through fissures, fractures and other connecting conduits from the shaft to the surface even in cases where these shafts have been capped or covered by mining residues. Fig. 8.28 indicates that diffuse seepage of mine water out of old unlined shafts into the disturbed outcrop zone (which was assumed to be present over the entire strike length of the outcropping MR extending some 40 m below surface) could only affect some low lying areas at ERPM (red straight arrows in Fig. 8.28).

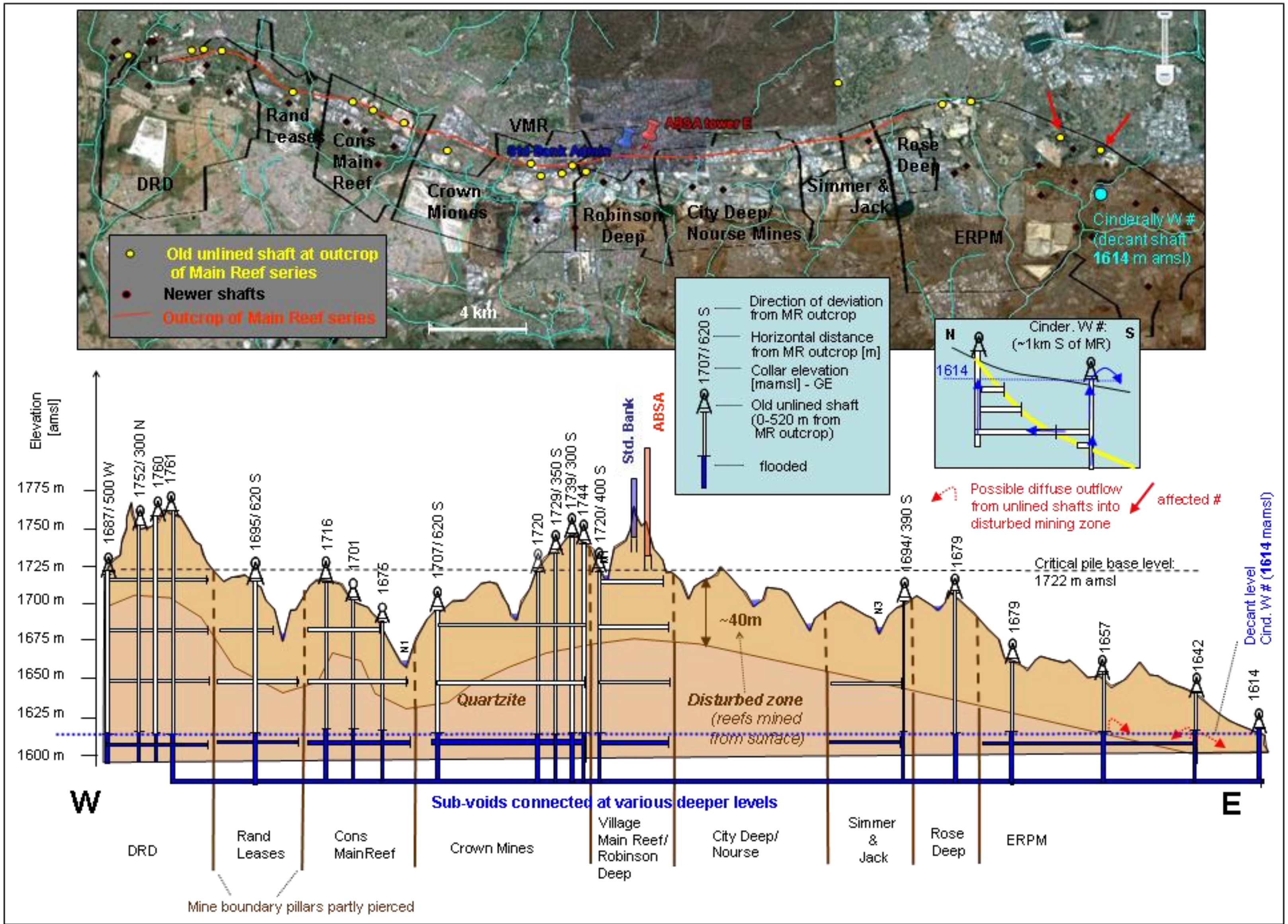

Fig. 8.28: E-W cross section indicating where mine water from old unlined shafts could possibly seep into the disturbed mined MR outcrop zone causing geotechnical stability problems based on an assumed decant level of 1614 m defined by the Cinderella West shafts at ERPM

Figure 8.29 indicates which areas in the CR are lying below the decant level of 1614 mamsl potentially rendering them prone to be affected by mine water seepage from old shafts.

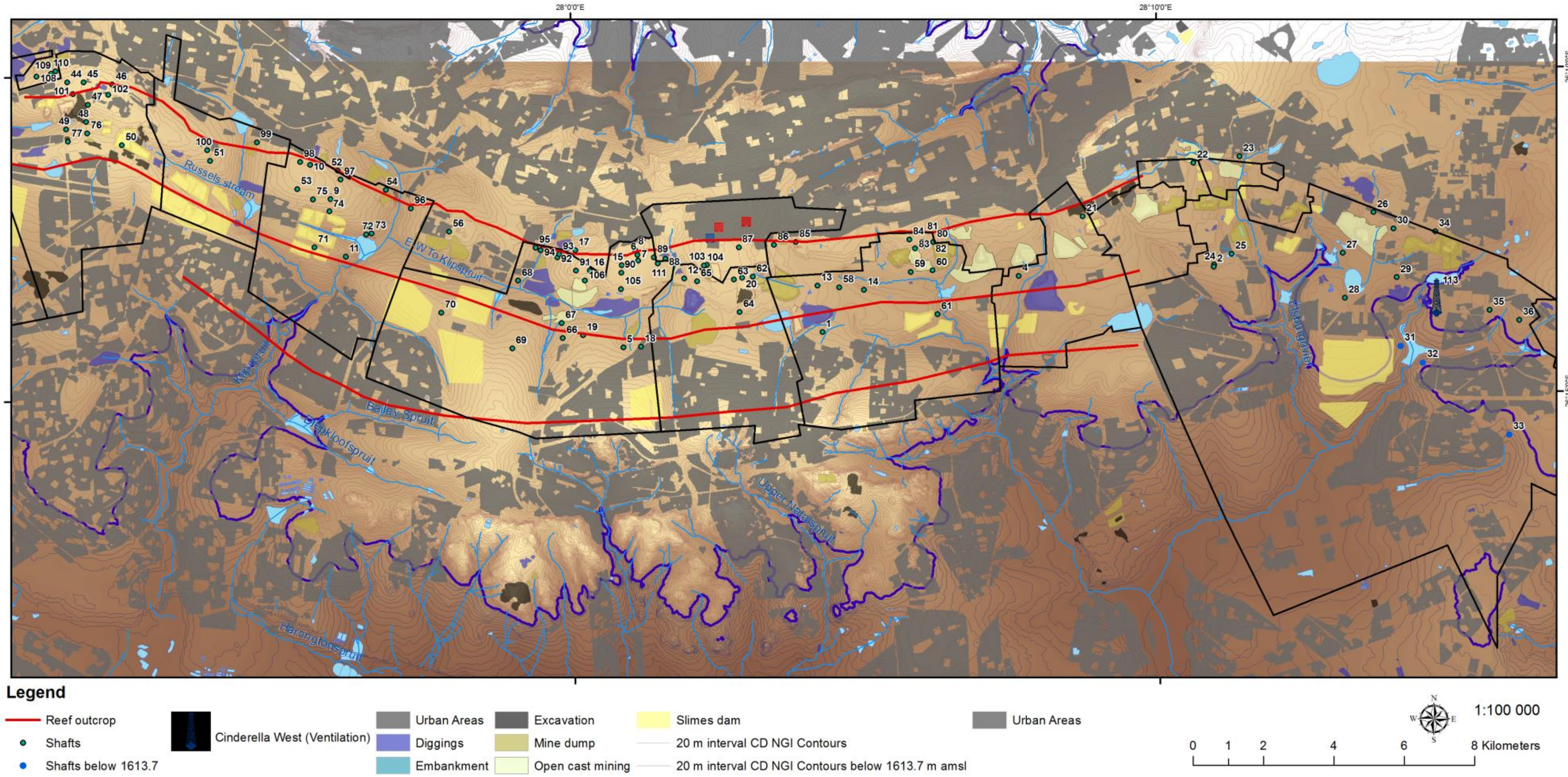

Fig. 8.29: The purple contour line indicates the decant elevation of 1614 mamsl delineating all areas below which are potentially affected by mine water seeping diffusely from the flooded void along transmissive features such as faults, dykes as well as low shafts with compromised lining..

Using a 3 dimensional view Fig. 8.30 depicts areas potentially affected by diffuse seepage of mine water in the eastern part of the CR.

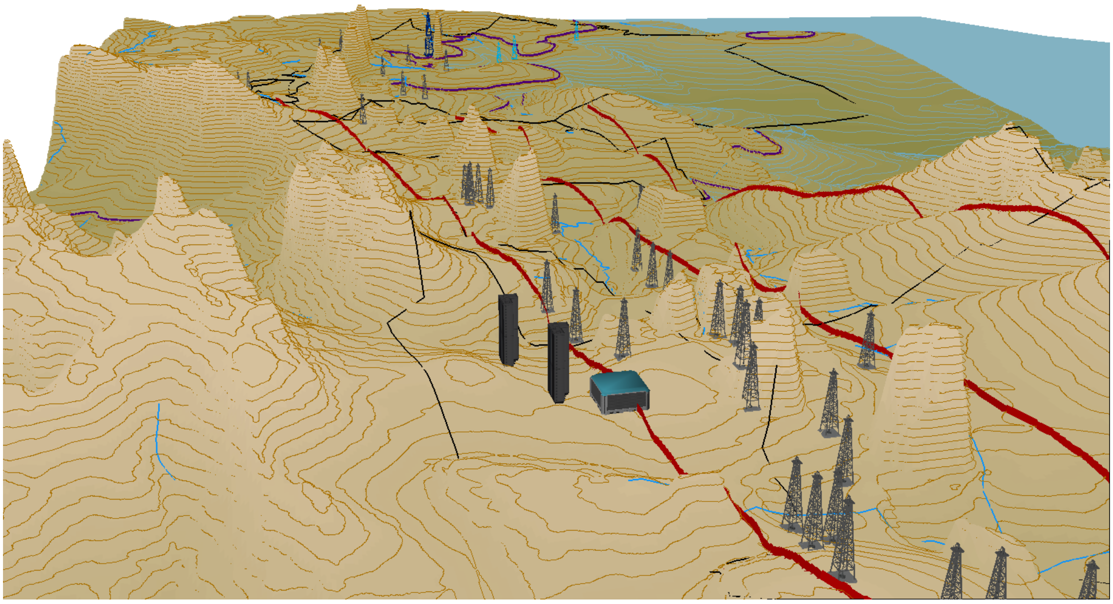

Fig. 8.30: 3D view of the CR looking from W to E with light blue transparent cover indicating all areas located below the decant level of 1614 mamsl. In the foreground the 3 x key bank buildings are depicted, the red lines indicate the outcrop zones associated with the packages of the MR, Bird Reef and Kimberley Reef (from left to right).

Note: the possibly created impression of large areas lying under water is wrong as seepage will be relatively low in volume and easily carried southwards by receiving streams and rivers thus preventing any build up of water that could submerge the low lying areas.

Besides the decant shaft mentioned above, mine water seepage is only likely to daylight where hydraulic connections between the flooded part of the mine void and the low lying surface areas exists. These could be provided by old unlined shafts, low lying disturbed outcrop zones of the 3 x reef packages and transmissive geological features such as faults dykes. Although the Cinderella West shaft can accommodate all ingressing water (i.e. the decant) diffuse seepage of mine water is likely to occur where hydraulic heads perhaps of several metres between the void (shafts, stopes haulages) and the receiving environment exist.

Fig. 8.31 (below) depicts the FMWL for ERPM suggesting that portions of 2 x valleys will be located below the mine water level, rendering streams running in these valley prone to be affected by diffuse mine water seepage.

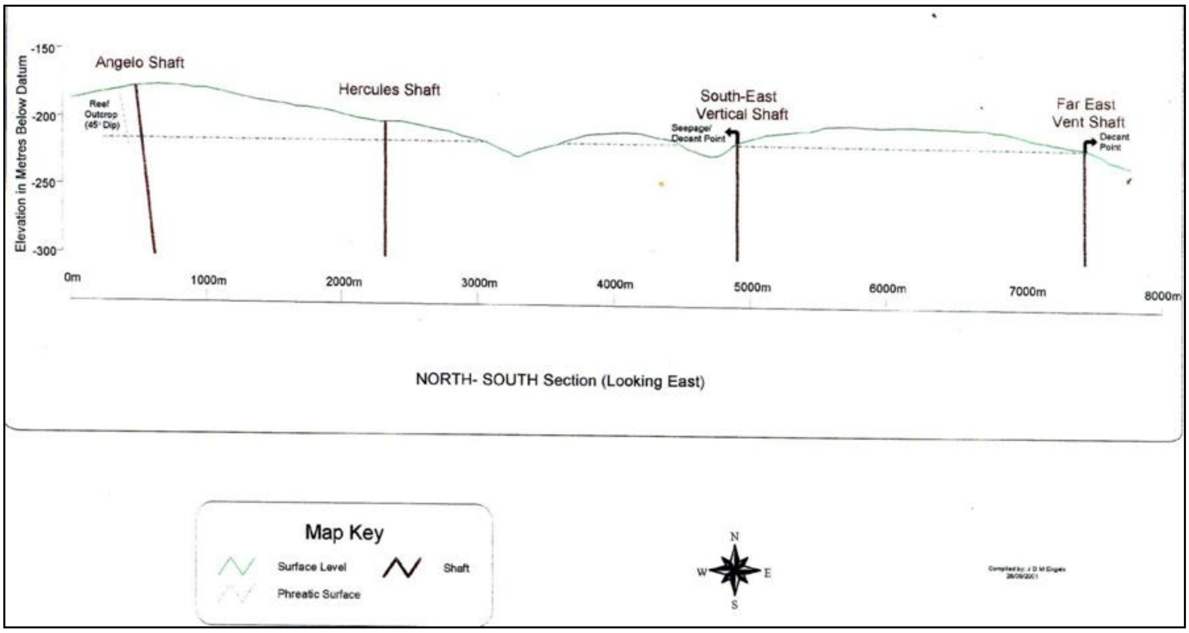

Fig. 8.31: Elevations of potential decant shafts at ERPM including the predicted level of the piezometric surface after flooding of the mine void is complete indicating the intersection of low lying relief lines that therefore may also act as (diffuse) decant points (EMPR of ERPM, 2001)

Since recent water table measurements indicate a largely synchronised rate of rise pointing to freely flowing water across the 9 x interconnected sub-voids (from DRD in the W to Rose Deep in the E) the above worst-case scenario has a low probability of being applicable. Unfortunately no monitoring data for water levels in this sub-void are available.

Although the formation of a topography-related hydraulic head across the 40 km-long mine void cannot be excluded (Scott, 1995 estimates an up to 60 m high head between DRD in the W and the outflow point at ERPM) this is not supported by current measurements showing a rather flat water table across the entire system from DRD to Rose Deep.

Should a further reduction of the final water level be required it should be explored how lower lying shafts have been closed (filling to depth vs. capping) in order to explore possibilities for re-opening.

According to the ‘Generic code of practice for the sealing of shafts’ (Marais, 2000), all capped shafts could be re-opened with a limited amount of costs and effort as commonly only 1 or 2 concrete slabs are placed on the top of the shafts. Furthermore, integrated ventilation pipes piercing through the concrete cover as well as compulsory cuts through the shaft lining for testing the competency of the surrounding bedrock that supports the concrete slabs, will in any case allow for some mine water to escape where the water level rises high enough.

9 Predicting the final mine water elevation (decant level)

9.1 Factors controlling mine water levels

As pointed out earlier, the assumed ability of a single outflow point such as a low-lying shaft to control the elevation of the final water table in the completely flooded mine void system, requires that all sub-voids making up the larger mine void system (‘basin’) are somehow hydraulically connected. A second requirement is that the hydraulic conductivity of all connecting links between the different sub-voids is large enough to accommodate the entire ingress volume (which includes the laterally imported water from the adjacent sub-void) the upstream sub-void receives. Where a link between two neighbouring sub-voids is too small to accommodate all of the water received by the upstream void will result in a build up of a hydraulic head in the upstream void. This, in turn, would result in different water levels across the basin and limiting the role a low-lying outflow point could play in controlling the FMWL. It is therefore crucial to establish whether, and if so to what extent, the different sub-voids are hydraulically interconnected. This will mainly be done by analysing historic water levels in the different sub-voids, aided by information on pumping volumes and schemes at different times, as well as on structures such as haulages, holings etc. connecting the different sub-voids. It will also require estimating the volume of water recharging the mine void (ingress volume) during the process of filling up, as well as identifying the various sources of this water (ingress sources). A first order apportionment of the contribution of the different sources may aid the estimation of future decant volumes, as some of the sources may no longer contribute to the ingress once the mine water level reached the final elevation. As this holds consequences for the volume of water flowing from the completely flooded mine void (decant volume) it also needs to be addressed. Since the various ingress sources differ regarding the associated pathways and temporal patterns of their ingress contribution, exploring the ingress dynamics of the different sources may assists with quantifying their respective contributions to the recharge of the different sub-voids.

Possible future changes such as the collapse of existing links and their effects on the FMWL also need to be considered. Based on the above, the following factors were identified as the main controls of water levels in the mine void.

(1) Characteristics of the mining basin/ void system (geometry, shape, volume and structural links between sub-voids, vertical profile)

(2) Ingress rates (volumes), sources, pathways and dynamics

(3) Hydraulic interconnectivity between different sub-voids of the basin

(4) Pumping of water from the mine void (rates/volumes, levels)

Each factor will be discussed separately in the following sections.

9.2 Characteristics of the mine void system

9.2.1 Basic concepts

In the following the term ‘mining basin’ is defined as a system of underground mine workings that are connected to surface by vertical shafts. It consists of a number of individual mining voids (‘sub-voids’) that have been created by different gold mining companies located in a common goldfield (in this case the Central Rand). The lateral extent of each sub-void is defined by lease area boundaries on surface projected vertically underground. Many sub-voids are somehow interconnected through various underground structures such as tunnels allowing for lateral water flow between sub-voids. However, not all adjacent sub-voids have necessarily been hydraulically linked and in some cases existing connections have been sealed using watertight concrete fillings (‘plugs’). The degree to which water ingressing into the sub-voids can move across lateral boundaries is termed ‘hydraulic interconnectivity’.

9.2.2 Geometry and shape of the mine void

Since much of the hydraulic properties of the mine void is inaccessible to direct measurements various proxies have to be used including the rate of rise of the mine water table. This rate, however, is controlled by a number of factors of which one is the vertical volume distribution along the depth profile of the mine void. This vertical volume distribution is determined by the geometry/ shape of the mine void. E.g. a cone-shaped void which is wide at the bottom (large volume) and gets increasingly smaller towards the top (surface) will – when being flooded with a constant volume of ingress water - display an ever increasing rate of rise as the water table approaches the surface. In order to assess such effects on the mine water table dynamics the geometric shape of the mine void is analysed.

In essence, mining in the CR removed a number of relatively thin layers (10-50cm thick) of tabular Au-bearing material (‘reefs’[2]) sandwiched between hard rock. In contrast to other metal mines which rely on veins, gold mines in the Witwatersrand’s basin mined so

[2] While the auriferous ore bodies are generally referred to as ‘reefs’ this term is not without ambiguity. For instance, in terms of stratigraphy, reference is made to the ‘Main Reef’ (indicating a total thickness of 25m). In terms of mining, however, the ‘Main Reef’ is only a 1-6m thick ore body that – together with 6 other ore bodies which are also called ‘reefs’ make up the stratigraphic unit of the Main Reef. Sometime these sub-reefs are referred to as ‘reef bands’. This, however, is also not done consistently leading to confusion as to what type of reef concept is referred to. In this report the term ‘reef series’ is used if an ore body consists of more than one reef band. Alternatively the term ‘reef package’ can be found in literature.

called placer deposits, i.e. Au that was deposited along with other sediments on floodplains below an ancient mountain belt, bordering on an inland sea . Today's gold mining essentially takes place at the some 300km-long, crescent-shaped shore line in the NE-SW part of this ancient inland sea. This part is sometimes also referred as 'Golden Arc' and consists of 7 x different goldfields.

This in essence flat deposit was subsequently buried by other sediments and tilted to give it its present average of 45 degree dip, varying between 80 and 20 degrees Given this dip a mining progression of only 1-3 km on surface results in a drastically increased mining depth, rendering South African gold mines the world's deepest. For hard rock such as quartzite in which most South African gold mines are operating, the breaking depth is at around 2000 m below surface. At deeper mining levels the pressure exerted by the overlying 2 km-high rock column exceeds the strength of the rock (Malan, 1999). Despite this obstacle to mining, it is estimated that in 2010 some 30% of the South African gold mines operate below 3000 m (Malan, 1999). In the Central Rand the deepest mining level is reached at ERPM with some 3400 m below surface. This unprecedented extension of mining depth in South Africa is only possible (apart from engineering ingenuity), due to 2 x geological factors namely the tabular and continuous nature of the mined gold reefs, and an exceptionally low geothermal gradient. In contrast to metals mined from irregular isolated veins randomly distributed in the bedrock, the generally tabular shape and continuous nature of the gold reefs allows to reduce the amount of waste rock that has to be removed to a minimum and thus to mine economically at these depths. A consequence for this project is that the resulting mine void has a relatively small proportion of 'off-reef development' (waste rock removed to access the ore body). This, in turn , is of significance for the final volume of the resulting mine void as underground infrastructure to access the ore lasts much longer than the short-lived workings (stopes) which tend to close over time reducing the volume of the mine void and its hydraulic properties.

The second factor that aided South African gold mines to reach record depths in the Witwatersrand is a geothermal gradient which is comparably low to other parts of the world, being situated on a relative old and thick continental crust that reduces the impacts of heat emanating from magma in the earth mantle (10°C/km compared to 30°C/km elsewhere). As the core of an ancient continent, this thick crust is termed a craton (in this case the Kaapvaal craton). This too has consequences for the project, as high geothermal gradients could drive thermal convection in a flooded mine void, resulting in mixing of deep and shallow mine water, with associated consequences for the water quality of the decanting mine water. Furthermore, especially the inner centres of old cratons are regarded as geologically stable regions, with little exposure to tectonically induced large-scale and often catastrophic seismicity.

The overall shape of an underground reef is perhaps best illustrated by imagining the reef being a sheet of paper (i.e. very thin in comparison to length and width) inserted at landscape format with at a 45° angle into a sand pit, where the sand represents the host rock. At these dimension the miners would approximately be the size of ants. At greater depths the sheet somewhat bends (reducing the dipping angle) resulting in a vertical reef profile that resembles the run-up of a ski-jump tower.

In case of the Central Rand several sheets of papers are stacked behind each other separated by layers of hard rock varying from tens of metres in thickness to several kilometres. In none of the other reef series has a reef band been as extensively mined as the MRL and the SR in the MR series. In fact, the other reef packages have only been sporadically mined resulting in much less void space created to the south of the MR void 'sheet'.

The upper part of the sheet (reef) that intersects the surface is termed 'outcrop'. In the CR this outcrop runs over 44 km (termed 'strike length') from Roodepoort in the west via central JHB to Boksburg in the east. While the W-E running outcrop represents the length of the reef, the sheet width equals the depth to which the reef is mined. In the CR this varies from 1900 to 3400 m below surface with an average depth of an est. 2800 mbs (calculated based on Scott 1995: 121 depths of sub-voids weighted according to their proportion of the total strike length). A total of 6 x different reef packages of the Central Rand Group run nearly parallel in a N-S sequence over a distance of approx. 5 km in the following outcrop sequence from N to S (in stratigraphical order this equals the sequence of deposition from the lowest level, i.e. oldest sediments, to the highest, i.e. youngest ones) separated by hard rock (mainly quartzite):

- Main Reef (MR) series (consisting of 7 x reef bands with a total thickness of 25 m)
- Johnstone Reef (25 m thick, never mined)
- Livingstone Reef series (25 bands totalling 20 m thickness, never mined)
- Bird Reef series (consisting of 3 to 5 reef bands each up to 0.5 m thick totalling a maximum of 2.5 m)
- Kimberley Reef (300 m thickness)
- Elsburg Reef (500 m thickness, rarely mined in the CR)

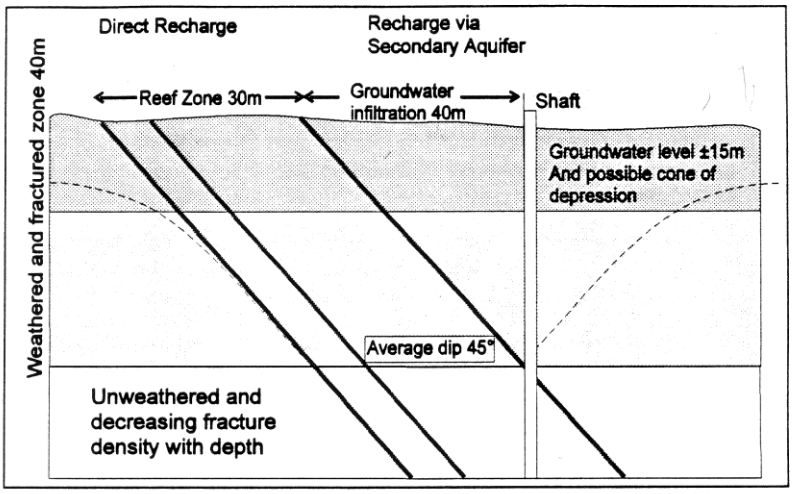

Fig. 9.1: N.S Cross section through the Main Reef outcrop zone indicating the ingress of groundwater from the fractured aquifer into the mine void (adopted from Scott 1995)

The 7 x seven reef bands of the MR series vary in thickness and Au contents and not all of them have been mined (Tab. 9.1)

Tab. 9.1: Reef bands mined in sequence of the outcrop (N to S)

no	**Reef band**	**Reef series**	**Thickness of reef band [m]***	**Occurs at which gold mine**	**Where mined**
1	North Reef (NR)	Main	0.15–0.45	DRD-Rose Deep	sporadic
2	Main Reef (MR)	Main	1-6		DRD and ERPM
3	Bastard Reef	Main	?	S&J to ERPM	?
4	**Main Reef Leader** (MRL)	Main	**0.5**-2	DRD-ERPM	Rand Leases to S&J – **most important reef** - initially mined from surface
5	Middle Reef (MiR)	Main	0.76	Crown Mines to Witwatersrand GM	Village Main Reef
6	**South Reef** (SR) (15-60m above MR)	Main	**0.35**-3	DRD to ERPM	DRD – ERPM **Second most important reef** - initially mined from surface
7	South South Reef (SSR)	Main		City Deep – Witwaterrand GM	only at S&J
8	Livingstone Reef (25 x bands)	Livingstone	0,9	DRD-ERPM	Wolhuter mine
9	White Reef / Monarch Reef	Bird	0.5	DRD-ERPM	Rand Leases to Crown Mines
10	Kimberley (4 x bands)	Kimberley		DRD-ERPM	DRD to Crown Mines; ERPM
11	Elsburg Reef	Elsburg		DRD-ERPM	sporadic

* The reef thickness indicated is somewhat misleading if used to estimate the height of stopes used to extract the ore as in, some instances, 3-6m high stopes would have to be used. It is important to note that the geological concept of a 'reef' given here is somewhat different from the 'reef' concept used in a mining sense. While the former refers to the geological stratum such as a conglomerate band in which the metal of interest is generally accumulated, the latter only refers to that part of a reef where the Au contents is high enough to warrant (rather expensive) extraction. In most gold mines this is only a small part of the geological reef commonly in the range of 20 to 50cm. Therefore, stoping heights in most deep level gold mines are not much higher than 1.1.-1.4m, just high enough to allow (crouching) miners accessing the often much thinner reef.

After the first reefs were discovered on the farm Langlaagte in 1886, initial mining of the outcropping reefs was first done from surface often by hand digging. With Au grades of 36 g/t in the oxidised shallow ore bodies, yields were up to 7 times higher than encountered by deep level mines today, resulting in a rapid spread of mining activities along the entire Witwatersrand. Within a year of the Au discovery a new gold field opened in the West Rand. Early mining in the CR concentrated on the reef outcrop zone and accessing reefs by removing the overburden through digging pits and trenches, sometimes employing incline shafts to access shallow underground workings. During this period of shallow surface mining, much of the outcrop area was disturbed. However, accurate records on the true lateral extent of the surface mining zone along the MR outcrop are not available. As a first order approximation it is assumed that at least 80% of the total strike length of 44 km (i.e. some 35 km) have been affected by surface mining, creating a much enlarged mine void entrance compared to the underlying mine void.

While on surface it is still economical to remove the 30m thick layer of waste rock between the two major reef bands of the MR series, namely the Main Reef Leader and the South Reef, this is prohibitively expensive in deep level mining. Fig. 9.2 illustrates how the initial surface mining was conducted using open trenches as well as shallow incline shafts.

Fig. 9.2: Early surface digging at MR outcrop illustrating the disturbance of natural surface conditions, top: Shallow incline shafts at MR outcrop (Photos: top left: Antrobus, 1986; bottom: Editorial Committee, 1986; top right: Mendelsohn and Potgieter, 1986)

An example for S & J is given in Scott (1995) where the oxidised ore has been mined by Anglo American in an open cast trench for a kilometre along strike. After having been dormant for many years the western and later the eastern portions have been filled with municipal waste. Currently a similar method is used by the Central Rand Gold (CRG) mine which operates open pits just south of the CBD (near the Top Star slimes dam (Labuschagne, 2011: personal communication).

With similar surface diggings taking place along the strike length of the outcropping MR package a 35-40 km-long highly disturbed zone was created where mining and subsequent filling with unconsolidated material such as sand, rubble, municipal waste etc. allows for high infiltration rates. Following the outcrop of the Main Reef Leader and the South Reef (which are separated by some 30 m of quartzite) for the Main Reef this zone is reportedly about 30 m deep and some 70 m wide. Although not explicitly mentioned it is inferred from information provided in Scott (1995) that the same outcrop mining affected the other 2 x reef packages, i.e. the Bird and Kimberley reefs. However, owing the lesser extent of mineable ore in these reef packages Scott (1995) assumed that the extent of surface mining (i.e. the width of the disturbed zone) for the Bird Reef is about half the width of the MR and for the Kimberley Reef about half the width of the Bird Reef zone.

With mining residues deposited in between these reefs and most of the mining infrastructure located along the outcrops, an elongated 2-5 km-wide area developed which remained largely free from any urbanisation owing to mining being the dominant land user. This elongated area is termed ‘mining belt’ in this report. The location of the mining belt is shown in Fig. 9.3.

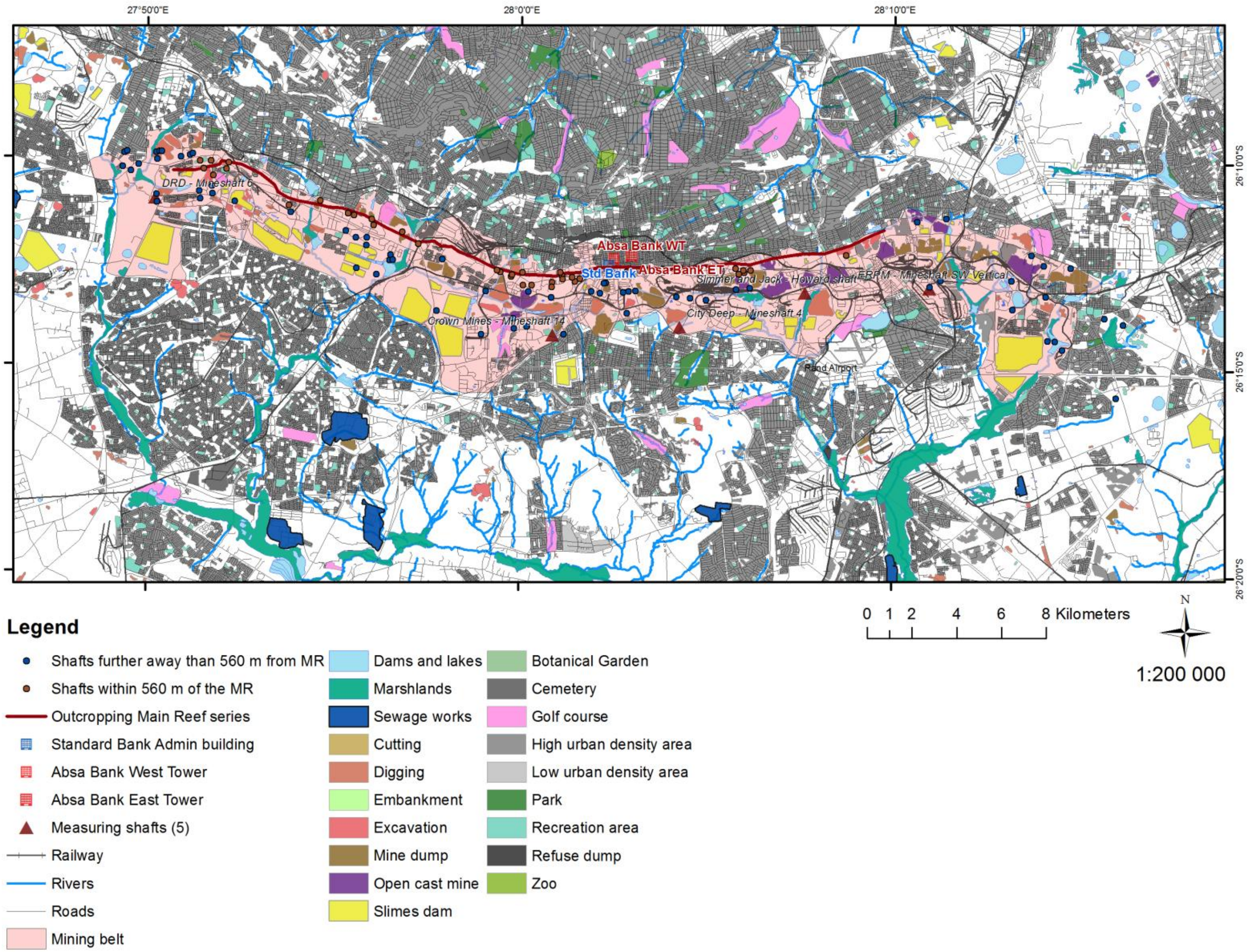

Fig. 9.3: Map of the study area indicating the location of the mining belt and associated land uses within the highly urbanised metropolitan area of Johannesburg

Each of the reef-sheet displays an (underground) surface area of some 132 km² over the mined depth. Since all reefs continue below the deepest mining level the CR still hosts considerable Au-reserves. However, under the conditions in the early 1970s when most mines closed, these were too deep to be mined economically. As mentioned before, aided by a much improved gold price DRD in recent years explored in its Argonaut project the feasibility of mining these deep reefs via ultra-deep mining. After mining rights have apparently recently expired the Argonaut project may have been meanwhile abandoned.

The 3 x most important reef packages gradually approach each other from W to E and finally merge at ERPM into a single ore body called the ‘composite reef’. With a dipping angle of 25° (Labuschagne, pers. comm. 27.4.2011) this reef package dips less steep than reefs elsewhere in the CR. As this increases the rock pressure on the reef (as it will be discussed later in the report) it is likely that stope closure at ERPM, which is also the deepest mine in the CR, is particularly pronounced.

With the thickness of the mineable part of the reef ranging from 20 to 60 cm the combined width of all 6 x mined reefs ranges from 1.2 m to 3.6 m. Assuming an average combined reef thickness of 2 m (as not all reefs have been mined everywhere) results in a total volume of the 6 x reefs of approx. 246 mio. m³ (= 2800 m av depth x 44,000 m strike length x 2 m av. combined reef thickness).

Since the removal of non Au-bearing rock (‘waste rock’) from great depths is costly all mines minimise waste rock generation and confine the underground development as much as possible to the paying reefs. Therefore, the overall shape of the resulting mine void largely resembles the sheet-like, tabular shape of the reefs, where the thickness of the sheet height is extremely small compared to the lateral extent of the mine ore body, which can span several square kilometres. The total mined-out volume, however, is necessarily somewhat larger than that of the reef, as structures are needed to access the reef (‘off-reef development’). For underground metal mines in the USA a ratio of milled ore to waste rock of 64 mio. t to 3 mio. t is given, suggesting off-reef development accounting for less than 5% of the total mine void. For deep level gold mines in the KOSH area a value of 10% has been suggested, while Scott (1995) assumes off-reef tonnage constituting some 15% of the reef tonnage (= tons of milled ore) indicating as well that the latter may generally be somewhat overstated as occasionally also waste rock is milled. A reduction of the void volume is associated with pumping tailings back into the mined out void (‘backfilling’), as well as the input of supporting material such as wood and concrete. For the latter alone Dierling (2000) estimates some 1000 m³/month (12 Ml/a). Compared to the average tailings production (in this case of ERPM between 1998 and 2008: 0.6 mio m³) this amounts to approx. 2% of the removed ore/ stoping volume.

Notwithstanding the differences, all figures illustrate that the overwhelming majority of the mine void (85-95%) is created by removing the (paying) ore. This is important to realise as the areas where the ore is removed (termed 'stoping area') are often only temporary in nature and may – depending on the mining depth amongst others – soon close after the ore has been removed. In view of the fact that these areas account for nearly all of the void volume it is important to understand to which extent and over what period of time stoping areas indeed close. While stope closure in principal has been mentioned before, we did not encounter a single reference so far attempting to quantify the associated reduction in the mine void volume or to explore impacts on hydraulic properties and chemical processes such as AMD generation. This will be provided in this report.

Structures developed for accessing the underground ore bodies include vertical elements such as shafts that provide the principal access to the deep reefs from surface as well as horizontal elements linking shafts to the areas where the reef is excavated (stoping areas). Compared to stoping areas horizontal access structures such as haulages, tunnels, cross cuts drives and box holes are needed for much longer periods of time, and therefore more permanent in nature. Especially the haulages are commonly designed for a life span of 30-40 a (Diering, 2000). With heights of 3.5 m or more, and widths of 4-10 m depending on the purpose, the stability of lateral access structures (tunnels) is ensured by reinforced support, and larger haulages are often cemented all around to prevent rock falls (using different techniques including the use of 3m-long tensioned anchors, mesh and lace covers around the hanging wall, as well as different types of concrete covers such as 'shotcrete' and 'fibrecrete', Diering, 2000). The basic components of an underground mine infrastructure is depicted in Fig. 9.4.

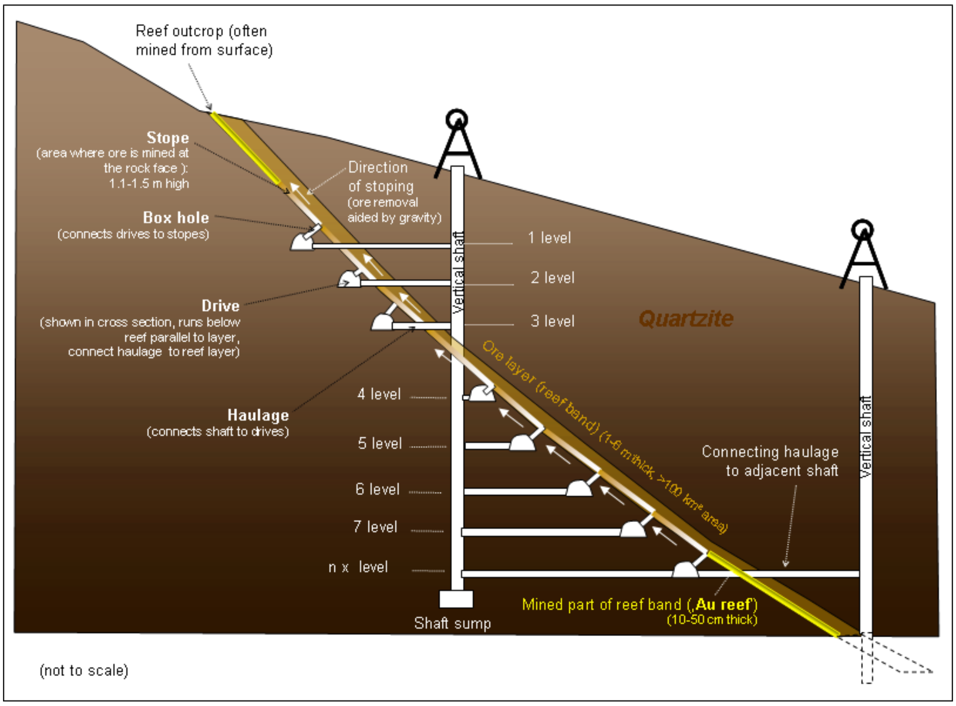

Fig. 9.4: Sketch illustrating the main vertical and lateral components of the underground mine infrastructure

As mentioned before, the mine infrastructure situation is very different to the stopes which are only supported by wood or cement-/ tailings-filled bags and tend to close soon after active excavation stops. Stoping areas are generally designed to be just high enough for mine workers to access the rock face with the needed drilling equipment. Stoping heights in modern mines range from 1.1-1.4 m (average for the SA gold mining industry in 1980: 1.33 m, Malan, 1999) increasing to up to 4 m where reefs widen (so called ‘pay streaks’). In older mines smaller stoping heights of only 50 cm are possibly to be found having forced workers to leopard crawl to the rock face.

Waste rock removal in deep level mining needs to be minimised in order to remain economically viable. Based on the fact that 90% of the material removed from underground originates from the ca. 1 m-wide stoping areas through which 2 to 6 x tabular ore bodies have been mined, the average widths of the overwhelming majority of the mine void is only around 2 to 6 m. This compares to a width of 50 to 75 m of the MR outcrop zone mined on surface. Thus, should the water table recover up to the level of this shallow surface mine zone a slow down in the rate of rise by a factor of approx. 10 to 30 (= 50-75m : 2-6m) is to be expected.

9.2.3 Volume of the mine void

(i) Purpose of determining the void volume

The determination of the approximate size (volume) of the mine void is needed for assisting with estimations of the time it will take for the mine void to flood (based on estimation of the ingress volumes) as well as the underground pollution potential associated with the exposure of ingressing water to unmined pyrite-containing ore, ore stockpiles, etc. Taking depth- and time-depended changes in the volume of excavations into account will also allow to better the understanding of changing hydraulic properties along the vertical mine void profile related to time.

Since the exact volume of the underground excavations cannot be directly measured given the lack of relevant data, a number of proxies are used to arrive at approximate estimations. This includes the measuring of mined out areas depicted in shareholder plans, and multiplying this with an (estimated) average stoping height. Owing to the fact that many mines in the Central Rand are no longer existing, and shareholder plans are only available for some of the mines, this methods is prone to underestimate the true extent of the void volume. Having applied this method to the WR basin and acknowledging its potential for underestimation, Krantz et al. (1999) arrived at 50 mio. m^3 total void volume compared to 125 mio. m^3 estimated by Winde et. al (2006) and 135 mio. m^3 by Usher & Scott (1999).

The latter two studies estimated the mined out volume by the tonnage of milled ore reported to the chamber of mines. Applying a specific gravity of 2.65 t/m^3 for the mined reef (sometimes also 2.75 t/m^3 is used) allows to calculate the volume of the resulting mine void. As mentioned earlier, the volume of non-ore rock has to be added that needs to be excavated to access the actual ore body via shafts, haulages, drives, cross cuts and box holes. For deep level gold mines in the Central Rand this off-reef development is assumed to be comparable to other goldfields in SA, amounting to approximately 10%-15% of the milled tonnage. However, taking partial backfilling of the mine void with tailings (differs between sub-voids, difficult to quantify) into account, as well as the reduction in void volume associated with the support material (2% of milled ore) a general volume of removed non-ore rock of 10% of the mined ore volume is assumed.

Based on values of milled ore for mines in the Central Rand for 1985 (provided in Scott, 1995) and for 1998 (Whymer, 1999) as well as interpolations between the two dates to estimate the production of ERPM as the only mine still producing after 1998, a total mine void volume of 467 million m^3 was calculated for the CB (Fig. 9.5).

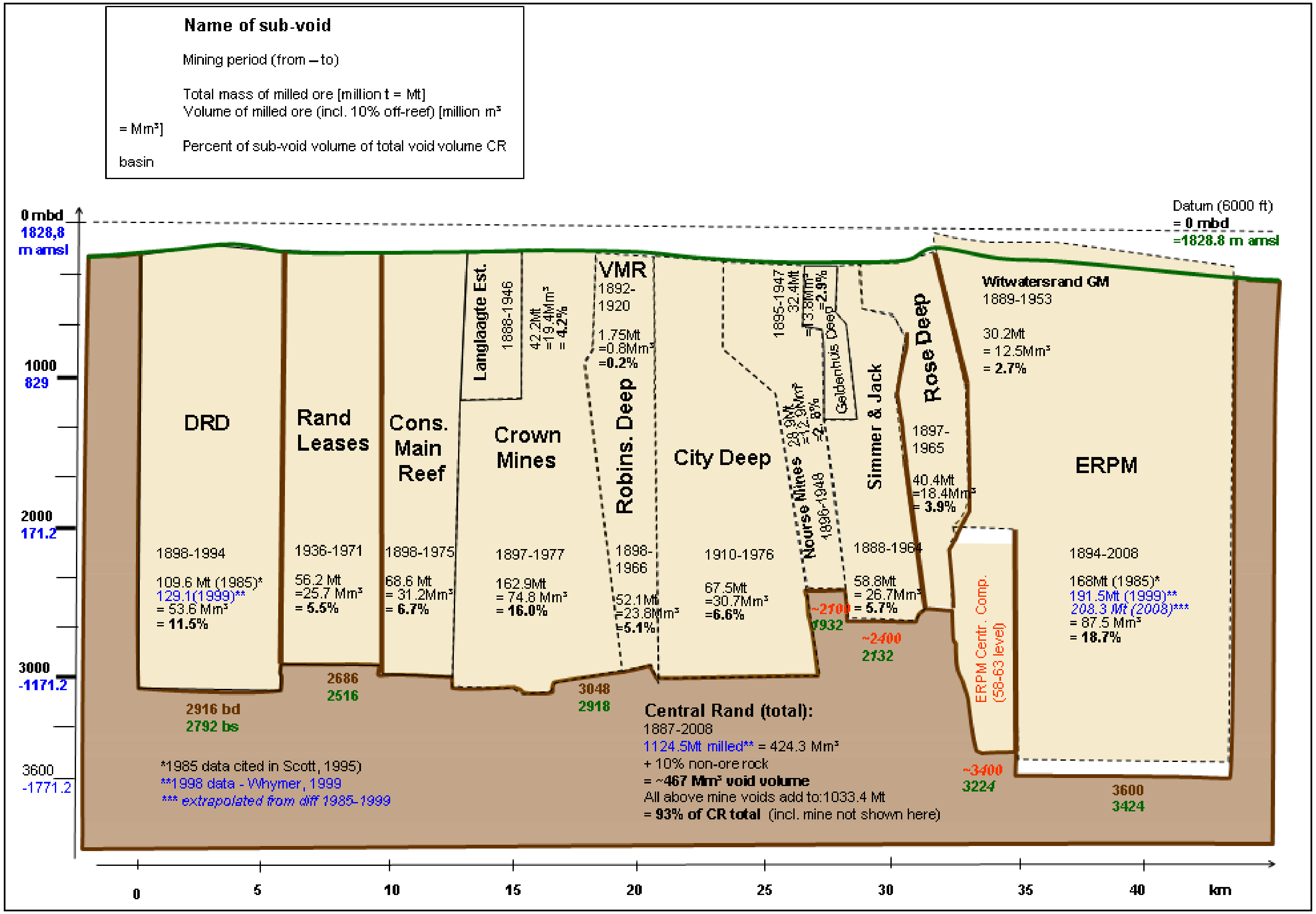

Fig. 9.5: Estimated volumes of the different sub-voids as well as the total mine-void system (basin) calculated based on tonnages of milled ore and an off-reef development of 10% of the mined ore volume (original data from sources indicated inside of the figure.)

This contrasts with Scott (1995) who employed both methods and arrived at a total volume of 224.5 million m³ using map-based calculations (assuming an average stope height of only 0.5 m and a mined out area of 163.3 km²) and 392.9 million m³ respectively based on milled tonnage. Both volumes are significantly lower than our result.

Adjusting the stope height used by Scott (1995) from the very low value of 0.5 m to a more realistic value of 1 m (as the average between the smaller older stopes and the average height of 1.33 m of later stopes) the Scott volume based on maps would change to 449 million m³ which is in good agreement with our calculation (4% deviation). In comparison, Boer et. al. (2004) quote a void volume for the CR of 587.8 million m³ almost a quarter larger than the Scott and our estimates, and more than double the original map-based volume estimates given in Scott (1995). Since Scott used an unrealistically low stoping height that – if corrected - results in a very similar volume to ours, which in turn lies halfway between Scott's tonnage-based estimate and the volume given in Boer et al. (2004) we are generally confident about the approximate correctness of the calculated volume.

Using tonnages milled as a base for void calculations incorporates a number of uncertainties, of which the most important one is that these tonnages are compiled only for mines that are member of the Chamber of Mines. Since tonnages milled by non-member mines are thus omitted, a certain potential for underestimating the true tonnage exists. This error, however, is relatively small as most mines have been members of the Chamber, and is additionally balanced by over-reporting milled tonnages by member mines as pointed out by Scott (1994). This over-reporting relates to the fact that surface plants are often designed to run at full capacity. As a consequence, during lower production periods caused by strikes, accidents, holiday periods etc. waste rock from surface dumps was apparently run through the mills to avoid switching off machinery and keep plants running at optimal levels. Also, over-reporting caused by mining non-ore areas stimulated by bonus incentives may have occurred although this would have the effect of increasing the mine void. A further factor that may result in the reduction of volumes calculated based on milled tonnage is the practice in older mines to pack waste rock into underground workings for additional support (Scott, 1995). Since this only applies to some of the older mines where it affects only a fraction of the 10% void volume associated with waste rock (off-reef development), the impact on void overestimation is regarded as marginal.

Since these factors are counteracting the possible underestimation due to the omission of non-members of the CoM, the tonnage-based void calculations appears to be comparatively reliable.

(ii) Estimating depth- and time-depended reduction of the void volume

While in some reports the possibility is mentioned that the mine void volume may be reduced over time due to the collapse of stopes (although opposing views are also advanced that – not aware of the phenomenon of plastic rock creep also occurring in hard rock – argue that the total void volume will remain constant as brittle fracturing and collapse will only redistribute void space but not reduce the total void volume), not a single study has been found that attempted to scientifically support this assumption by the latest insights from rock mechanics, leave alone to quantify the degree to which this happens. Such an attempt will be provided in the following section (although it has to be borne in mind that this was done via a limited desktop exercise).

Compared to other underground mines world wide South African gold mines are in a class of their own regarding to the depths at which they operate. This may explain that the behaviour of rock conditions found at these ultra deep mines is not well understood. This is particular true for the plastic behaviour (flow type of deformation) of hard rock such as quartzite (termed 'creep') which, according to Malan et al. (1997), was until recently regarded as only applicable to soft rocks like halite and potash. Malan et al., in 1997 stated that no systematic study of the creep of hard rock has ever been conducted in the South African mining industry probably due to the long duration of the required tests and a lack of funding. However, measurements in several deep level gold mines including the Central Rand, indicate that plastic closure of stopes (in contrast to brittle rock failure associated with fracturing, scaling and finally collapse of tunnels) is taking place in deep level gold mines of the Witwatersrand (below 2000 m) at rates of up to 3cm per day depending on the mined reef and the mining depth (Malan, 1999). While rapid closure is confined to the first few weeks after blasting, long-term, creep continues thereafter. Stopes are frequently reduced to 'mere pencil lines' (Brouwer, 2003) with timber pieces sticking out as the only reminder. With the hangingwall (roof of the stope) and the footwall (floor of the stope) completely moved together, such stopes can for all practical purposes be regarded as water tight as no obvious fractures are visible in the few exposures where this phenomena was observed.

Since creep is induced by the vertical rock pressure constantly exerted from the overlying rock column its intensity is a function of mining depth. While according to Brummer (1987), rock pressure problems start to occur at mining depths of 300 m below surface, Scott (1995) reports for DRD and ERPM that no closure has been observed above 700 mbs despite wooden supports being completely friable. Scott further states for ERPM that little closure occurs above 1500 m amsl while stopes at 3200 mbs close totally within 5 months. For reefs mined at depths below 1800 m, Scott (1995) cites observations by local geologists that 'total collapse' has taken place. Based on these citations the depths of 1500 mbs is taken as a threshold from where on complete closure of stopes occur, caused by the creep-like bulk expansion of highly stressed crystalline rock.

The closure rate not only depends on the mining depth, but also on the dip angle of the mined reef. The reefs are less exposed to vertical stress the closer the dipping angle is to 90°, at which point the reef would be completely vertical and not exposed to additional pressure from an overlying rock column. However, flatter dip angles at depth results in a higher exposure to the vertical rock pressure. Since most reefs mined in the Central Rand start with a rather steep dip angle close to surface of up to 80° in certain areas, on average 45°, and flatten out at depth (to some 20-25°), stope closure in the deeper parts of the Central Rand mine void is presumably particularly pronounced due to the combined effect of an increasing rock column and higher exposure to the resulting rock pressure.

In this context it is important to distinguish between the plastic stope closure and the (non-plastic) collapse of horizontal and (to a lesser degree) vertical underground infrastructures such as tunnels and shafts, after exceeding their designed life span of 30 to 40 a. The collapse of haulages, drives, cross cuts, box shots etc. designed to access the stoping area does not result in a significant overall reduction of the mine void volume, but merely in a re-distribution of available pore space, as fallings rocks that close the passages, for example, free up space in the hangingwall. While no overall reduction in volume is associated with the non-plastic closure of the more permanent underground infrastructure, it may change the hydraulic properties of affected tunnels.

The filling of conduits with rubble, fine material and large rocks may change the rapid and free flow in open haulages to a much slower flow through fractures of different sizes and porous fill material. In contrast to this non plastic mine void deformation, it is proposed that the plastic closure of stopes is mainly caused by the stress-induced bulk expansion of a solid rock mass, until the mining gap is completely filled. Being plastic in nature, this filling is not associated with the opening of new spaces somewhere else (e.g. in form of brittle fracturing), and thus truly reduces the volume of the created mine void through bulk expansion[3].

Given the closure rates mentioned above, some stopes may be completely closed only a few months after being mined. This means that from a certain critical mining depth onwards, much of the created void disappears, while new void space is created below

[3] This assumption is made following a brief review of topic related rock mechanic articles. Since the mid 1990's plastic closure mechanisms were included in stope closure equations as it fitted the observed closure better. However, as rock mechanics is primarily involved with mine safety issues, none of these equations or observations extended to more than a few months after mining ceased in a certain area. As far as the author's knowledge extends, only one drive is known to date that has cut through an already stoped and closed area, indicating full plastic closure without any discernable tension (open) fractures associated with it (Brouwer, 2003). Post mining stope closures rates vary tremendously, pending on the geological setting of the stope, but appears to peak at 30mm per day (Malan 1999) during the initial stope closure phases, there-after gradually relaxing) All relevant literature sources implied indirectly that plastic stope closure could play a major role some time after mining ceased in a certain area.

via new mine out areas. That would mean that at no stage during the life of a deep level mine, the total void volume created fully exists. After mine closure, when no new development takes place, gradual closure of stopes – perhaps even at less deep mine levels - results in a continuous reduction of the void volume. It is assumed that this reduction would gradually become less intense and asymptotically approach a final value. In what time after mine closure this final value will be reached is uncertain. For practical purpose it is assumed that at 1 x year after mine closure, all stopes in the deeper parts of the mine void (below 1500 mbs) are completely closed as somewhat higher lying stopes had already time to close while mining was still ongoing, leaving 12 months for the deepest lying stopes (i.e. those exposed to the highest rock pressure) to close.

Since mined-out areas are cordoned off as soon as mining stops, and declared unsafe, access to these areas are restricted. This, together with the fact that mined out areas are of little economic interest to mines, may explain why no systematic, long-term studies on the extent of plastic stope closure have been conducted to date (as far as the researchers could ascertain). Considering that stoping areas account for the overwhelming majority of the mine void and make up some 90% of the total underground volume, their complete closure in deeper parts of the mine void is likely to result in a significant reduction of the total mine void volume over time.

As a first order approximation the reduction in mine void volume can be calculated. For this it is assumed that the average gold mine in the CR is 3000 m deep, and that all stopes below 1500 mbs (i.e. half of the void) will close completely. Below 1500 mbs only the semi-permanent structures such as the shafts and the horizontal tunnel system would stay open as void space, accounting for only 10% of the originally removed ore volume. For stopes above **700** mbs no plastic closure at all is assumed leaving the totality of the mine void (100%) intact. For the depth between 700 mbs and 1500 mbs, partial stope closure is assumed. For simplicity reasons the average between the void part above (0% volume reduction) and the void part below (90% volume reduction) is used resulting in a 45% volume reduction i.e. 55% of the original volume remain.

Since the volume distribution over the depth profile of the void was found to be largely homogenous the actual remaining void volume for the entire depth of the void can now be calculated.

In a first step the percentage each void zone makes up of the total void depth is determined:

(1) upper zone: 0-700 mbs: *23%*
middle zone: 700-1500 mbs: 27*%*
lower zone: 1500-3000 mbs: *50%*.

In a second step, the proportion each zone of different void reduction occupies from the total void is multiplied with the corresponding percentage of the remaining void volume in the respective zone:

(2) upper zone: *23%* of total void x 100% remaining volume = **23%** of total remaining void volume
middle zone: *27%* of total void x 55% remaining void volume = **15%** of total remaining void volume
lower zone: *50%* of total void x 10% remaining void volume = **5%** of total remaining void volume

The last step is to add up the percentages of the total remaining void volume of all the 3 zones in order to arrive at the total remaining volume for the entire void from 0 to 3000 mbs.

(3) upper zone: **23%** of total remaining void volume
\+ middle zone: **15%** of total remaining void volume
\+ lower zone: **5%** of total remaining void volume
= 43% remaining void volume in entire mine void

This implies that plastic stope closure would have reduced the original mine void volume by 57%.

The 3-zone model on closure related void volume reduction in an average Central Rand gold mine and associated calculations are depicted in Fig. 9.6.

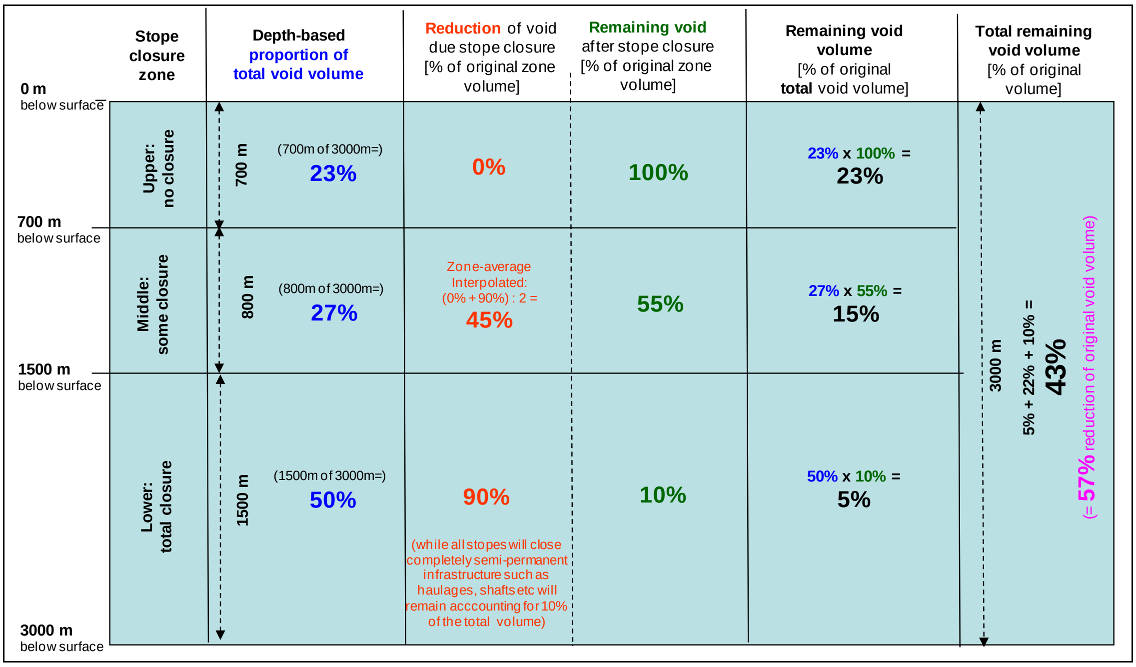

Fig. 9.6: Generic model of a deep level mine void depicting how different degrees of plastic stope closure result in different reductions of the mine void volume depending on mining depths (3-zone model)

According to the above calculation it is estimated that less than half (43%) of the originally created void volume will remain due to the bulk expansion of highly stressed rock and associated closure of stopes. As this is only a first order approximation largely based on theoretical models and assumptions an attempt was made to verify the calculation by using data from real world examples. For this purpose a case study of a mine void in the West Rand was used that has already been flooded. Based on calculated void volume and measured decant flow rates as proxy indicators the reduction of the original void volume was calculated. The same approach was applied to a part in the Central sub-void of the Central Rand that has been flooded in the mind 1970s. Both cases studies are discussed below.

Case study 1: West Rand mine void (flooded 1998-2002)

After deep-level mining stopped in the West Rand in 1998 the mine void filled up within a span of approximately 4 years starting to overflow in 2002 (first observed in September).

Since then mine water is decanting from the flooded void via the so-called black reef incline (BRI) shaft at an average rate (2002–2011) of some 20 Ml/d. Krantz et al. (1999) estimates that some 11.3% of the mine void are not flooded as this part is located above the decant level.

Based on milled tonnages reported by the Chamber of Mines for the mines in the WR (Whymer, 1999) Winde (2006) calculated a mine void volume of approximately 125 million m³ (including 10% off-reef development). This compares favourably with calculations of Usher & Scott (1999) who, using the same method, estimated the total void volume being 135 million m³. As mentioned earlier, the estimate of Krantz et al. (1999) of 50 million m³ is, by own admission, based on incomplete mining plans and has a considerable potential for underestimating the true volume. It is therefore omitted from consideration in this report.

Using a total mine void volume of 130 Mm³ (average from Usher & Scott, 1999 and Winde, 2006) and a time of exactly 4 years (it might have been some months less or more) during which the void was filled results in an average hypothetical flow rate of 89 Ml/d required to achieve filling within this span of time. This hypothetical flow rate is, however, some 3-4 times higher than the actually observed decant volume observed over the past 9 years or so.

Based on the widely applied assumption that the flow rate of the decanting mine water is approximately equal to the flow rate of water that filled the mine void in the first place (i.e. ingress volume = decant volume) the filling of the mine void in 4 years can only be explained if a much smaller void volume was to be filled. Based on an average ingress volume of 20 Ml/d during the 4 year long flooding period a total void volume of some 30 million m³ could have been filled. Compared to the original void volume (130 Mm³) this represents only a quarter (23%) implying that the original volume had been reduced by 77%.

Considering that an estimated 11% of the mine void lies above the decant level and is therefore not be flooded results in an associated void reduction of **74%**. This is an even larger reduction than suggested by the generic 3-Zone model and may suggest that plastic stope closure is perhaps more intense above the 1500 m-zone than suggested by Scott (1995).

Case study 2: Central Rand sub-basin (partial flooding 1974 to 1976)

Scott (1995) reports that the deepest part of the central sub-basin (at the time consisting of 10 x sub-voids stretching from Cons. Main Reef in the west over some 25 km to Rose Deep in the east) was allowed to flood after active mining at this depth stopped at Crown Mines in 1974. Subsequently, in February 1975, the pumping station at Rose Deep (SWV shaft) was lifted from 49 level (at 2058 mbs) to 24 level (at 1253 mbs). From there pumping resumed in October 1976. During the 20-months period in between (2/1975-10/1976) the mine water level rose by 805 m (= 2058 mbs – 1253 mbs) resulting acc. to Scott (1995) in a rate of 1.22 m/d. This is rate is, however, not entirely correct as Scott used a duration of 660 d while only 600 d would be applicable given the reported period of 20 months. Applying the latter as 600 days (instead of 660 d) results in a rate of 1.34 m/d, which is ca. 3 times faster than currently observed.

For the later calculation of the associated void volume that has been flooded during this period it is important to note that Scott's information implies that the lowest part of the mine void – i.e. the part below 49 level (2058 mbs) was already flooded even though this is not explicitly mentioned. This flooded part of the mine void must thus be subtracted when calculating the void volume flooded between February 1975 and October 1976.

Similar to the approach applied to the WB, the void volume flooded during the 600 days will be compared to volumes of ingress water estimated based on measured pumping rates and related auxiliary information. Based on the difference between the (calculated) original void volume and the (estimated/ measured) ingress rate that indeed filled the volume the degree to which stope closure reduced the void volume will be estimated.

As the first step the volume of the flooded part of the void will be calculated, i.e. the 805 m void part between 2058 mbs and 1253 mbs (Fig. 9.7.)

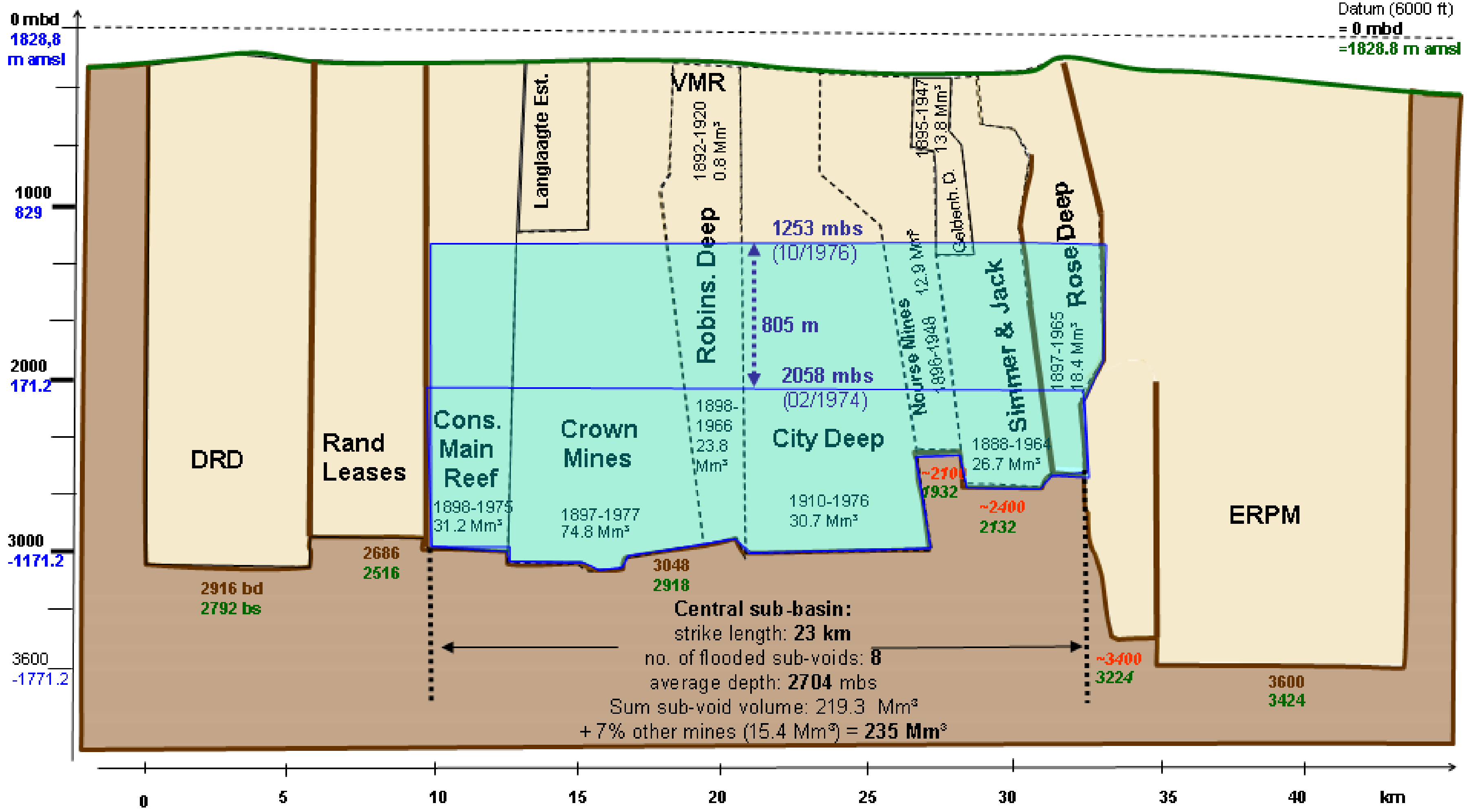

Fig. 9.7: Location of the 805 m high zone in the central sub-basin flooded between February 1975 and October 1976

Since the volume is equally distributed along the vertical void profile (with the exception of the upper 50 m of surface mining) the volume for the 805 m high flooding zone can be calculated as a proportion of the total average void depth. The average void depth was calculated from the depths of the individual sub-voids weighted according to their proportion of the total strike length of the Central sub-basin and resulted to 2704 mbs (Fig. 9.7). For the entire depth of the sub-basin this corresponds to a total volume of 239 Mm^3 (= sum of all flooded sub-void volumes: 219.3 Mm^3 – Langlaagte and Geldenhuis Deep left out as their voids are still above the mine water table after flooding - plus 7% of the sub-void sum to account for the fact that the mines listed in Fig. 9.7 only account for 93% of the total CR void volume with the reminder created by mines that were no longer active at the time).

The void volume of the 805 m-thick flooded part can now be calculated as a proportion of the average depth of the sub-basin void (2704m). With 805 m accounting for 29.8 % of the total average void depth the associated volume for the part flooded during the 20-month period amounts to 70.26 Mm^3 (29.9 % of 235 Mm^3 = 70265 Ml). To fill this volume in 600 days would have required some 117.1 Ml/d. According to Scott (1995) the ingress volume into the Central Rand sub-void is approximately 30 Ml/d (calculated from pumping figures subtracting re-circulated service water). Using pumping data from the SWV shaft which drains the Central Sub-void for the period 1951 to 2006 this ingress rate could be confirmed. After all mines were closed in 1977 the average pumping rate from the SWV shaft up to 2006 was 30.7 Ml/d (compared to 37.5 Ml/d during active mining. Evaluation of DRD daily pumping figures from SWV shaft for the period 6.8.2006 to 29.2.2008 when pumping stopped showed an average of 34.4 Ml/d largely confirming the Scott (1995) figure). This is 25.6% of the rate that would have been required if no void reduction would have taken place implying that the reduction of the void due to plastic stope closure was **74.4%**. The calculation model is schematically depicted in Fig. 9.8.

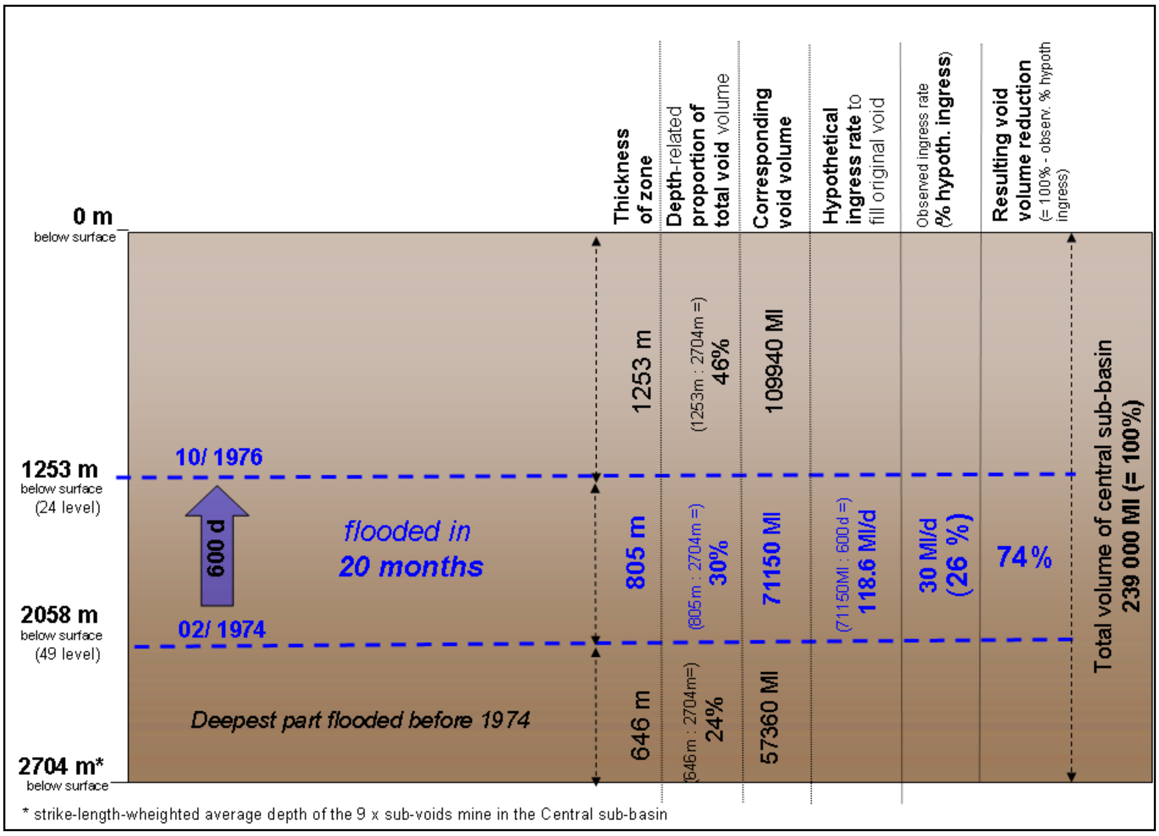

Fig. 9.8: Model and data used for calculating the mine void reduction due to plastic stope closure using a partial flooding event in the central sub-basin of the Central Rand as an example.

The fact that this is almost exactly the amount of void reduction found for the Western Basin void found for a different mine void using different underlying data confirms to an unexpected high degree that creep-induced (rheolithic) stope closure has indeed a profound impact on the reduction of the original mine void.

The high similarity of the calculated amount of reduction is perhaps also caused by the similarity of the WB and the CR mine void system which include the following features:

- in both goldfields reefs have been mined directly from surface (in contrast to the Far West Rand for example where reefs are overlain by up to 1500 m-thick non-auriferous rock such as dolomite and Ventersdorp lavas);
- both voids display similar mining depths (i.e. similar rock pressure impacting on the stope);
- both areas display a similar lithology (quartzite dominated) as well as similar reef dipping angles which also impact on the intensity of stope closure;

- at both voids the mining technology used was similar as both areas were mined over the same period. This results in similar void properties regarding the absence of any lining in old shafts (relevant for ingress rates), stope heights and support (impacts on closure rates), filling of old voids with ash from steam engines, surface operations (open pits exist in both regions acting as potential ingress areas), creating of highly permeable sand dumps (i.e. not slimes dams) as major ingress sources etc.

- both areas are located in the head water region of south-draining small rivers whose catchments are covered to a large degree with slimes dams, rock dumps and other types of mining residues impacting adversely on the quality of surface run off and groundwater;

Apart from very similar closure rates this similarity may also allow for using the WB as an analogy as study on water quality implications associated with the flooding risk in the CR. This aspect is explored in the following section.

Implications for the mine water quality

A likely implication of plastic mine closure, is that the exposure of unmined reefs which are of particular concerns as sources of pyrite, uranium and other toxic heavy metals is considerably reduced, limiting the degree to which ingressing water can be contaminated. The closure of stopes especially in the deeper part of the mine void is also of importance for possible post-flooding convection of deep mine water due to the geothermal gradient which may no longer be heavily polluted. For mine voids where much of the underground infrastructure is below the critical threshold of 1500 mbs – such the FWR – plastic stope closure may reduce the potential of stopes for future mine water pollution during and after flooding considerably.

In the West Rand decanting mine water was first observed at a low lying borehole in the Twee Loopie Spruit valley and reported to have been highly acidic. Since the relatively small borehole could not accommodate the total decant volume, the water level in the WR basin rose further until an old incline shaft used to mine the Black Reef (known as Black Reef Incline shaft or BRI) which was apparently not shown on official maps, acted as main decant point. Located at a level of 1665,45 mamsl, Krantz et al. (1999) assumed that 11.3% of the mine void is still above the decant elevation and therefore still not flooded. It stands to reason that such filling will not occur as long as all inflow into the mine void can be discharged through the current outflow points. The relative stability of the mine water level over the past 9 years seems to support this assumption.

Later measurements of pH levels in the decanting water directly at the outflow point show consistent pH-values of just below 7 i.e. in the near neutral range. This changes to acidic conditions (pH 3 to 4) after the mine water flows on surface for a couple of hundred metres, suggesting that the oxidation of dissolved reduced iron (termed 'ferrous' Fe, Fe^{2+}) contained in the mine water to so-called 'ferric' iron (Fe^{3+}) takes place. This is indicated by the massive coverage of the stream bottom with 'yellow boy', an insoluble, orange-coloured iron hydroxide ($FeOH_3$) that is formed by the oxidised iron (Fe^{3+}) binding to the OH^- of dissociated water, creating a surplus of free H^+ ions that lowers the pH (the pH value is an inverse indication of the concentration of free hydrogen ion, i.e. the lower the pH the higher the concentration of H^+-ions). Current average U-levels in the decanting water are around 100 µg/l compared to initial levels of well over 1000 µg/l for the first 2-3 years after decant (Dorling 2000).

The fact the water flowing from the mine void is neutral and only turns acidic after turbulently mixing with atmospheric oxygen while flowing on surface suggests that no (or not enough) oxygen is available in the mine void to trigger acidification *in-situ*, i.e. when the water is still inside the void (known as 'Acid Mine Drainage'). Thus, the oxidation of iron sulphide (known as pyrite, FeS_2, or 'fool's gold' owing to its gold-resembling appearance) from remaining underground ore, often advanced as the ultimate source of water pollution (since the reduced sulphur from pyrite becomes oxidised and forms sulphate, that in turn forms sulphuric acid and dissolves minerals) cannot take place inside the mine void. I.e. the dissolution of U-bearing oxide minerals from sourceucs sh as non-mined ore, backfilled tailings etc. located within the mine void by acidic water (sulphuric acid, i.e. acid mine drainage, AMD) which is so far accepted as being the major mechanism of liberating U and other heavy metals, can in fact not be the cause for the pollution of the decanting mine water.

Furthermore, in the context of this report, the frequently advanced scenario that the acidic mine water is highly corrosive and will dissolve solid concrete structures such as piles and basement walls of buildings in the CBD of JHB must be reviewed if the underground water is not yet acidic. Since the basement structures are well below surface possibly arriving mine water would not be oxidised and thus not able to acidify. In this case any flooding of basement structures by mine water could be dealt with by simply pumping from the earlier mentioned pre-installed pumping infrastructure in the same way as it is done by several other buildings in the CBD.

Moreover, the fact that acidification only happens after the mine water leaves the void also provides a possible hint as to the origin of the void water (i.e. ingress source). To date it is generally accepted that much of the decanting mine water is generated by rainwater indicated by corresponding rises in decant volumes soon after rainfall occurred in the area. This includes a comparatively weak initial response only hours after the rainfall started as well as a delayed, much stronger response with a lag of 3-4 days. Since fresh rainwater is highly oxygenated it is questionable that the majority of rainwater is indeed directly recharging the void as this would result in water flowing from the void being already acidic before exposed to atmospheric oxygen. Since this is not the case it is likely that rainwater percolates through some or other substrates which reduce its oxygen levels before it finally reaches the void. Such passage through intermediate substrates is also indicated by the 3-4 days time lag to rain events which accounts for the majority of the rainwater. The rainfall reaching the void directly indicated by the rapid (albeit small) response of rising decant volumes, may be too small in volume to cause any significant degree of oxidation/ acidification. Since microbial action will reduce the dissolved oxygen levels in infiltrating rainwater nearly everywhere, any suggestion for a possible substrate that could act as intermediate rainwater reservoir, also needs to account for the high uranium- and sulphate-levels in the decanting mine water as these must already be present in the ingressing water since pyrite oxidation obviously does not take place in the mine void to any significant degree. In this regard it is likely that much of the void ingress is in fact porewater from mining residues such as sand– and rock dumps as well as slimes dams that cover a large percentage of the catchment area that is displaced by infiltrating rainwater.

Thus the rise in decant volume observed 3-4 days after rainfall is in fact caused by old (i.e. pre-event) highly polluted porewater in slimes dams and sand/ rock dumps explaining the high U and SO_4 concentrations. The latter may not necessarily be associated with acid mine drainage, as in many instances tailings seepage with near neutral pH was found which also displayed high sulphate and uranium levels. A possible explanation for the latter relates to capillary fringe effects on slimes dams, which leads to the formation of evaporative salt crusts on the surface of the dams. These crusts form when near-surface porewater ascends via capillary action to the surface and evaporates, leaving precipitated salts and trace metals behind. Own analyses of these crusts (from the FWR and the KOSH area) indicated U-levels of 600 ppm to over 2000 ppm. This is 3 to 10 times the uranium concentration in ore mined in SA for the radioactive metal. Since these crusts were found to readily dissolve on contact with water, any rainfall onto slimes dam will recharge the sub-surface porewater with highly sulphate and uranium polluted water. For SDs it is assumed that the displacement of tailings porewater is effected by a kind of piston effect where infiltrating rain water pushes old, pre-existing porewater downward (i.e. no physical flow through the very fine grained deposits of low permeability is required). Thus, high U and sulphate levels in mine water are not always associated with acidic mine water and may therefore not exclusively caused by pyrite oxidation (i.e. AMD).

This, in turn, would mean that the AMD drainage problem is not so much generated by unmined ore, backfilled tailings and other potential pollution sources in the flooded mine void but by the vast amounts of mining residues deposited on surface. During active deep level mining this seepage migrated into the underground workings from where it was pumped back to surface to be used for tailings disposal or discharged into streams such as the Wonderfonteinspruit. For the period in which the (WB) mine void filled up (after the cessation of underground mining), percolating seepage has been stored in the mine void. Since September 2002 when the mine void was completely filled and started to decant, the void only acted as a temporary reservoir, collecting and disposing the poor quality seepage at the decant area. Thus, the problems in water quality associated with the decanting mine water are, by and large, problems which have been there all long. The only difference is now that much (although not all) of the poor quality water is concentrated at a single outflow point where it impacts on a very small receiving stream (the Twee Loopie Spruit), that lacks the required dilution potential to cope with the influx of polluted water. The fact that the mine void cannot by fully flooded (owing to the convex shape of the topographic surface the upper most 10% or so of the void volume remain dry), seepage will continue to drain from surface into the underlying void. How far this may be different in the Central Rand where parts of the mine void is expected to flood right up to surface levels, cutting some of the ingress sources off, will be discussed below.

In conclusion it is recommended to review the current understanding of the mechanisms and importance of acid mine drainage for the Western Basin, taking into account (a) that plastic stope closure reduces possible exposure of mine water to potential pollution sources such as unmined ore to a significant extent (three quarters), and (b) that water currently emanating from the WR mine void is not acidic when it leaves the void.

This review may have consequences for choosing appropriate response strategies to address what may in essence be no longer a ‘new‘ AMD problem, but a rather existing water pollution problem, caused by vast amounts of mining residues deposited on surface.

9.3 Sources of water entering the mine void (ingress sources)

As the flooding risk for the bank buildings in question ultimately results from water that recharges the mine void, it is important in the context of this report to identify the sources of this water and determine their elevation, in order to assess whether, and if so to what extent, they will continue to recharge the completely flooded void.

9.3.1 Ingress sources according to Scott (1995)

Scott (1995) distinguishes between two main types of ‘mine inflow’ sources, namely surface water and groundwater. For further distinctions a less clear combination of inflow pathways and water sources is used. The resulting ambiguity is perhaps best characterised by distinguishing between '*surface water losses via geological structures'* such as different types of faults, weathered dykes etc. and water lost from streams crossing the outcrop area after having pointed out that most streams in the area are structurally controlled (i.e. follow the orientation of faults, dykes, bedding planes etc.). Thus the main source of water for both ingress sub-categories is largely identical i.e. stream water. Therefore only 3 x major ingress sources are given:

(1) Rain falling directly onto the disturbed outcrop zone where a high proportion infiltrates owing to the high infiltration potential of the excavated and disturbed zone;

(2) Losses from surface water bodies (streams and lakes) either directly to the underlying mine void where these water bodies are in contacted with the mined outcrop zone or via geological structures such as weathered dykes, faults, fractures etc. connecting the streams to the mine void.

(3) Groundwater from a near-surface, secondary aquifer developed in the fractured shales and quartzites

(i) Direct rainfall onto the mined outcrop zones (= mine entrance)

Following surface excavation of the shallow reefs along their outcrop in the CR historical mining created an excavated and disturbed zone of 75–95 m in width and close to 40 km in length. Being generally located in topographically low-lying areas with highly disturbed land coverage a high infiltration rate of up to 95% is assumed. In addition, the outcrops of the Bird Reef series and the Kimberley Reef series also have associated surface mining belts in which infiltration is elevated.

Since rehabilitation and urban development makes it difficult to trace the excavated reef zone for its entire strike length, Scott (1995) only used 14.8 km of the 40 km as effective recharge length arriving at a direct recharge volume of 1.25 – 2.5 Ml/d.

Direct contributions from this zone have been observed at the Village Main Reef GM where strong water flows via incline shafts were found at shallow depths. At the main shaft of Waverley GM located near the Witfield stream, 2 x separate inflows from inaccessible areas have been detected close to surface (5 level some 122 mbs) both displaying typical AMD water quality with low pH (2.5-3) and elevated EC values totalling 3 Ml/d (Scott, 1995). Since contact to underground ore is limited at such shallow depths, we assume that this water largely consists of tailings seepage, suggesting that Scott's estimate of ingress from this zone may indeed have been too low. Direct evidence of slimes dams from the mining belt contributing to the recharge of the underlying void is reported for the ***Primrose GM*** where grey slimes eroded from slimes dams on surface have been deposited by ingressing water. This was confirmed by observation of interviewed underground workers, describing slimes flowing into the mine void shortly after rain storms. During February 1994, after a particularly wet period, slimes dams and sand dumps collapsed down an incline shaft that had been covered by the mine wastes (Scott, 1995) further corroborating the importance of this zone in general, and old shafts specifically, as ingress areas and points. Since cementation and grouting techniques to control inflow during shaft sinking were only introduced from ca. 1915 onwards, many of the older shafts are not lined or the lining has meanwhile failed owing to old age. Since mining started from surface to ever greater depths, the first shafts are located close to the MR outcrop (Scott, 1995: within 520 m).

(ii) Water losses from streams and lakes

Apart from loosing water via geological structures such as faults and dykes that may be connected to the underground mine void Scott also reports that streams crossing the excavated and disturbed MR outcrop area are prone to lose water directly into the mine void. It is also pointed out that many formerly non-perennial streams became permanently flowing due to the discharge of industrial and municipal waste waters generated by the growing metropolitan area of JHB which generally increased associated ingress volumes. This includes surplus water from tailings recovery plants such as the Rand Mines Mining and Milling plant (RM3). Examples given in Scott (1995) for *observed* stream losses include the following:

- the stream from Bloudam at Amalgam was channelised to prevent losses to the MR outcrop at ***Langlaagte GM***;

- in 1972 ***DRD*** channelised the stream from Florida lake where it crossed the MR outcrop to prevent some 0.02 Ml/d of stream water being lost (representing one third of the total stream flow), another 0.37 Ml/d were reportedly lost where the stream crosses the Bird Reef outcrop (total stream loss: 0.4 Ml/d)

- stream water also enters the DRD sub-void via ***incline shafts*** and ***outcrop workings*** near the Klip River

- the stream from Witfield dam on ***Knights GM*** loses to underground workings

- the ***Rand Lease GM*** no. 6 shaft derives water from Florida Lake, with 13 x shafts being located within a distance of 1.3 km from the Fleurhof dam the associated ingress potential is considerable especially as much of this intensely mined area is covered by a wetland.

Extrapolating the observed stream loss of 0.37 Ml/d to the Bird Reef outcrop at DRD to other streams crossing reef outcrop areas in the Central Rand (assuming equal flow rates and that losses to the Main Reef are double the losses to the Bird Reef outcrop and 8 times the losses to the Kimberley Reef, no losses to Elsburg reef) Scott (1995) estimates the total contribution of stream loss in the Central Rand as 8.22 Ml/d. Since probable losses from streams flowing in structurally controlled valleys have been ignored Scott (1995) regards this as an underestimation.

(iii) Groundwater

Groundwater in the area directly South of the continental devide, is mainly associated with a low yielding, fractured shallow aquifer that developed in the shales and quartzites underlying much of the Johannesburg CBD, showing a generally low storativity and a high transmissivity. While the water table has been shallow when mining started in 1886, increasing urbanisation and the associated reduction in rainwater infiltration through ever increasing surface coverage with impervious material, resulted in a drop of the water table to currently about 15-20 m below surface. While the rocks have a low primary porosity, subsequent weathering and fracturing of the near surface zone resulted in the formation of a secondary aquifer extending to some 40 m below surface. From 40 mbs to 100 mbs the fractures density decreases until no water can be stored except in features like major fault zones and deeply weathered dykes as discussed previously.

The groundwater inflow was calculated by Scott (1995) as the difference between the long-term average pumping rate by the mines (48.6 Ml/d) and the cumulative volume of all other ingress water including service water used by the mine (12 Ml/d), surface water losses, direct rainfall onto the outcrop area (1.2 to 2.5 Ml/d) and dumped tailings (accounting for 2.62 Ml/d). Thus the combined contribution of surface and groundwater amounts to 31.3-32.6 Ml/d**.** In order to determine the contribution attributable to groundwater, Scott (1995) estimated the natural groundwater recharge. For this the catchment area associated with streams flowing into the outcrop zone (309 km²) and a mean annual precipitation (MAP) of 730 mm/a was used. Assuming recharge coefficients of 3% and 4% of the MAP, the groundwater ingress was calculated as 18 Ml/d and 24 Ml/d respectively, the latter value representing the maximal contribution (Scott, 1995). Fig. 9.9 illustrates the different ingress sources and pathways identified by Scott.

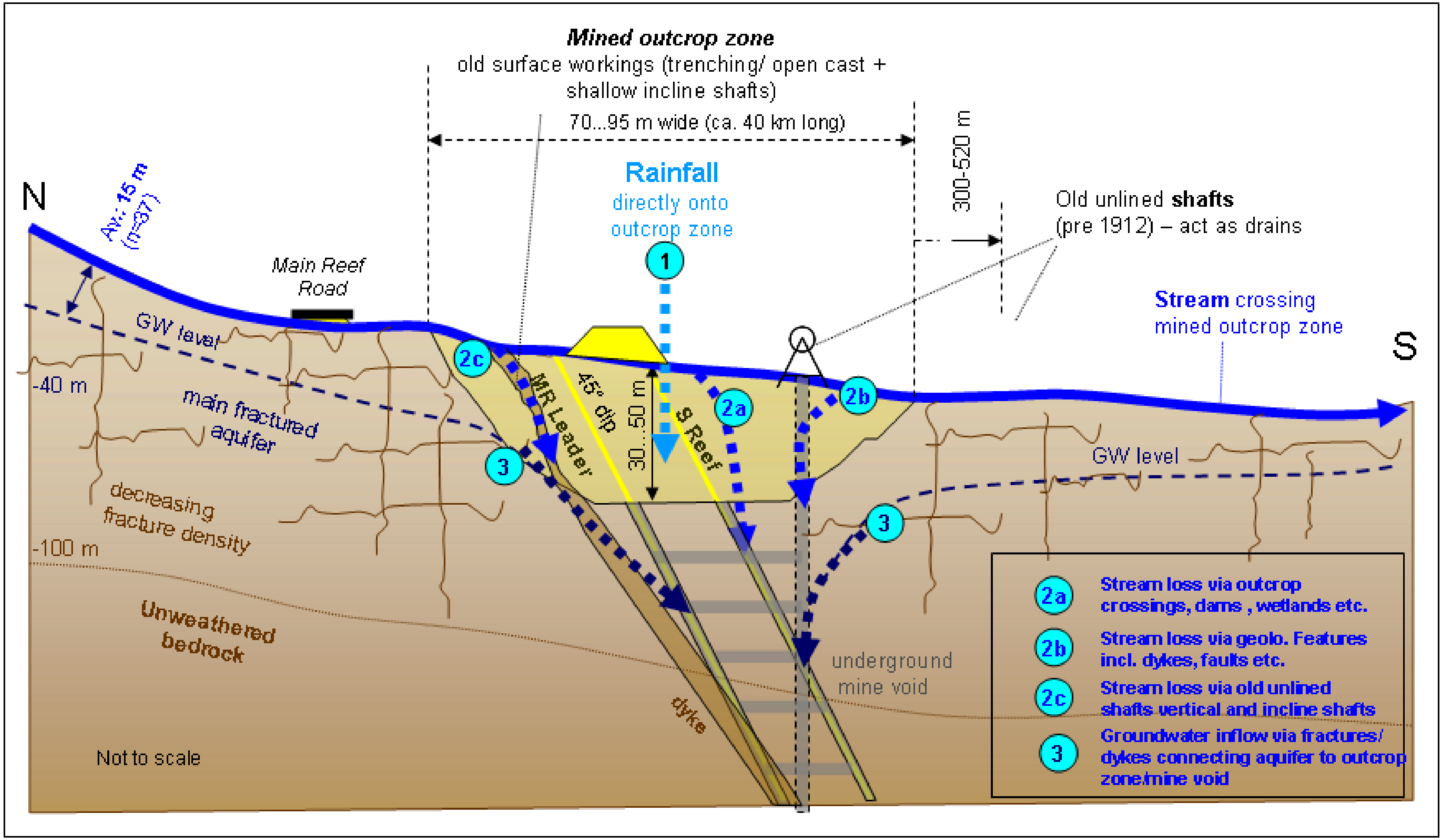

Fig. 9.9: Sources and pathways ***of natural*** ingress identified by Scott (1995)

Tab. 9.2 indicates for each of the identified ingress sources the calculated rate of associated inflow into the mine void including the underlying data and assumptions.

Tab. 9.2: Quantification of the contribution of 4 x ingress sources for each of the 3 x sub-basins in the Central Rand (Scott 1995) (blue: own calculations)

Ingress source	**[Ml/d]**	**[%]**
Average pumping from mine void	**48.56**	**100%**
Service water input	12.02	24.8
Underground tailings disposal (VMR:1982-1992: 1000 Ml/a = **2.74 Ml/d own calculation*)	2.62*	5.4
Resulting groundwater + surface water inflow (***natural ingress***) *(= pumping – mine water input) ** given at page 135*	**33.92** *31.3-32.55***	69.8
Stream loss (extrapolated from 11 Ml/month lost from Florida Lake to Bird Reef; loss to Main Reef= 2 x loss to Bird Reef = 8 x loss to Kimberley Reef)	8.22	26.7
Groundwater inflow from secondary aquifer (catchment of streams in outcrop zone vicinity: 309.29 km², MAP: 730mm/a, recharge 3-4% MAP)	18.04–24.3 Av: **21.17**	68.8
Direct rainfall onto outcrop zone (14.79 km measurable, 75-95m wide, 10-95% infiltration, MAP: 730 mm/a)	1.25-2.5 Av.: **1.38**	4.5
Total ingress	27.51-35.02 Av.: **31.26**	**100%**

Tab. 9.2 shows that some 30% of the pumped water from the mine void originates from the mines themselves in form of service water and tailings disposal. After subtracting this water Scott arrives at some 34 Ml/d of water flowing into the void from non-mining sources. This inflow is termed in this report ‘natural ingress’.

Based on data and assumptions the order of magnitude for the respective contribution of each of the identified ingress sources to the total natural ingress is calculated largely confirming the earlier determined ingress rate (31 Ml/d).

It thus follows that the total natural ingress consist of approximately two thirds of continuously inflowing groundwater, about one quarter is seepage from surface water bodies, with rain falling directly onto the mined outcrop zone accounting for the reminder.

9.3.2 Ingress source identified by other studies

Most later studies aimed at identifying and quantifying ingress in the Central Rand, in some way or another, referred to Scott (1995) even though their estimates on total ingress rates and contributions from various sources may have differed. The findings of these studies are briefly discussed below:

(i) SWAMP (19967)

Conducted as part of a larger study across the entire Witwatersrand that investigated the possibility of mine water treatment in different goldfields the Central Rand was also addressed. Regarding hydrological aspects this study admittedly relied almost exclusively on Scott (1995). While the study confirmed the general model of Scott regarding ingress sources and pathways (thus adding confidence in the applicability) the report differs on some key aspects (mainly owing to omissions or misinterpretations). The main findings are:

- mine ingress is derived mainly from shallow groundwater of a near surface aquifer (NSA) consisting of weathered shales and quartzites
- inflow via hydraulic continuity between the NSA and mine void is enhanced by a dewatering cone caused by mine void
- 3 x major inflow pathways: (a) faults and dykes; (b) 'natural interface' between NSA and mine workings; (c) vertical shafts
- Ingress correlates with rainfall with 4-7 d lag (this ignores that Scott, 1995 specified this relationship as 'non-conclusive' and found no significant correlation using a variety of different techniques including time series analyses)
- Set ingress volume equal to the average pumping volume in 1994/5 (48.6 Ml/d) (this ignores that Scott specified that service water needs to be subtracted reducing the ingress)
- In 1977 the mine water level reached 1083 mbd (22 level: 745.8 mamsl, 946 m below CBD surface) (acc. to Scott, 1995 the pumping station at SWV shaft was lifted from 49 level to 24 level at 1253 mbs – some 300m below the SWAMP elevation - were it resumed pumping in October 1976. No reference is made in Scott that it was lifted only 3 months later to 22 level.).
- From 1977 onwards the mine water level remained constant due to pumping from 3 x shafts
- Confirms the 5 x major ingress sources of Scott (1995): Groundwater baseflow (24 Ml/d), Stream loss (8 Ml/d), Rainfall onto main reef outcrop workings (1.3-2.5 Ml/d), Mine service water (12 Ml/d, ERPM only), Tailings dumping: 2,5 Ml/d
- Predicted decant volume: since steady state will develop once the mine void is filled and overflows, decant will only be fed by natural GW baseflow (24 Ml/d) with no other natural or mining sources contributing any longer (i.e. no stream loss, no direct recharge of outcrop mine area.)
- Discharge (=decant) points: final mine water level expected to correlate with groundwater level near Florida lake: 1660m amsl
- all shafts with collar elevations below 1660 m amsl are potential decant points:
    - all ERPM shafts
    - Witwatersrand GM & Co. Shaft
    - Cons. Main Reef #6 shaft

(ii) Van Biljon and Walker (2001)

This is a study on ingress into the Central Rand Basin forming part of the ERPM EMPR. The main results of this study are:

- The study identifies 3 x sub-basin within the Central Rand basin: (1) DRD/ Rand Lease sub-basin; (2) Central sub-basin (CRM-Rose Deep); (3) ERPM sub-basin.

- 4 x ingress sources have been identified:
  - (i) Stream loss via geological structures such as dykes and faults
  - (ii) Groundwater inflow from a perched aquifer
  - (iii) Direct rainfall onto the outcrop area
  - (iv) Underground slimes disposal

- each of the ingress sources has been quantified for the 3 sub-basin (Tab. 9.3):

Tab. 9.3: Quantification of the contribution of 4 x ingress sources for each of the 3 x sub-basins in the Central Rand (Van Biljon & Walker 2001) (blue: own calculations)

Sub-basin **Ingress source**	**DRD** [Ml/d]	**Central** [Ml/d]	**ERPM** [Ml/d]	**Total CR [Ml/d]**	**[% total ingress]**
Stream loss via faults and dykes*	**13** (6 DRD, 7 Rand Leases)	**8,7**	11	32.7	37.6%
Groundwater from perched aquifer (6,5% assumed recharge rate from rainfall)	**12** (8 DRD, 4 Rand Leases)	15,3	10	37.3	43.0%
Direct rainfall in outcrop (50m width for each reef) length:	**3** (9 km)	7,8 (30 km)	3 (9.4 km)	13.8 (48.4 km)	15.9%
Slimes disposal underground (early 1980s- early 1990s)	0	3	0	3	3.5%
Total ingress [Ml/d]	**28**	**34.8**	**24**	**86.8**	**100%**
[% of Central Basin]	32.3%	40.1%	27.6%	100%	

Tab. 9.4: Estimated rates of water inflow into the mine void from 3 x main ingress sources separately indicated for 4 x sub-basins in the Central Rand distinguishing between wet and dry seasons (data: Van Biljon& Walker 2001)

Ingress source	Rainfall [mm]	Rain infiltrating via **outcrop** zone [Ml/d]	**Groundwater** inflow [Ml/d]	**Stream loss** [Ml/d]	**Total ingress** [Ml/d]	% of total ingress
Sub-basin	**Wet season** (Oct-Mar)					
DRD	107.7	1.74	5.71	9.84	17.29	17.5%
Rand Leases	107.7	0.81	4	12.12	16.93	17.1%
Central	107.7	14.35	15.3	8.74	38.39	38.8%
ERPM	107.7	5.05	10	11.23	26.28	26.6%
Total wet season	**107.7**	**21.95**	**35.01**	**41.93**	**98.89**	**100%**
	Dry season (Apr-Sep)					
DRD	15.3	0.25	5.71	8.4	14.41	18.7%
Rand Leases	15.3	0.11	4	10.4	14.51	18.8%
Central	15.3	2.03	15.3	8.74	26.07	33.9%
ERPM	15.3	0.72	10	11.23	21.95	28.5%
Total dry season	**15.3**	**3.1**	**35.01**	**38.77**	**76.94**	**100%**
av. both seasons	**61.5**	**12.52**	**35.01**	**40.35**	**87.96**	
% of total ingress	-	14.2%	39.8%	45.9%	100%	

Compared to Scott (1995) and SWAMP (1999) it appears that van Biljon and Walker (2001) estimated significantly higher ingress rates reaching close to 300% of the ingress estimated by the former. In view of the fact that these estimates also exceed all pumping rates ever reported for the CR even at the peak of mining (where a large service water percentage was added) renders these estimates unrealistically high. Compared to Scott (1995) the contribution of direct rainfall is an order of magnitude higher (1.25 vs. 12.5 Ml/d) while the water lost from streams is 500% higher (8 vs. 40 Ml/d). With a 46% higher contribution (24 vs. 35 Ml/d) the difference between the 2 x studies is the smallest, albeit still considerable, for the groundwater component of the ingress.

(iii) AMD report to IMC (2010)

The report compiled by a 27 member strong team of expert was submitted to the Inter-ministerial Committee on Acid Mine Drainage and attempted to address the entire Witwatersrand basin. For the purpose of this report only facts relevant to the Central Rand have been extracted. The report lists the following ingress sources:

(1) Infiltrating rain falling directly on open mine workings, stopes and old surface workings;

(2) Groundwater seeping into workings owing to mining-related disturbances of natural conditions

(3) Surface streams losing water to mine openings compounded by high run off from paved urban areas

(4) Mine water from open pits (Western Basin example)

(5) Seepage from mining residues

(6) Possible losses from the water, sewage and stormwater reticulation systems crossing the old shallow mine workings

Regarding pumping and ingress volumes the following stated facts are relevant:

- the rate of mine water rise is 0.35–0.9 m/d depending on rainfall

Comment: it is unclear on what this statement is based as no statistical relationship between rainfall and ingress – i.e. water level rise could so far be established.

- an average volume of approximately 60 Ml/d was neutralised and discharged into the Klip River

Comment: citing Scott, 1995 it is unclear what this is based on as the cited reference does not contain such statement. Instead of 60 Ml/d discharged by DRD into the Klip River Scott (1995) indicates that 7.19 Ml/d of the total 15.8 Ml/d pumped at DRD was released into the Klip River.

- using a linear extrapolation of the WL rise observed in the Central sub-basin (monitoring shaft not specified) for a 3 months period in 2010 (est. Oct-Dec. as the figure is of a very poor quality and nearly illegible) a decant on surface is predicted for March 2013.

Comment: To what surface level in the CR this refers is not specified. The validity of an approach that uses an est. 3 data points – WL in Oct. to Dec. 2010 – to base a 2-3 year prediction on is questionable especially as longer time series for the WL rise in the CR are available. Also, no mentioning is made where in the Central Rand this WL rise was observed, which is of critical importance as the ERPM sub-void for example is not connected to the central sub-basin.

9.3.3 Ingress sources and pathways identified in this report

Since the reduction of ingress volumes is accepted as a cost-efficient method to reduce future decant volumes and associated treatment costs it is important to identify all possible sources which contribute to ingress. It is important to note that the identification of additional ingress sources will not necessarily mean that the total ingress volume is higher than previously determined, but rather that the apportionment of contributions by the different sources may need to be reviewed. It may also assist in identifying which proportion of the ingress can indeed be managed and where to do this most effectively.

Based on the aforementioned studies and own observations, it appears that two major types of ingress sources exists namely *natural and semi-natural* water sources and *man-made* inputs.

Natural and semi-natural ingress sources include the following:

(1) *Direct rainfall* onto the 3 x disturbed reef outcrop zones with an exceptionally high infiltration potential varying from 1.2 to 12.5 Ml/d according to different sources explaining between 4 and 14% of the total ingress (Scott, 1995 and van Biljon and Walker, 2001 respectively).

(2) Diffuse inflow of *groundwater* from aquifers above the mine void along the mined reef outcrop zones at rates of 21 (Scott, 1995) to 35 Ml/d (van Biljon and Walker, 2001) accounting for more than two thirds (69%, Scott, 1995) to less than half (40%: Van Biljon and Walker, 2001) of the total ingress.

(3) *Loss of surface water from streams* (bed loss) in particular where stream channels cross the highly infiltrative disturbed outcrop zones or transmissive geological features such as faults and dykes along which water can migrate into the underlying mine void. Considering only the latter, van Biljon and Walker (2001) estimated a total influx of water lost from streams of 40.4 Ml/d nearly 5 times more than calculated by Scott (1995) based on measured losses across the CR reef outcrop. This explains approximately a quarter (26.7% acc. to Scott, 1995) to almost half of the total ingress (40.4% acc. to van Biljon and Walker, 2001). While the strong seasonality of rainfall in the Highveld and the low storativity of the aquifers naturally results in seasonal flow regimes, most streams are now perennial. It has been suggested that this is due to the exponential growth in population and the associated increase in discharged sewage effluents (van Biljon & Walker, 2001). However, most of the streams crossing the mining belt only receive appreciable volumes of sewage effluents further south since all major sewage works are located well downstream of the mining belt (outside the city). Thus, sewage effluents cannot explain why streams crossing the mining belt became perennial. Since few industries developed in the mining belt or the upstream urban area, associated waste water discharges are probably limited. Since mining over more than a century deposited large amounts of tailings in the headwater areas of many of these streams, the seepage continuously emanating from the slimes dams, rock- and sand dams is a likely reason for streams now flowing all year round. This is supported by the poor quality of most streams featuring low pH and high EC values which both are indicative of mining pollution (AMD). During summer (October-March) this is augmented by sporadic inflows of massive volumes of urban stormwater runoff following intense storms which bring most of the rainfall in the Highveld. Due to the massively reduced infiltration caused by paving and the slope of many urban areas, stormwater runoff collected in

underground drainage systems is exceptionally high. However, owing to the strong seasonal occurrence stormwater cannot explain the perennial flow now displayed by most streams.

(4) *Loss of surface water from standing water bodies* such as dams (of which many have been created by mining), pans, lakes and wetlands. Rates of surface water losses are determined by the hydraulic conductivity of the bottom sediments and underlying bedrock as well as of the height of the water column driving the flow underground. While bottom sediments in streams tend to be rather coarse, consisting of sand and pebbles with a high infiltration capacity (unless tailings from adjacent slimes dams are washed into the river), those in standing water bodies are generally much finer (often sludge-like) with lower transmissivity. While this reduces the water flux through the bed it may be counterbalanced by generally higher water columns in dams and lakes compared to streams which are normally less deep. Therefore it is assumed that most standing water bodies are as important potential ingress areas as streams. This in particular where dams are located on top of the mined reef outcrop zones as observed at various sites.

(5) *Non-captured (diffuse) stormwater run-off* from slopes above the mining belt refers to that part of the runoff that is neither captured by urban drainage systems (subsequently discharging into nearby streams) nor intercepted by streams directly, but refers to runoff which in places may directly enter the mined reef outcrop zone. This can occur either on surface (via drainage lines or as sheet flow) or below surface as interflow. In dolomitic areas of the FWR diffuse (non-captured) stormwater runoff was found to trigger ground instability such as sinkholes. In response drainage canals were installed along roads and other preferential pathways cutting off and collecting the uncontrolled surface runoff for diversion into safer discharge areas. This implies that diffuse stormwater runoff may amount to considerable volumes, that could be prevented from entering the mine void. For stormwater occurring directly within the mining belt (runoff from tailings dams, mine dumps etc.) it is likely that some enters the reef outcrop zone, perhaps assisted by low-lying old unlined shafts acting as preferential pathways into the mine void.

(6) *Seepage from old, decommissioned slimes dams* and other deposits of mining residues such as sand and rock dumps. Owing to the fine grained nature of tailings, infiltrating water filling the pore space commonly forms a so called piezometric surface some distance below the top of the slimes dam. Being elevated by several tens of meters above ground, this drives a continuous flow of seepage from the slimes dams into underlying aquifers and nearby streams. In dolomitic areas of the FWR, sinkholes frequently formed underneath slimes dams owing to the continuous outflow of seepage. Due to the oxidation of pyrite contained in the tailings, the seepage from old inactive slimes dams generally is of

a very poor quality often displaying typical AMD features with low pH values and high levels of sulphate and certain heavy metals especially iron. For decommissioned slimes dams in the Witwatersrand an average pore water content of some 20 vol. % was established. While the porewater is recharged via infiltrating rainfall, the outflow of seepage is continuous, showing no seasonality. Since most decommissioned slimes dams are no longer actively managed, their contribution to the ingress into the mine void is regarded as semi-natural. Ingress may be particularly pronounced in area where slimes dams were placed directly on top of the disturbed reef outcrop zone (as it is the case at various mines) or on dykes and faults connected to the underground mine void. In some instances slimes dams appear to have been placed on top of old shafts which are likely to act as French drains providing direct ingress pathways into the mine void.

In addition to natural and semi-natural water sources some of the ingress into the mine void is ***man-made*** and relates to continued surface and underground mining activities in the area as well as urban settlements:

(1) Water inputs associated with the ***reclamation of tailings deposits on surface***[4].

(a) Water used to *hydraulically mine the old tailings deposits* using high-pressure water canons. Detailed volumes of water required per ton of tailings mined and how much of that water is reused could not be obtained. It is, however, assumed that much of this water is lost to the underground owing to the high infiltration capacity of the disturbed mining belt. The resulting seepage may contribute to the total ingress. As no deposition capacity is left at the Nasrec slimes dams DRDGold, in October 2010, announced that a new 50-km-long pipeline (including associated pumping stations) to the costs of R 300 million will link the Crown and City Deep recovery plants with the ERGO tailings dam near Brakpan. The envisaged completion date is August 2011. As the price for U increased considerably since 2003 the radioactive metal is also extracted for producing nuclear fuel. This considerable pipeline investment and indicated resources of over 175 million t at 0.3 g Au/t (Barker and Associates, 2003) suggest that reclamation of tailings in the central part of the CR will still be ongoing for some time in the future. This also means that the import of water

[4] Reclamation of old slimes aims to leach remaining gold from tailings produced at times when extraction technologies were less efficient leaving some gold in the dumped slimes. Although grades are normally relatively low (well below 1 g/t) reclamation is viable as no costs for expensive deep level mining are incurred while the required infrastructure (metallurgical extraction plants) is in place. Since the CR is the oldest of all goldfields in the Witwatersrand basin it exhibits many old slimes dams which still contain appreciable amounts of gold. Therefore, the CR is a focus area for tailings reclamation which started in the early 1980s and continued at varying intensity ever since. A major operator in the CR is the Crown Gold Recoveries (CGR), a wholly owned subsidiary of DRDGold's South African operations, CGR owns 3 x metallurgical plants of which 2 x are located directly south of the CBD (Crown Mine and City Deep) and the third one at Knights GM in the eastern part of the CR.

for hydraulically mining of the old slimes dams will continue to contribute to the ingress. During 2007 CRG started to mine the famous landmark slimes dam known as ‘Top Star’ for its drive-in cinema on top of the dam which contains 7.6 million tons of tailings and some 115000 oz of Au with an estimated value of some R 30 million (Mining Weekly, 3 August 2007). As the SD is located right on top of the heavily mined outcrop zone of the Main Reef series this may result in a significant contribution to ingress of water used to hydraulically mine the tailings dam. Labuschagne (2011) estimates that some 1 m³ water is needed to mine 1 m³ of tailings much of which apparently can be recovered and reused.

(b) Scott (1995) reports that Village Main Reef surface operations *deposited tailings directly in the mine void* resulting in an ingress volume of 2.6 to 3.5 Ml/d. While this was authorised by the DWA other operators (e.g. at the Badenhorst mine) evidently employing the same practice did not have the needed permission so that no official record on the associated ingress volumes exist (Scott, 1995). Thus, the actual volume of slurry pumped into the void is likely to be *larger than the 3.5 Ml/d*.

(c) CRG, according to themselves the ‘world’s largest tailings reclamation operation’, reportedly pumps all reclaimed tailings to *active slimes dams* at Nasrec some 3-12 km away from the two recovery plants at Crown Mines and City Deep. Seepage from active slimes dams is commonly well above the MAP and can therefore be a significant source of ingress. For the Cooke slimes dam of Harmony GM (Doornkop section of REGM) in the FWR a seepage rate of some 7 Ml/d was established indicating that slimes dams, despite their fine grained consistency and resulting low transmissivity, are major water sources. Assuming that seepage would only be equal to the MAP (730 mm/a) the 3 x SDs at Nasrec (covering a total area of approximately 5 km²) would generate some *10 Ml/d* of seepage. Over the past 25 years CRG recovered 2.8 million ounces of gold from an estimated 200 million t of mine dump material from a surface area of 1.91 km². With an average throughput of 8 Mt/a and a tailings density of 1.45 m³/t CRG treated some 5.5 million m³ of slimes annually (= 15.1 Ml/d) (Mining Weekly, 3 August 2007) . Since tailings are deposited as a 1:1 (volume) water-tailings mixture this resulted in about *15.1 Ml/d* of water put onto the slimes dams, i.e. one and a half times the volume of rainwater. I.e. over the past 25 years these slimes dams received an average of about 25 Ml/d of water, 40% of which were derived from rainfall and 60% from slurry water. Since all 3 x SDs are located directly on or up-gradient of the Bird Reef outcrop zone chances are that a significant proportion of this water finds its way into the mine void either as seepage migrating through the base of the tailings dams or as surface runoff. According to Labuschagne (2011) some 80% of the slurry water is recovered via penstock decant and return water dams and subsequently re-used. Compared to other GMs in the FWR this would be an unusual high re-use percentage.

(d) Since all tailings need to be transported from the metallurgical plants at City Deep and Crown Mines to the slimes dams at Nasrec 3-12 km-long pipelines are needed. Based on experiences in the FWR pipelines carrying process water are pressurerized resulting in relatively frequent leaks and associated spillages which, over the past 29 years or so, may also have occurred in the CR. With the current construction of an even longer pipeline (50 km from Crown plant to ERGO) leaking pipes and the associated ingress may continue in future. Like the water used for hydraulic mining this leaking slurry water is likely to be of a rather poor quality displaying typical AMD properties.

(2) Water inputs associated with active *(shallow) underground mining:* After ERPM decided to stop mining in 2008[5] it is commonly assumed that no underground mining takes place anymore in the CR. This is however incorrect as a new company called Central Rand Gold (CRG) aims to produce 1 million oz of Au per year from 2012 onwards which (if achieved) would render CRG the single largest gold mine in the world. The focus is on old worked-out mines immediately south of central JHB (Consolidated Main Reef), where unmined reef parts are to be scavenged. Underground mining was reportedly started in the second half of 2009 (Schumacher, 2009; Ryan, 2009). Due to difficulties owing to ‘bad ground conditions’ production was recently reported to be behind schedule (9321 of 19308 ounces planned) (van Rensburg, 2011). With estimated resources of 35 million ounces of Au in the entire CR and 5.2 million ounces (at average grades of 4-5 g Au/t) in the part CRG holds prospecting permits for the envisaged life span is 20 years. While details on the exact mining methods are sketchy the general approach is described as ‘all variations of cut and fill’ (Ryan, 2009) involving the deposition of all waste underground to minimize the impact on the environment (The Telegraph, undated).Together with service water required for underground operations the dumping of waste rock as well as tailings into the mine void is likely to accelerate the water level rise especially in the central part of the void by (a) providing additional water and (b) reducing the void volume through tailings and waste rock. This concentrated input of water and material may perhaps explain the sustained hydraulic head observed over the past 5 months at City Deep 4 shaft.[6]

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[5] Following a gas explosion that caused a fatality, just after a government-subsidised R 29.1 million plugging programme was completed that originally aimed to extent the life of the mine to 2011 through preventing ingress from the flooding central sub-void.

[6] According to Labuschagne (2011), however, CRG currently does not engage in any underground operations but only mines the main reef on surface in open pits near the Top Star slimes dam. The ore is milled and leached at DRD facilities and tailings deposited at the Nasrec SDs. This is in contrast to claims made by the company on their website that the mine is exclusively focussing on shallow underground mining aimed at scavenging un-mined ore deposits. According to Schumacher (2009) the executive manager of CRG Wayne Epstein as recent as March 2009 indicated the intention of the company to even ”mine underneath the water level” using their own pumps after ERPM stopped pumping in November 2008 (which he described as ‘not an unwelcome development’). This assessment

(3) *Water losses from leaking municipal reticulation systems*: This refers to water lost from pressurised systems such as the drinking water reticulation network as well as non-pressurized systems such as stormwater drainage canals and sewage pipes and canals. With an average of some 40% of the total water fed into the drinking water pipe system being lost in the greater JHB area this may constitute a possible source of ingress where pipes in or above the mining belt are affected. Chances that this indeed occurs in this area are perhaps increased as much of the mining belt is exposed to mining-related ground stress such as collapse of shallow underground workings, compaction of unconsolidated fill material in former open mine pits, heavy truck traffic, frequent seismic events etc. However, the actual contribution attributable is difficult to quantify. Apart from water seeping out of leaking pipes and canals directly into the mine void or the mined reef outcrop zone, reticulation losses also recharge the fractured aquifer underneath JHB which itself is a major source of water flowing into the mine void.

(4) Generally of minor importance but locally perhaps of significance could be the return flow associated with the irrigation of golf courses (which are a standard feature at almost every GM) and vegetated slimes dams. Since irrigation is usually adding more than double the MAP to the area in questions a significant proportions infiltrates and can thus contribute to ingress. Since the golf courses of the mines as well as the SDs are all located in the highly disturbed mining belt on soil with above-average infiltration capacity the irrigation return flow may contribute at places to the ingress.

9.3.4 Ingress water quality aspect

Since many of the ingress sources release considerably polluted water often already displaying a typical mining signal (i.e. low pH and high salt and heavy metal load as e.g. found in seepage from slimes dams), much of the water entering the mine is polluted even before it enters the mine void. This will make it difficult to assess after the mine void is complete flooded how much of the water entering adjacent rivers is indeed decanting mine water, and how much water comes from surface sources that can no longer flow into the mine void or has bypassed the mine void.

Also, during active mining, much of the pumped water has been discarded into nearby streams. Assuming that the water quality parameters of these discharges was similar to the decanting water (U levels of close to 600 µg/l have been reported), uncontrolled decant of mine water from the flooded mine void may, after all, have consequences which are not that much different from those experienced in the past. A possible

seems to have changed drastically since then as in February 2011 CRG is reported to complain that gold reserves will soon be inaccessible due to flooding (van Rensburg, 2011).

difference is that the majority of the decant water will concentrate at a single point (decant shaft near Cinderella dam) and thus affect only one receiving stream (in this case the Elsburg Spruit), while in the past mine water discharges were more evenly distributed affecting also other streams, some of which (such as the Klip River) may have a higher dilution capacity owing to larger flow rates.

Where mine water impacts on shallow aquifers underneath inhabited areas, the risk of the resulting radon exposure to inhabitants should be evaluated. This is particularly true for situations where the radioactive gas can migrate from the water table through the overlying material (e.g. sand) into basements and living rooms of overlying buildings (or shacks) and accumulate indoors. As a known cause of lung cancer radon exposure constitutes a sever health risk.

9.3.5 Temporal ingress patterns

Depending on the type of ingress, the associated temporal patterns of contributing to the flooding of the mine void are quite different. Thus temporal patterns of different sources may assist in identifying their respective contributions to the total ingress and thus help to predict decant rates. Tab. 9.5 lists from all identified ingress sources and sub-sources the associated temporal patterns as well as other features relevant to characterise the different ingress sources.

Tab. 9.5: Overview of different types of ingress sources and their characteristics regarding spatial extent (point or diffuse source), volume, temporal characteristics, water quality and the manageability in terms of interventions aimed at ingress control and reduction

Type of ingress	Ingress source	Sub-source	Point/ diffuse	Volume* [Ml/d] (%)	Temporal pattern	Water quality	Ingress reduction measures
Natural	**Direct rainfall** onto outcrop zone	Infiltrating part of received rainfall	diffuse	**1.4** (4.5) **12.5** (14.2)	strongly seasonal	good - poor	Capturing in drainage canals
	Groundwater from fractured aquifer		diffuse	**21.2** (68.8) **35** (39.8)	continuous	good	Lowering groundwater table by pumping
	Surface water loss	Streams crossing outcrop	point	**8.2** (26.7)	continuous	medium-poor	Prevent seepage - canals, pipes
		Stream crossing dykes and faults	point	**40.4** (45.9)	continuous	medium - poor	Prevent seepage by canals, pipes
		Lakes, dams on outcrop	point	to be quantified	continuous	medium - poor	Drain water, lining
		Lakes on dykes/ faults	point	to be quantified	continuous	poor	Drain water, lining
Semi-natural	**Urban stormwater runoff**	Diffuse runoff entering mined outcrop zones	diffuse	**? %** of **20** (32 summer, 8 winter)	Event related, seasonal	good - medium	Capture runoff in drainage canals
		Captured in drainage canals discharged into streams	point	**? %** of **81** (130 summer, 40 winter)	Event related, seasonal	medium-poor	Run-off reduction measures
	Surface runoff from slimes dams	May directly enter mined outcrop zones	diffuse		Event related, seasonal	poor	Capturing in drainage systems
	Seepage from old **inactive SDs**		point	24% of stream flow	continuous	very poor	Coverage of SDs, Removal of SDs
Man-made	**Reclamation of tailings** on surface	Water for hydraulically mining old SDs	point	1:1 water-tailings vol ratio (?)	daily/ weekly	poor	Difficult
		Underground disposal of reclaimed tailings	point	**2.6** (5.4) **3.0** (3.5) **>3.5**	daily/ weekly	very poor	Deposition outside of CR
		Seepage from active SDs used for disposal of reclaimed tailings	point	**10-15**	continuous	very poor	Deposition outside of CR
		Tailings pipeline leaks/ spillages	point	? % of 15 Ml/d pumped slurry	sporadic	very poor	Patrolling, monitoring, maintenance
	Shallow underground mining	Service water	point	**12** (24.8)	daily/ weekly	good	Difficult
		Underground disposal of waste rock and tailings	point	Depends on production: Past: 3 Ml/d	daily/ weekly	n.a.	Deposition outside of CR
	Losses from reticulation systems	Pressurized drinking water pipes	diffuse	Av. loss JHB: 40% of input	Long-term/ continuous	very good	Pressure reduction, replacement, monitoring, maintenance
		Non-pressured sewage canals	diffuse	?% of **600**	continuous	poor	Maintenance, monitoring
		Non-pressured stormwater canals	diffuse	**? %** of **81** (130 summer, 40 winter)	Event related	medium-poor	replacement, monitoring, maintenance
	Irrigation in mining belt	Golf courses	point	15% of 1700 mm/a	seasonal	medium-poor	No or drip irrigation
		vegetated SDs	point	10% of 1000 mm/a	seasonal	medium-poor	drip irrigation

* received by the CR basin void system [Ml/d] and (% of total ingress) according to different sources: Scott (1995); Van Biljon & Walker (2001) and this report (2011); ?% of - proportion of ingress from total volume unknown

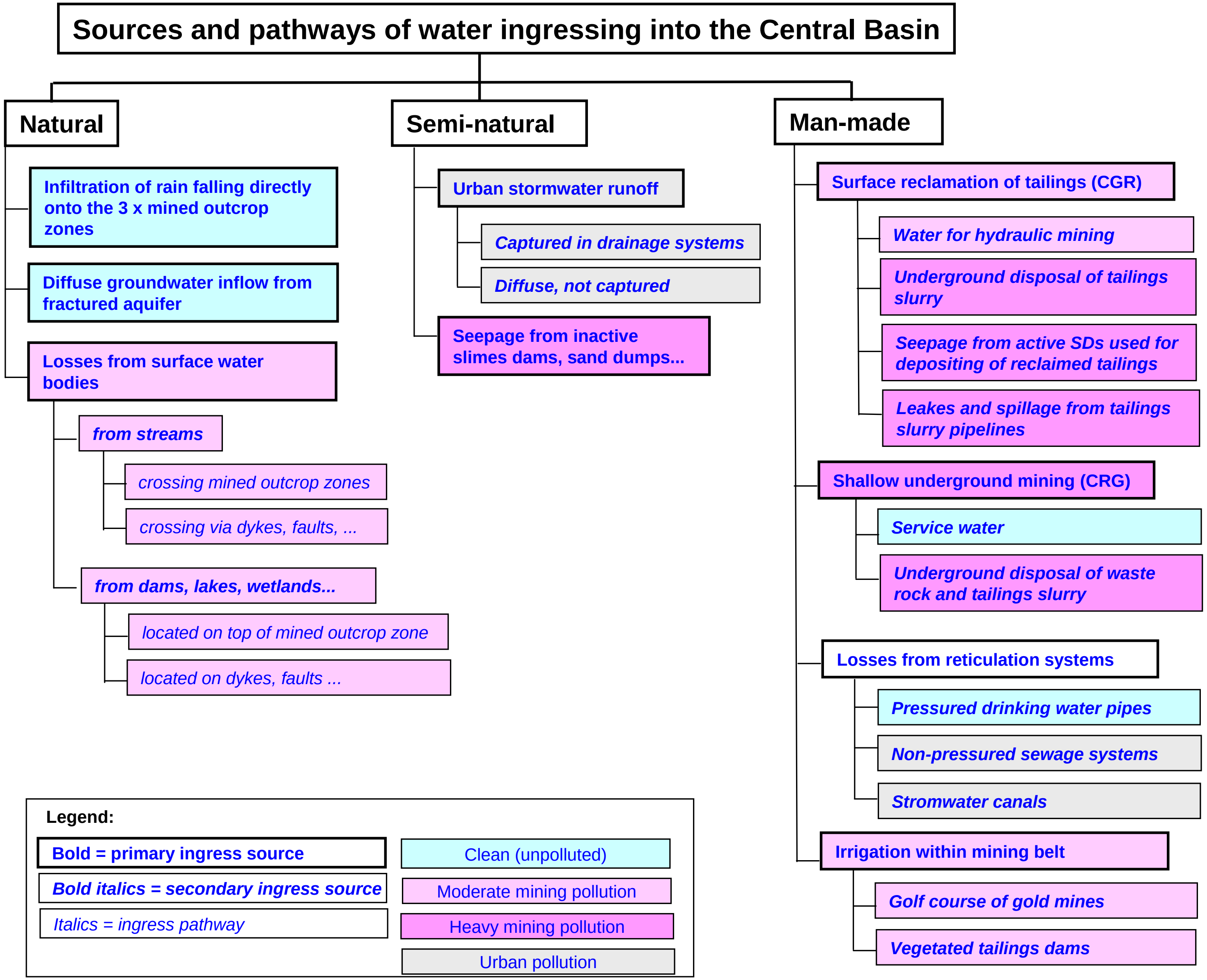

Fig. 9.10: Overview on source and pathways of water recharging the mine voids of the Central Basin

In total 9 x different ingress sources have been identified of which 5 x are classified as natural or semi-natural and 4 as man-made. Together with the various sub-sources a total of 18 x water sources were identified that potentially contribute to the flooding of the mine void. Of these 18 x sources only 4 x provide unpolluted water while 10 x are associated with water which is either moderately or heavily polluted by mining activities even before it enters the mine void. This water is typically low in pH and high in electrical conductivity (EC) indicating elevated concentrations of salt (mainly sulphate) and dissolved heavy metals such as Fe and U. This renders much of the polluted ingress indistinguishable from acid mine drainage that may emanate from the flooded void.

The remaining 4 x sources display a pollution signal which is different from mining especially with regard to the pH as it is generally neutral to alkaline. While the EC may also be elevated (especially in insufficiently treated sewage and stormwater runoff from roads, roofs and parking lots) the spectrum of elements will differ with sulphate becoming less dominant. Also, the spectrum of dissolved heavy metals is likely to change prominently featuring elements originating from typical urban sources such as metal pipe reticulation systems and roof gutters (such as Zn and the accompanying Cd as well as Cu and Pb in places), human excretion (Zn, Fe) as well as traffic-related metals (Cr from corrosion of cars, Cd from wear and tear of tyres, Pb from leaded petrol etc.). Sometimes Hg concentrations are high where dental practices are located in stormwater catchment areas. However, owing to the deposition of windblown tailings dust on nearby urban areas also tailings-related metals such as U, Ni and the metalloid As maybe elevated. Of concern are also the frequently increased levels of bacteriological contamination owing to insufficient sewage treatment and the discharge of raw sewage into open stormwater canals frequently practiced in informal settlements.

The number of ingress sources identified in this report significantly exceeds those in any other report we had access to. The relative contribution of each identified sources could not in all cases be determined and perhaps never will. It is, however, of importance that these sources are investigated should the reduction of ingress still be desired. In our opinion prevention of ingress should be given priority as this not only reduces the volume of future decant that may need treatment for very long periods to come but in some cases could also avoid the pollution of clean water. Some possibilities of reducing the ingress volumes for the different sources are listed in Tab. 9.5. When assessing the acceptability of the associated costs the alternative should also be considered, i.e. treating large volumes of polluted water for very long periods.

Although chemical end-of-pumping-pipe treatment has been suggested by the AMD report to Cabinet other low-costs and low energy options which would not require ongoing subsidy by society should perhaps also be considered. This could include the investigation of a scenario where no pumping takes place and water naturally overflow at the identified decant point. With est. volumes of 30 to 40 Ml/d possibilities of using the water untreated for suitable purpose should be explored. Scott (1995) gives an example from the Germiston municipality using 4 Ml/d of acidic mine water for enhancing nitrate digestion in their sewage works. Given that 5 x large sewage works are located south of the mining belt alone (i.e. water could get there by gradient flow) this could accommodate two thirds to about half of all the decant water with no treatment costs incurred and clean water, otherwise used for this purpose, saved. A pumping pipeline across the continental divide to the north of JHB may well be able to bring the water to additional sewage works which could utelise the untreated mine water. As these pipelines need to be rubber-lined to avoid corrosion assistance from mining industry should be explored as many old pipelines exists that used to transport tailings slurry which may no longer be needed. In such a way all of the decanting water could perhaps be sued without large capital investment, high running costs for treatment as well as high energy costs for continuous pumping. Apart from saving large amounts of required capital expenditure and relieving the taxpayer/ water users from a long-term costs burden such type of low costs, low energy solution would also have a much smaller ecological footprint and would thus be more sustainable.

Temporary uses of decanting mine water could also include the irrigation of vegetated and unvegetated slimes dams to prevent dust pollution especially during the dry and windy winter months adversely affecting residents in the densely-populated surrounding areas. Apart from implementing the constitutional right to an environment that is not harmful this, from a macroeconomic point of view, would also reduces costs associated with poor health such as expenditure related to medical care, absence from work, etc. This could be practiced for an interim period until the slimes dams are removed as part of a comprehensive rehabilitation programme possibly aligned with the ongoing reclamation of tailings deposits in the area.

9.3.6 Relationship between ingress and rainfall

Despite the fact that stream flow and infiltration of rainwater falling directly onto the outcrop are evidently major ingress sources, Scott (1995) could not establish a relationship between ingress rates and rainfall volumes. This is in contrast to a number of references claiming that a direct link exists (e.g. SWAMP 1996, van Vuuren 2011). The latter source states (quoting Prof. TS McCarthy from WITS University that the ‘rate of rise increases exponentially during the rainy season’). As similar assertion is made by Coetzee et al. (2010).

Using daily pumping rates for a 6.3-year period (2696 data points) as well as water pressure registered by underground plugs, he found no statistically significant correlation (r= 0.21 for rainfall vs. daily pumping and r = – 0.01 vs. plug pressure). A time series-based correlation using smoothed data (moving averages) also only yielded a non-significant correlation coefficient reaching a maximum of 0.34 at a time lag of 20 days. Comparing monthly averages instead of daily means finally suggested a – non-conclusive- ingress-rainfall relationship with a 4-7 days response time (Scott, 1995).

In view of the fact that much of the identified ingress originates from continuously flowing sources it is not surprising that no correlation between rainfall and ingress could be established. While groundwater is ultimately depended on rainfall this relation is lagged and often difficult to establish. Furthermore, unless the fractured aquifer dries up completely, it is believed that the influx of groundwater is rather steadily controlled, more by the dewatering cone of the mine void and the resulting steep hydraulic gradient than by the (rain-dependent) elevation of the water table. This also applies to surface water sources such as lakes, perennial streams and dams which no longer dry up in winter and thus act as continuous sources for seepage migrating through the stream bed of rivers or the bottom of unlined standing water bodies such as wetlands, lakes, dams and pans. A possible rainfall-controlled component is related to increased infiltration rates triggered by rising water levels after rainstorms as found for a mining-impacted stream on sandstone by Winde (2005), and possibly associated flood events which may temporarily increase recharge rates. However, compared to the majority of ingress water which ingresses continuously, the rain-controlled component is relatively small and thus difficult to detect. A significant relation between ingress and rainfall would require that much or even most of the water is derived directly from rainfall (such as stormwater runoff) or from non-perennial streams with rainfall-controlled flow regimes.

Furthermore, pumping rates and pressure plug recordings us used by Scott (1995), as proxies for the ingress, have their own dynamics which obscures any possible relation to rainfall and which are quite often unrelated to the actual water volume entering the mine (ingress). Pump regimes, for example, in most mines show distinct day-night differences as well as weekly patterns induced by the varying electricity tariffs for peak and off-peak times. Having used daily rates these diurnal fluctuations have affected the statistics. This is in addition to scheduled and unscheduled interruptions due to maintenance, pump failure, power cuts, accidents, strikes etc. Pressure recordings from plugs, on the other hand, are controlled by the height of the water column above the plug. This, in turn, is often artificially regulated by opening and closing of valves between sub-voids, pumping regimes as well as the location of links to other voids through which water may decant out of or into the measured mine void. Thus it is not surprising that Scott (1995) or any other study to date could not find a statistically significant relation between ingress and rainfall. This is somewhat counterintuitive as rainfall is the ultimate driver of all surface water sources and because its effects can be seen underground in the form of subsequent stormwater inflows. While the visible link between rainfall and the resulting influx of stormwater is convincing evidence for rainfall affecting ingress, it is of too little relative importance in terms of volumes to be of statistical significance.

In order to assess the impact of rain on the ingress changes, the rate at which the mine water level rises were analysed comparing the average water level rise before and after a period of heavy rainfall in the CR which occurred late December 2010 to end of March 2011. Comparing the average rate of rise over the 4 months preceding the rain period where only sporadic low intense rain occurred (32 cm/d) with the average rate in the 3 month of heavy rains (44 cm/d,) suggests that an increase of some 12 cm/d can be ascribed to rainfall. The highest rainfall intensities reaching up to 130 mm/d were measured in late December and mid January where water level rise rates peaked at 49 cm/d. Compared to the total pre-rain average precipitation, this water level rise accounts for an increase in ingress (as the cause of the water table rise) of 37.5% , suggesting that this percentage of the ingress is derived from rainfall-controlled ingress sources (Fig. 9.10).

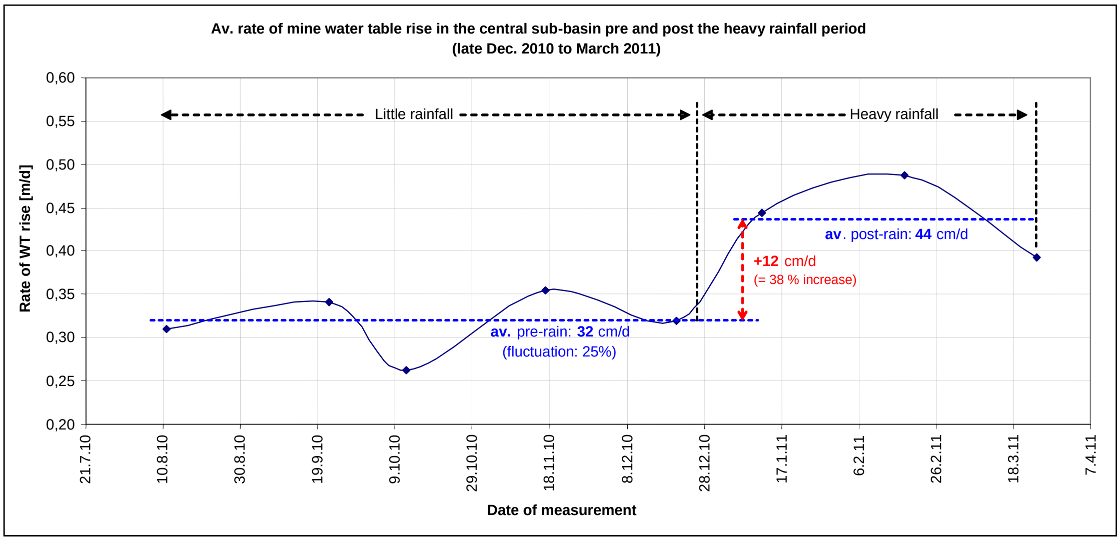

Fig. 9.11: Comparing rates of mine water rise in the central sub-basin before and after a period of heavy rainfall indicates that some 38% of the ingress may be somehow rainfall depended

This compares favourably with the estimate of Scott (1995) who suggested that direct rainfall and rainfall-depended stream loss together account for 31.2% of the total ingress. Van Biljon and Walker (2001) calculated a combined surface water contribution (rainfall plus stream loss) of 60.1% of the total ingress volume. Since much of the stream loss which accounts for 46% is no longer directly rain-controlled (i.e. seasonal flow that only occurs in the wet season) but only indirectly controlled via higher stream levels that increase the gradient which drives water through the stream bed, the calculated proportion of 38% of rain-depended ingress is also not unreasonable.

9.4 Identification of ingress areas

As high-lying areas are more likely to determine the piezometric surface in the interconnected mine void system, these are of particular importance to the project. Also, being located above the future decant point(s) these areas are likely to continue recharging the flooded void thus maintaining a somewhat elevated decant volume.

With DRD and adjacent mine areas forming the highest lying part of the Central Rand potential ingress areas have been identified for this area (Fig. 9.12):

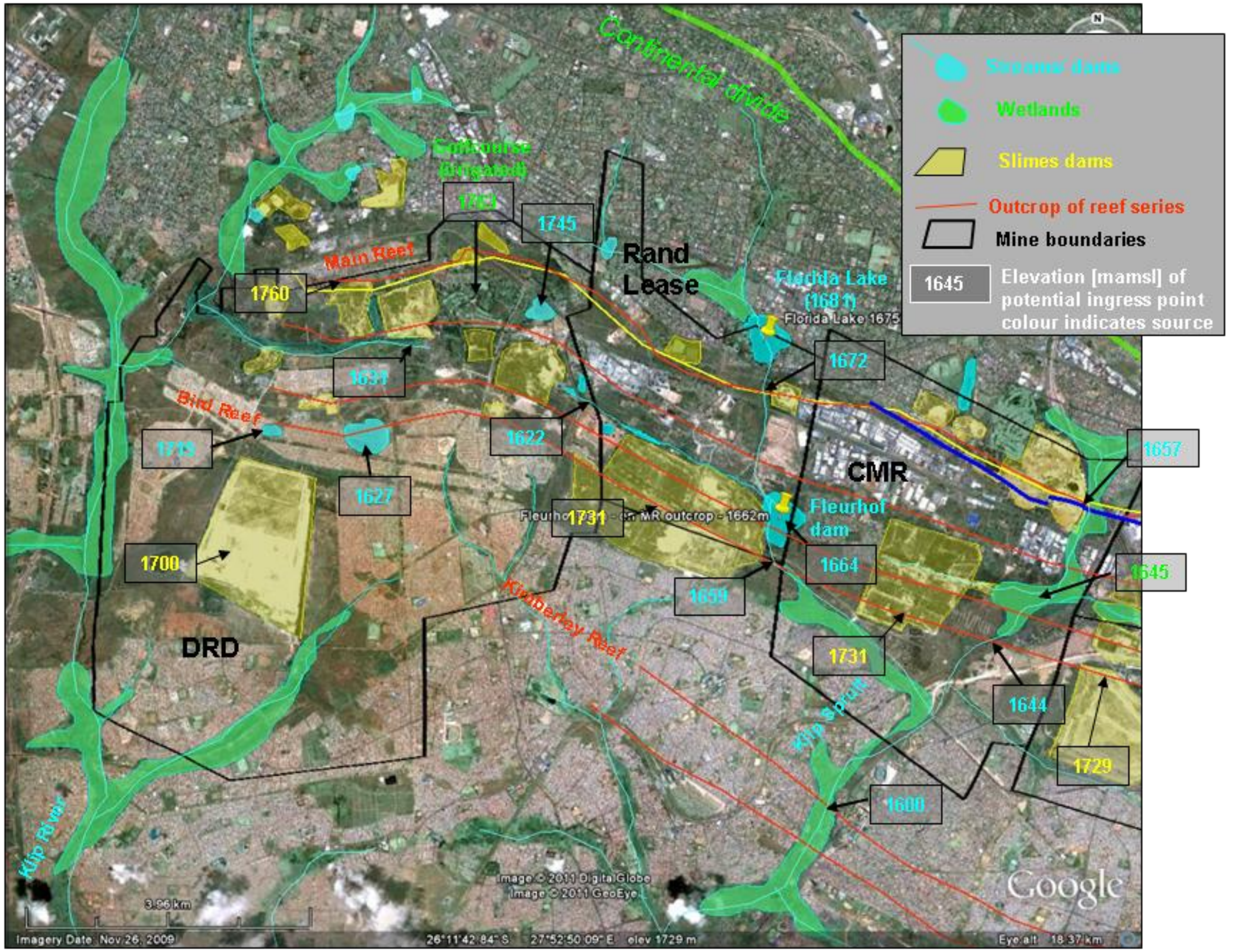

Fig. 9.12: Elevations determined in Google Earth for various ingress sources at DRD, Rand Lease and CMR as the highest lying part of the Central Rand where ingress levels are likely to determine the piezometric surface for the Central basin.

Based on recent satellite imagery retrieved from Google Earth the following potential ingress features have been identified:

(a) N-S running streams crossing the mined outcrops of the Main Reef series (2 x crossings) and the Bird Reef series (2 x crossings); Water loss from the stream flowing from Florida Lake into the Fleurhof dam has frequently been reported to be a major ingress source. This stream (a tributary to the upper Klip Spruit) crosses the Main Reef outcrop at 1672 mamsl and the Bird Reef outcrop (after leaving Fleurhof dam) at 1659 mamsl. Scott (1995), citing internal mine reports, quantifies the loss via the outcrop zone of the Bird Reef only at 11 Ml/month, on which he bases further estimates for stream losses over the entire Central Rand. Apart from bed loss, ingress to the mine void is also likely to be generated by shallow alluvial aquifers in the associated floodplains. Owing to increased stream flow sustained by the continuous outflow of tailings seepage, many rivers developed accompanying wetlands where elevated water tables in the connected alluvial sediments allow for reed beds to develop. During flood events following intense rainfall (which is common in the area) flood water from the stream may temporarily increase the ingress from submerged wetlands.

(b) 4 x E-W or W-E running streams/ drainage lines partly with associated wetlands that run parallel to the outcropping reefs right in the centre of the mining belt for a total length of approximately 12.2 km. These streams are highly polluted as much of their base flow component consists of acidic seepage carrying high loads of sulphates and dissolved heavy metals from adjacent tailings deposits that cover most of the small catchment areas. During and after rainfall stream pollution is exacerbated by heavily polluted stormwater run off entering the streams on surface often carrying eroded tailings particles into adjacent river beds. In a screening sampling campaign the Institute for Water Quality Studies (IWQS) of the DWA found for Russel's stream (a right tributary draining DRD slimes dams towards Fleurhof dam) a concentration of 4100 µg/l dissolved uranium, the maximum of all mining areas of the entire Witwatersrand (Kempster et al., 1998). During active mining it is likely that sewage from hostels housing the large workforce on the mines diluted the input of tailings seepage and run-off. However, this is no longer the case as most mines are now defunct. By the same token it is assumed that the closure of mines also resulted in less pollution through discharged mine effluents from metallurgical plants, pumped surplus water etc. The fact that most streams continue to remain perennial even after most mines stopped discharging sewage and mine waste water indicates the role of tailings seepage as a major non-seasonal baseflow component.

(c) In total 8 x standing water bodies (dams and lakes, including the highly polluted Fleurhof dam) are located south of the MR series outcrop with elevations ranging from 1745 mamsl to 1664 mamsl. Fed by highly polluted streams as well as by tailings seepage entering the standing water bodies diffusely, water from dams and lakes is also of extreme poor quality.

(d) With more than 12 x slimes dams a significant proportion of the surface area is covered by tailings deposits which are a known source of air and water pollution. Apart from an elevated piezometric surface formed by contaminated porewater driving seepage into underlying aquifers (incl. the mine void) and adjacent streams slimes dams are also a source of particle-related pollution filling nearby river beds and dams with eroded tailings material (in some cases to such an extent that the deposited tailings are mined like e.g. at Lancaster dam in the WR).

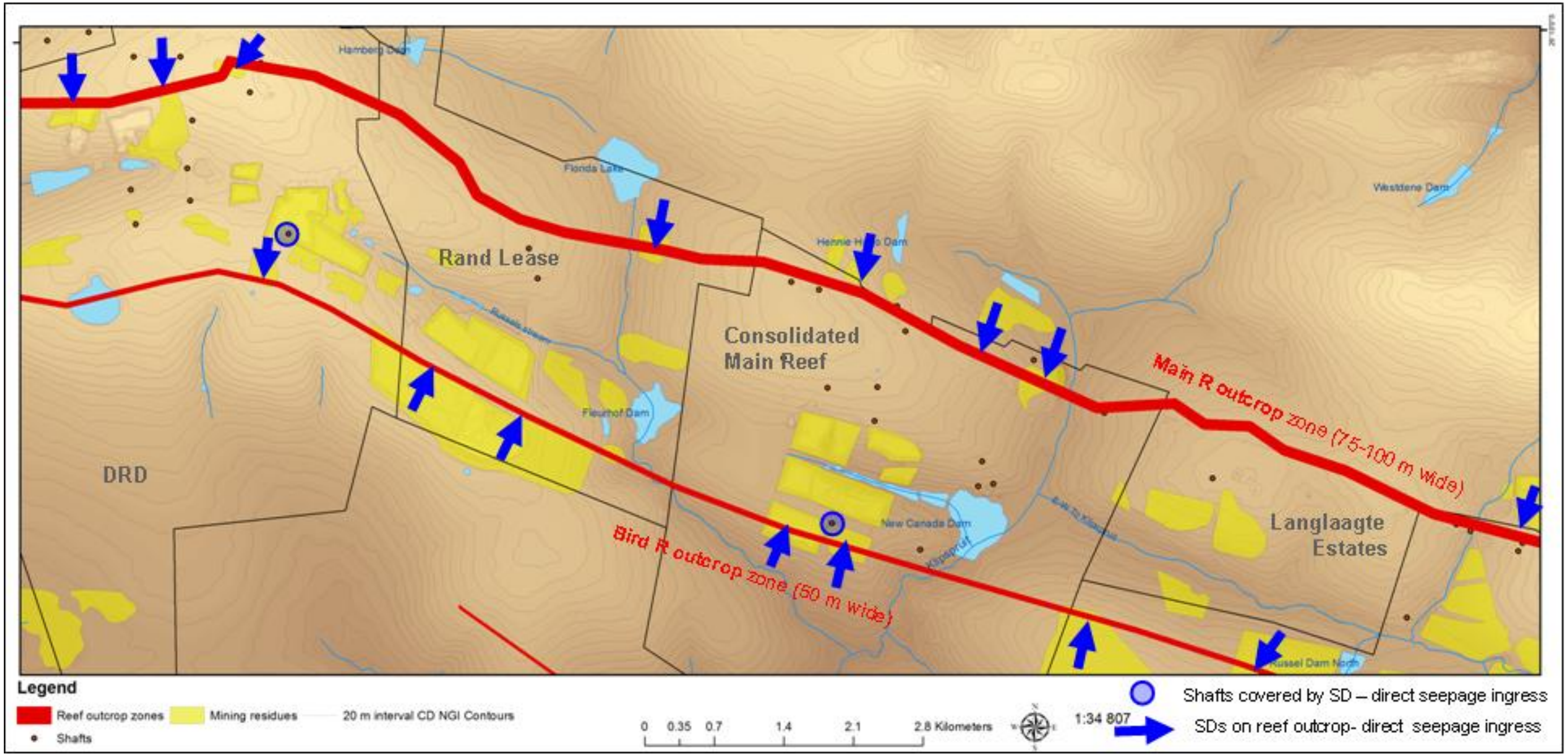

Fig. 9.13: Map of the western Central Rand depicting slimes dams located directly on or upstream of the mined reef outcrop zones as well as shafts buried underneath SDs acting as French drains for tailing seepage entering the mine void.

Even though it may be of lesser relative importance, the role of golf courses as potential ingress areas should be mentioned as golf courses are a standard feature of most GMs in the CR. In most cases, golf courses are irrigated all year round, during active mining commonly using pumped mine water. Scott (1995) reported for DRD that mine water was also used by staff living in the mining village for watering private gardens. With average irrigation rates in the region of some 1700 mm/a, resulting in a water volume about triple the natural rainfall, irrigated gold courses – where they are still maintained after mine closure – are potential ingress sources, presently using presumably imported water (Rand Water).

In addition to DRD and Rand Leases higher-lying ground also occurs in the central part of the mining belt close to the CBD occupied by the Crown Mines, Robinson Deep/ Village Main Reef and City Deep/ Nourse mine lease areas. The geographical distribution of potential ingress sources for this part of the CR is depicted in Fig. 9.14.

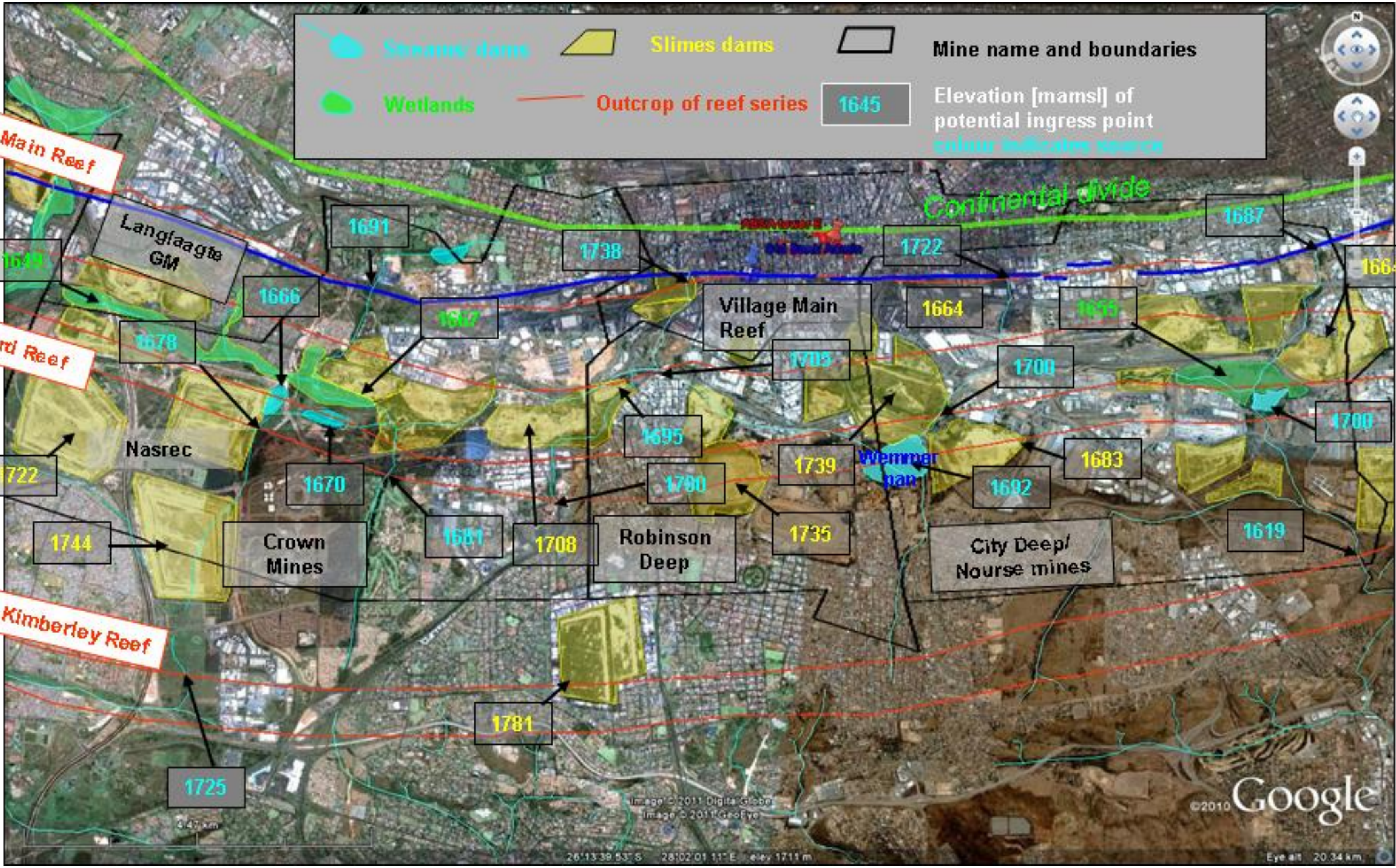

Fig. 9.14: Elevations (determined in Google Earth) for various ingress sources at near CBD mines as second highest lying part of the Central Rand (CPBL for Google Earth: 1722 mamsl)

Generally the overall land cover and use of this area is similar to the DRD-Rand Lease properties with mining residues and abandoned mine infrastructure dominating land use patterns. Fig. 9.14 indicates a number of identified ingress areas associated with the following features:

(a) *N-S and S-N running streams crossing* the mined outcrop areas of the Main Reef series (4 x crossings), the Bird Reef series (5 x crossings) and the Kimberley Reef series (2 x crossings) ranging in elevation from 1619 mamsl to 1838 mamsl;

(b) *2 x E-W running streams/drainage lines* and associated wetlands running parallel to outcropping reefs right in the centre of the mining belt for a total length of 13 km (1649-1700 mamsl);

(c) *4 x standing water bodies* (dams and pans) ranging from 1666 mamsl to 1700 mamsl in elevation

(d) *19 x decommissioned slimes dams and other deposits of mining residues* ranging in elevation (of the top surface) from 1664 to 1781 mamsl. However, since most slimes dams are decommissioned the associated porewater table is lower than in active dams. Based on the frequently observed thickness of the oxidation zone in old slimes dams of 2-5 m below surface the piezometric surface for old tailings is assumed to be some 5 m below the top of the dam. Thus the elevation of the actual ingress source is to be reduced by approximately 5 m. In several instances it appears that slimes dams and mine dumps have been placed on top of natural drainage lines and streams as well as on outcropping reefs mined on surface. Especially in the latter case a direct inflow of tailings seepage into the underlying mine void is likely.

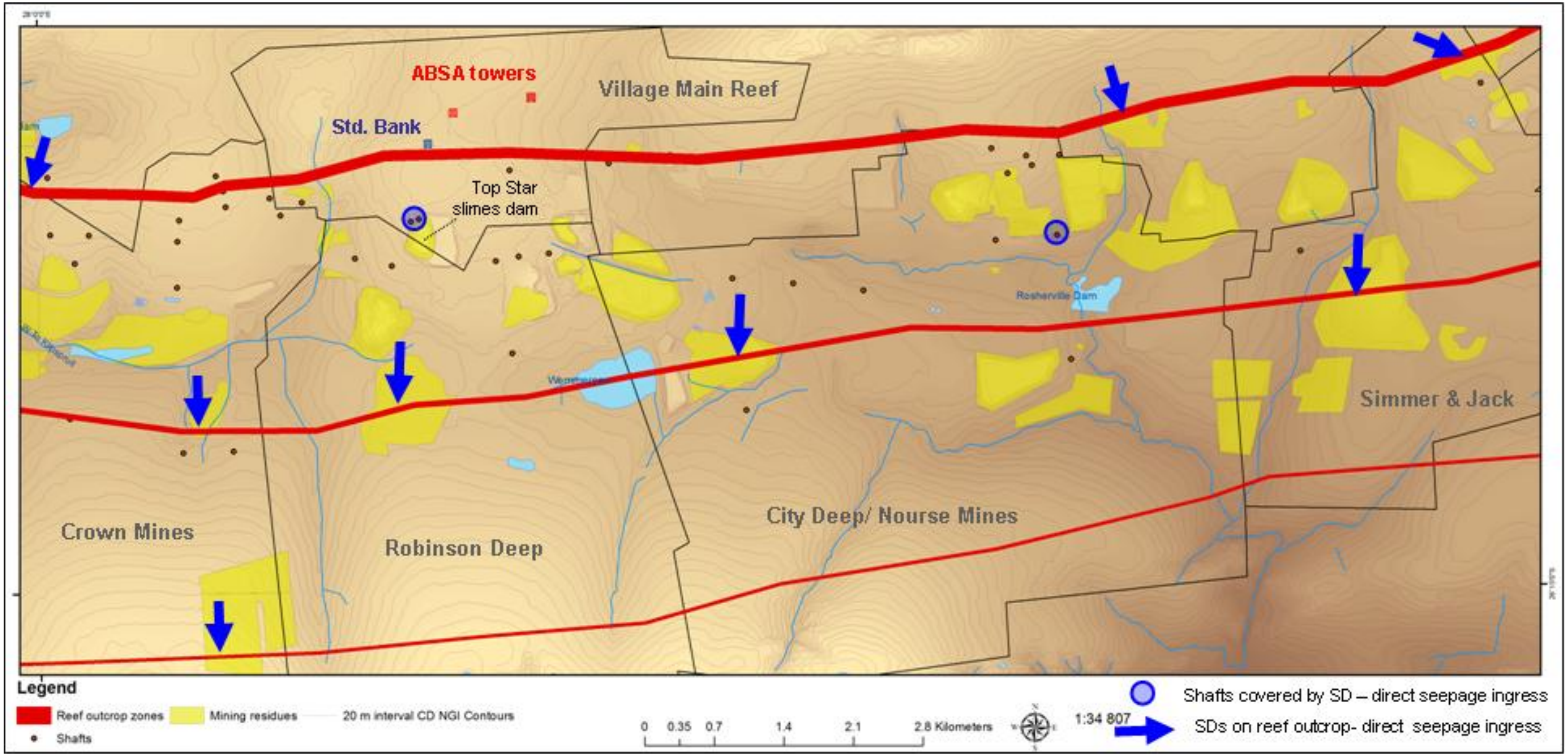

Fig. 9.15: Slimes dams located directly on top or up-gradient mined reef outcrop zones in the central part of the Central Basin

(e) *3 x large active slimes dams* at Nasrec where most of the reclaimed tailings mined in the central part of the Central Rand are deposited. Owing to the ongoing application of slurry (a 1:1 water-tailings mixture), seepage from these dams often well exceeds the mean annual precipitation. As mentioned earlier, for one of the Cooke slimes dam of Harmony GM in the upper Wonderfonteinspruit catchment a seepage rate of approximately 7 Ml/d was calculated. With about triple the total footprint area, the Nasrec slimes dams alone are estimated to release some 20 Ml/d of seepage. Of particular importance for their potential as ingress source is the fact that these slimes dams are located right on top or up-gradient of the Bird Reef outcrop zone which is directly connected to the mine void (Fig. 9.14).

(f) With the ongoing reclamation of old slimes dams being particular prominent in this area the import of water required for the hydraulic mining of the slimes dams using high-pressure water canons most probably involves considerable volumes. As much of this water used to liquefy the old tailings is running of freely, a significant proportion is probably lost to the underground. This in particular as the slimes dams are located in the highly disturbed outcrop zones which displays generally a high infiltration potential. In addition to direct losses of water from the high-pressure water canons the large-scale reclamation of surface tailings deposits also has possible indirect consequences affecting ingress such as leaks and spillage from broken pipelines carrying the slurry from the metallurgical plant over several kilometres to the slimes dams at Nasrec. Recently announced plans by DRD to install a new pipeline from its Central Rand surface operations to the ERGO tailings dams in the East Rand indicate that tailings reclamation is not only ongoing currently but will continue to do so in the foreseeable future.

(g) Fig. 9.14 clearly illustrates that areas adjacent to the E-W running mining belt are densely urbanised consisting of formal as well as informal settlements. In many instance informal settlements encroached on to slimes dams and houses have even been built on top of slimes dams. Since paving in urban areas limits infiltration, the run-off component generally increases, resulting in many urban water courses receiving unnaturally high volumes of frequently polluted stormwater. As the urban areas in this part of the CR are located on hill sides sloping toward the E-W running valley in which the outcrop zone is located, urban stormwater run off may also be a significant (albeit seasonal) source of ingress. Apart from runoff collected stormwater drainage systems (underground canals), this also comprises diffuse run off from informal areas which may be less rapid and somewhat lower in volume than the canalised stormwater flow.

9.5 Ingress volumes

The total ingress volume, amongst others, is a major control factor governing the rate of rise. However, it does not affect the flooding risk as this is based on the final elevation of the water table and not on how rapidly it may be reached. However, the total ingress volume may be important in identifying suitable interventions which, in turn, may have consequences for the flooding risk. Values given for the total volume of water flowing into the different mine voids of the Central Rand differ significantly ranging from 30 Ml/d to 100 Ml/d. Generally, pumping volumes from active or de-commissioned mines can be used as a proxy to estimate the total amount of inflow into the mine void. Tab. 9.6 indicates pumping volumes for the Central Basin from 1951 to 2008.

Tab**.** 9.6: Annual average pumping rates of GMs in the Central Rand from 1951 to 2009 (source of original data, excluding statistics: Boer et al., 2006)

Mine	DRD	Rand Leases	CMR	Crown Mines	Robinson Deep	City Deep	Other	Simmer & Jack	Rose Deep	ERPM		Total
Shaft	No 5									Hercules	SWV	
1951	no rec	no rec	no rec	no rec	no rec	no rec	no rec	no rec	no rec	11,3		
1952	4,9	3,7	7,1	7,4	4,4	7,4	3,2	4,4	4,5	10,4		57,4
1953	6	4,3	6,4	7,6	4	8,6	2,4	3,8	5,1	10,2		58,4
1954	6,1	3,1	7,6	7,1	3,6	8,2		4	4,4	11,6		55,7
1955	7,9	6,1	6,6	10	4,4	10,1		5,2	5,6	12,7		68,6
1956	7,4	4,3	6	9,2	4,5	9,6		4,1	5,4	11,6		62,1
1957	7	4,4	8,1	9,4	4,8	9,3		4,3	5,9	11,9		65,1
1958	5,7	5,2	5,8	8,7	4,4	7,1		4,4	5,6	12,3		59,2
1959	7,3	3,2	6,8	8,2	4,3	6,8		4,1	3,3	11,9		55,9
1960	no rec	no rec	no rec	no rec	no rec	no rec		no rec	no rec	12,9		
1961	no rec	no rec	no rec	no rec	no rec	no rec		no rec	no rec	12,5		
1962	no rec	no rec	no rec	no rec	no rec	no rec		no rec	no rec	11,2		
1963	no rec	no rec	no rec	no rec	no rec	no rec		no rec	no rec	10,2		
1964	no rec	no rec	no rec	no rec	no rec	no rec		Closure	no rec	11,9		
1965	no rec	no rec	no rec	no rec	no rec	no rec			Closure	no rec		
1966	no rec	no rec	no rec	no rec	Closure	no rec				no rec		
1967	no rec	no rec	no rec	no rec		no rec				no rec		
1968	no rec	no rec	no rec	no rec		no rec				6,9		
1969	no rec	no rec	no rec	no rec		no rec				17,6		
1970	no rec	no rec	no rec	no rec		no rec				19,7		
1971	no rec	Closure	no rec	no rec		no rec				21,4		
1972	8,1		no rec	no rec		no rec				19,6		
1973	8,3		no rec	no rec		no rec				14,2		
1974	9,6		no rec	no rec		no rec				18,3		
1975	11,5		Closure	no rec		no rec				no rec		
1976	18,9			no rec		Closure				16,2		W
1977	18,7			Closure						12,7	33,2	
1978	21,6									13,9	33,4	68,9
1979	16,9									14,6	23,0	54,5
1980	17,2									17,5	27,8	62,5
1981	17,9									18,4	29,0	65,3
1982	17,1									17,9	25,9	60,9
1983	15,8									17,9	24,6	58,3
1984	16									18,8	22,2	57,0
1985	no rec									no rec	no rec	
1986	14,3									19,1	25,6	59,0
1987	17									20,9	33,7	71,6
1988	21,4									no rec	28,9	
1989	no rec									18,4	26,8	
1990	19,2									17,4	19,9	56,5
1991	no rec									no rec	37,6	
1992	no rec									no rec	37,5	
1993	no rec									no rec	39,6	
1994	22,4									no rec	42,7	
1995	18,1									16,0	31,1	65,2
1996	17									no rec	no rec	
1997	18	**17,7**								18,0	16,2	**52,2**
1998	no rec									44,8		
1999	Stopped									no rec		
2000										no rec	no rec	
2001										16,4	33,4	49,8
2002										15,2	36,3	51,5
2003										15,5	38,5	54,0
2004										Stopped	35,0	35,0
2005												
2006											41,3	41,3
2007											39,7	39,7
2008										only 2 months:	54,1	
2009											Stopped 28 Febr. 2008	
n	29	9	8	8	8	8	2	8	8	40	26	26
av.	**13,7**	**4,3**	**6,8**	**8,5**	**4,3**	**8,4**	**2,8**	**4,3**	**5,0**	**15,7**	**32,2**	**57,1**
min	4,9	3,1	5,8	7,1	3,6	6,8	2,4	3,8	3,3	6,9	16,2	35
max	22,4	17,7	8,1	10	4,8	10,1	3,2	5,2	5,9	44,8	54,1	71,6

The change in pumping volume for the CR is depicted in Fig. 9.15

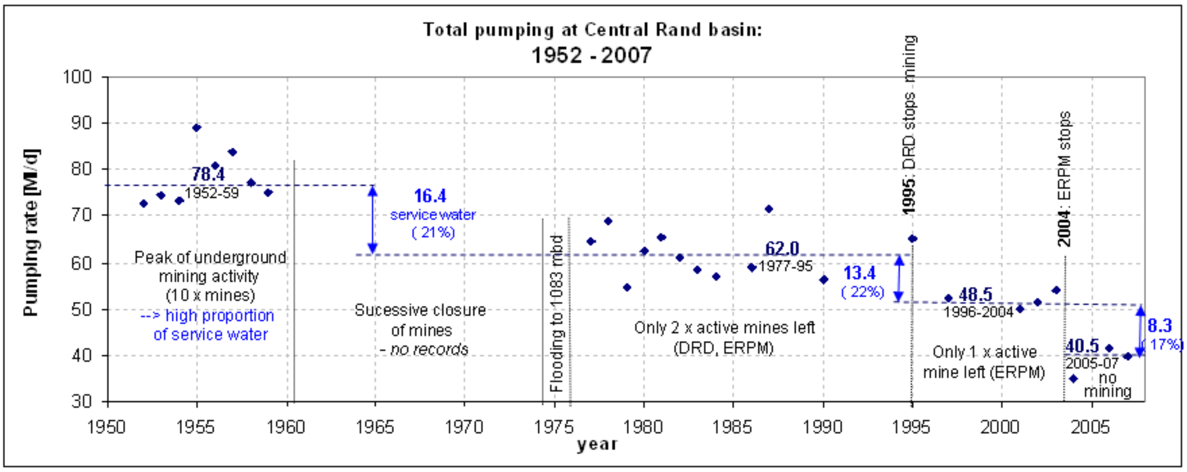

Fig. 9.16: Mean annual pumping rates in the Central Rand based on data in Boer et al. (2006) (dark blue – average pumping volume for the period, lighter blue: difference between long term averages – service water proportion)

In order to interpret the changes depicted above the following events and observations relating to the pumping history are to be considered:

- During the 1950 deep level mining in the CR peaked with 10 x different mines actively developing the underground void. Each mine pumped from their own shafts as the sub-voids were still hydraulically isolated.

- During the late 1960s and the early 1970s most mines in the CR Rand closed. After Crown Mines as one of the largest mines in the CR stopped in 1974 it was decided to flood the deepest part of the mine void. Thereafter all water in the sub-voids east of Rand Leases and west of ERPM was pumped from the SWV shaft, implying that these voids were all hydraulically interlinked, forming what is termed the 'central sub-basin'. The pumping at SWV shaft by ERPM kept the WL constant at 1083 m below datum to ensure that no water from the Central sub-basin could reach a holing where uncontrollable decant into the ERPM sub-void would have occurred jeopardizing continued mining at ERPM.

- DRD stopped underground mining in 1995 but continued pumping water from the linked sub-voids of RL and DRD until 1999.

- After pumping at DRD ceased, the water table rose at DRD and RL until, in July 2001, the WL at Rand Leases (according to Walker and Biljon, 2001) reached a holing to the Central sub-basin resulting in water from RL increasing the pumping volume at SWV. However, the claimed increase cannot be seen in the pumping records (Tab. 9.6). Walker and Biljon also predicted in 2001 that additional DRD water will decant into the Central sub-basin in Sep. 2002 further increasing the pumping volume at SWV shaft by another 17 Ml/d.

Again, the predicted increase is not supported by the pumping data at SWV shaft where pumping increased from 2001 to 2002 by only 2.9 Ml/d – a margin well within in the range of normal inter-annual fluctuations (Tab. 9.6).

However, the fact that the water table at DRD and RL remained constant after decant into the Central basin occurred indicates that all ingress received was passed onto the Central sub-basin. However, the actual volume appears to be much lower than predicted as no corresponding increase at SWV shaft could be observed.

- In 2004 pumping at Hercules shaft stopped allowing water in the ERPM central compartment to rise reducing the total pumping volume in the CR further. From this point on all pumping from the CB was conducted from the SWV shaft.

- In July 2005 water from the Central mines started to flow into the SE Vertical compartment of ERPM) causing the water table to rise at a rate of 0.9m/d to 800 m below surface.

- In order to prevent flooding of the ERPM part where still active underground mining took place a plugging programme was started in Dec. 2005 aimed to extend underground mining at ERMP to 2011. During Dec. 2005 and June 2007 Murry and Roberts installed 5 x new (high pressure mortar intruded) water plugs, upgraded a further 3 existing plugs at SE and Far East Vertical shaft (58/68 level) and thereby hydraulically isolated about 10-15% of the ERPM mine void (Anonymous, 2006).

- Following a gas explosion at the pumping chamber of the SWV shaft in Oct. 2008 pumping from this shaft stopped for good as the pumping chamber flooded soon thereafter.

The pumping volumes for the above-mentioned shafts and the respective proportion of service water for 1994 are compiled in Tab.9.7.

Tab. 9.7: Annual average pumping rates for the 3 shafts where water was pumped from the Central basin in 1994 as well as the proportions of the associated service water (from Scott 1995). Blue: ingress volumes calculated as difference between total pumping and service water portion.

Pumping shaft	Page in report	Total pumping rate [Ml/d]			Added water [Ml/d]	Natural ingress [Ml/d] (= total pumping – service water) (% of pumping)
		Av.	min	max		
DRD 6 #	120 (Tab. 4.3.4) 120 (Tab. 4.3.5) Mine DWA 121 (sketch) 124 (box)	- 18.06 15.96 **16** 15.8			3.09 (17-19%)	14.97 (83%) 12.87 (81%)
ERPM SWV #	120 (Tab. 4.3.4) 120 (Tab. 4.3.5) 121 (sketch) 122 (box)	16.58 16.58 **17** *26.93*	2.8	39.8	2.74 (17%)	13.8 (83%)
ERPM Hercules #	120 (Tab. 4.3.4) 120 (Tab. 4.3.5) 121 (sketch) 123 (box)	16.02 16.02 **16** *19.03*	2.1	22.1	8.93 (56%)	7.1 (44%)
Total CR	120 (Tab. 4.3.4) 120 (Tab. 4.3.5) 121 (sketch)	**49** **49**	**11**	**83**	**15** (31%)	**34** (69%)

Based on the pumping figures in Tab. 9.7 a total ingress volume for the Central Basin of 34 Ml/d can be deducted. This represent some 69 % of the total pumping volume with the remainder (31%) being attributable to service water added by the mines. The percentage of service water varies from 17% at DRD and SWV shaft to over 56 % at the Hercules haft at ERPM. It is also of interest to note that pumping volumes for DRD no. 6# indicated by the mine are somewhat larger than those recorded by the DWA. This may point to possible over-reporting by the mines as governmental subsidies are given based on a percentage of the total pumping volumes resulting in an incentive to report somewhat higher volumes.

Tab. xx also indicates that figures in Scott (1995) are not always consistent as pumping volumes for the SWV vary for example from 16.6 to nearly 27 Ml/d. In order to ascertain which data are applicable the volumes in SWAMP (1996) citing Scott (1995) are listed in Tab. 9.8.

Tab. 9.8: Pumping rates for the 3 shafts where water was pumped from the Central basin in 1994 as well as the proportions of the associated service water (data: SWAMP, 1996). Blue: ingress volumes calculated as difference between total pumping and service water portion.

Mine/ Pumping shaft	**Pumping level** (no) **[mbd]**	**Total pumping rate [Ml/d]**				**Pumped service water [Ml/d]**	**Natural ingress [Ml/d]** (= total pumping – service water)
		Av.	min	max	Max/min ratio		
DRD 6 #	(44) **2380**	**16**	6	21	3.5	3	13 (81%) (DRD: 8; Rand Lease: 5)
ERPM SWV #	(22) **1083**	**17**	3	40	13	3	14 (82%)
ERPM Hercules #		**16**	2	22	11	9	7 (44%)
Total CR		**49**	**11**	**83**	**7.5**	**15**	**34** (69%)

Tab. 9.8 shows that SWAMP (1996) adopted the most frequently given figures in Scott (1995) confirming that some 34 Ml/d are attributable to natural ingress. Ratios between minimum and maximum pumping rates vary from 3.5 to 13. As pumping volumes depend on a number of factors including availability of pumps (pump failure may occur), pumping regimes (more is pumped at weekends and at night time when electricity is cheaper) etc. these ratios do not necessarily reflect the variance in the volume of natural ingress.

Predicting the future decant volume

Predictions regarding the rate at which mine water will flow out of the mine void are commonly based on the assumption that the decant rate will be equal to the water flowing into the mine (ingress rate) during the flooding phase. The ingress rate, in turn, is usually determined using rates at which mine water was pumped out of the void in question. This approach has a number of potential shortcomings including the following:

(1) Ingress during the flooding of the void is generally larger than after flooding is complete since the rising mine water table in the void is prone to neutralise a number of low-lying ingress sources as it rises above their elevation. This reverses the hydraulic gradient that used to drive the ingress into the mine void and may allow for mine water to recharge former ingress sources. This is particularly likely to affect low-lying streams, dams etc. which used to lose water into the void as well as parts of the fractured aquifer located in stream valleys, depressions and other low-lying areas in the natural relief. Apart from rising *above* certain ingress sources it will generally be enough to just reach the level of an aquifer, for example, in order to prevent any further ingress as a hydraulic equilibrium will be established where the groundwater can no longer flow into the (now full) mine void. As a result the fresh unpolluted groundwater

may flow over the polluted and therefore somewhat denser mine water below. Depending on the degree of mixing either caused by thermal convection driving warmer water from deeper parts of the mine void to surface or possible turbulences caused by water pouring into the void from high-lying ingress sources, the skimming of clean groundwater over the filled void in places where the mine water table recovered to the aquifer level may result in less dense clean water permanently topping the polluted mine water (termed ‘stratification’). In that case the quality of the decanting water may improve over time. The degree to which low-lying ingress sources are cut-off by the rising mine water table depends of course on the elevation of the final decant point. It also determines which parts of the relief will remain above the mine water level and thus able to continue with recharging the below void. As all ingress is ultimately derived from sources located at or just below surface, the final decant volume will increasingly get less the closer the water table in the mine void rises to surface. By implication that means that any intervention aimed at keeping the mine water level artificially low will keep ingress volumes and associated pumping and treatment cost higher than incurred at natural decant levels. In order to cut off as much surface ingress as possible it may even be desirable to plug low lying outflow points such as shafts and thereby allow the mine water table to naturally reduce decant volumes saving long-term treatment costs.

Furthermore, ignoring the natural cut-off of ingress will result in overestimating the decant volume. As many currently proposed treatment options require minimum throughflow in order to be economically feasible, such overestimation could well jeopardise the viability of projects and result in lost investments. With only one mining basin (the West Rand) being completely flooded, examples to prove the suggested reduction in ingress are rare. According to Cousins (1978) REGM and West Rand Cons. pumped 40.8 Ml/d and 44.3 Ml/d respectively from the Western Basin totalling 85.1 Ml/d. As this was done as part of efforts to dewater flooded sections this volumes has to be higher than the natural ingress as otherwise no lowering of the mine water level could be achieved. Cousin (1978) estimated that pumping volumes at REGM and West Rand Consolidated will be reduced by mid to end 1978 to 15 Ml/d and 30 Ml/d respectively. It is assumed that these pumping rates were regarded as being sufficient to keep up with the ingressing water and prevent renewed flooding of the mine void. As such the combined pumping rate of both mines of 45 Ml/d is likely to present the approximate rate of ingress. As surplus water was at the time naturally available for underground mining purposes, it is uncertain whether the pumping rates included added service water. Assuming a possible service water proportion of maximal 5 Ml/d results in an ingress rate prior to flooding of some 40-45 Ml/d. This compares to a long-term average of some

15-20 Ml/d of decanting mine water after the mine void filled up, suggesting that the natural cut-off of ingress sources through the rising mine water table ***reduced the rate of inflowing** water **by** approximately **60%***. Owing to the relatively low level of the decant point at BRI the mine void in the WB could never fill up to surface leaving some estimated 12% of the void volume dry. This also means that a number of ingress sources could not be cut-off and continue to provide ingress which feeds the decant. Applying the reduction rate observed at the WB to the CR, for which a natural ingress volume of 30-40 Ml/ is proposed, a decant rate of 18 to 24 Ml/d is to be expected. Should the mine water table in the CR recover to relatively higher levels that in the WB (i.e. leaving less than 12% of the mine void dry) the reduction could well be higher, while it will be smaller if a higher percentage of the mine void remains dry. Another aspect to consider in this context it that fact that very deep-reaching aquifers exists in the WB (associated with deeply weathered dolomite extending up 80 m below surface) which are cut off relatively early by rising mine water. Such deep-reaching aquifers have not been reported to be present in the CR suggesting a less pronounced natural reduction of ingress compared to pumping.

(2) Another major potential pitfall is that the contribution of service water which is pumped underground for mining purposes is not subtracted from the total pumping volume and thus included in determining the ingress. As decant per definition only occurs from completely flooded – and thus abandoned - mine voids where underground mining takes no longer place, the input of service water will no longer contribute to the decant. Ignoring the contribution of service water to the total pumping rate will result in overestimating the future decant volume.

(3) As pumping figures are crucial for estimating ingress volumes it is important to understand their accuracy. In most mines pumping volumes are determined by the time (duration) the various pumps are operated multiplied by the pumping specific pumping rate given for each pump. This approach ignores that pumping efficiency generally decreases over time resulting in lower pumping rates per time unit and thus in a general overestimation of actually pumped volumes. Compared to the cut-off of natural ingress sources and ignoring the contribution of service water the resulting overestimation is believed to be relatively small.

(4) As pumping costs of mines are frequently subsidised by government as a proportion of the total pumping costs incurred, a principal monetary incentive existed for subsidy receiving mines to somewhat inflate the true pumping costs in order to get larger subsidy. Comparing calculated pumping rates based on original pumping hours data (provided by DRD), with reported data to DWA

indicated an over-reporting margin of 23% (39.7 vs. 30.5 Ml/d). Scott (1995) cites an example where pumping values given by the DWA are lower than those reported by the mines (18 vs 16 Ml/d).

Based on the above it can be concluded that predictions of decant volumes which are based on ingress and pumping volumes before and during flooding tend to overestimate the volumes of the future decant.
In the following an attempt is made to arrive at a reasonable estimate of the decant rate for the central and western part of the CR basin stretching from DRD to Rose Deep. This involves the following steps:

(1) Establish pumping volumes during and after mining and estimate the natural ingress by deducting water inputs related to underground mining.

(2) Determine what proportion of the mine void will remain dry at the decant level of 1614 mamsl to arrive at a possible decant reduction rate based on observations in the WB.

(3) Determine which ingress sources will be cut off at the expected decant level.

(4) Investigate an optimum decant level at which the decant volume is minimal while risks of basement flooding and geotechnical instabilities as well as possible diffuse mine water seepage into streams are kept at an acceptable level.

In a first step pumping volumes for this part of the void are determined (Tab. 9.9 and Tab. 9.10).

Tab. 9.9: Pumping from the 3 x sub-basins of the CR[7] during active underground mining (peak period) (raw data: Scott, 1995)

	Durban Roodep. Deep	Rand Leases	DRD sub-basin	Cons. Main Reef	Crown Mines	Robins. Deep	City Deep	Other	Simmer & Jack	Rose Deep	Central sub-basin		ERPM sub-basin
	DRD	RL	DRD+RL	CMR	CM	RD	CD	other	S&J	RD	(CMR ...RD)		(Hercules #)
1952	4,9	3,7	***8,6***	7,1	7,4	4,4	7,4	3,2	4,4	4,5	***38,4***		***10,2***
1953	6	4,3	***10,3***	6,4	7,6	4	8,6	2,4	3,8	5,1	***37,9***		***11,6***
1954	6,1	3,1	***9,2***	7,6	7,1	3,6	8,2		4	4,4	***34,9***		***12,7***
1955	7,9	6,1	***14***	6,6	10	4,4	10,1		5,2	5,6	***41,9***		***11,6***
1956	7,4	4,3	***11,7***	6	9,2	4,5	9,6		4,1	5,4	***38,8***		***11,9***
1957	7	4,4	***11,4***	8,1	9,4	4,8	9,3		4,3	5,9	***41,8***		***12,3***
1958	5,7	5,2	***10,9***	5,8	8,7	4,4	7,1		4,4	5,6	***36***		***11,9***
1959	7,3	3,2	***10,5***	6,8	8,2	4,3	6,8		4,1	3,3	***33,5***		***12,9***
av.	**6,5**	**4,3**	**10,8**	**6,8**	**8,5**	**4,3**	**8,4**	**2,8**	**4,3**	**5,0**	**37,9**		**11,9**

For the other two basins in which active underground mining continued, pumping volumes included service water which needs to be subtracted to arrive at the volume of pumped water derived from natural ingress. For ERPM Scott (1995) gives service water pumping rates as 12 Ml/d resulting in a natural ingress into the ERPM sub-void of 6.5 Ml/d.

[7] During the 1950s there were, strictly speaking, no sub-basins as pumping was conducted separately by each gold mine. This changed in 1975 when the deepest part of the Crown Mines – as the largest and longest producing mine in the Central sub-void - was flooded from 49 to 24 level. After the 2-year flooding period pumping resumed in October 1976 from the SWV shaft which – although on the ERPM lease area - pumped water from the Rose Deep sub-void. Even though faulting and collapse of haulages reduced the hydraulic connectivity between Rose Deep and the western sub-voids of the central sub-basin and allowed for the build up of an hydraulic head of some 90 m pumping at SWV shaft was since then able to control the water table from CMR in the west to Simmer & Jack in the east right next to Rose Deep. Thus, it can be assumed that water can move laterally across all former mine void boundaries to such extent that this part of the void is commonly regarded to act as a single hydraulic entity – termed Central sub-basin.

Tab. 9.10: Pumping rates for shafts in the 3 sub-basins of the Central Rand void system for the period 1977-2007 (sources of original data, Ferret Mining, 2004) (empty cells – no record available)

Sub-basin	*DRD-RL*	*CMR-RD*	*ERPM-sub-comp*	*Central Rand*
Shaft	**DRD #6**	**SWV #**	**Hercul #**	**Total**
year	Ml/d	Ml/d	Ml/d	Ml/d
1977	18,7	33,2	12,7	64,6
1978	21,6	33,4	13,9	68,9
1979	16,9	23,0	14,6	54,5
1980	17,2	27,8	17,5	62,5
1981	17,9	29,0	18,4	65,3
1982	17,1	25,9	17,9	60,9
1983	15,8	24,6	17,9	58,3
1984	16	22,2	18,8	57,0
1985				
1986	14,3	25,6	19,1	59,0
1987	17	33,7	20,9	71,6
1988	21,4	28,9		
1989		26,8	18,4	
1990	19,2	19,9	17,4	56,5
1991		37,6		
1992		37,5		
1993		39,6		
1994	22,4	42,7		
1995	18,1	31,1	16,0	65,2
1996	17			
1997	18	16,2	18,0	52,2
1998			44,8	
1999	stopped 1999			
2000				
2001		33,4	16,4	49,8
2002		36,3	15,2	51,5
2003		38,5	15,5	54,0
2004		35,0	stopped 2003	35,0
2005				
2006		41,3		41,3
2007		39,7	*(30,6*)*	39,7
2008		stopped 11/2008		stopped 11/2008
2009		** calculated based on original pump hour data at SWV shaft*		
period	1977-97	1977-2007	1977-2003	1977-2007
n	16	25	18	19
av.	***18,0***	***31,3***	***18,5***	***67,9***
% CR	27%	46%	27%	***100%***
period	1977-1997	1977-1995	1977-1997	1977-1997
av.	***18,0***	***30,2***	***17,3***	***65,5***
% CR	28%	46%	26%	***100%***
period	1997-2007	2002-2007	2001-2003	2001-2007
av.	***0,0***	***38,2***	***15,7***	***53,9***
% CR	0%	71%	29%	***100%***

empty cell - no record   red - outlier

grey shaded cells - last (full) year for which pumping records are availabe before pumping stopped

The average for Jan. and Febr. 2008 at SWV shaft was 54.1 Ml/d suggesting that the 2 x wet months display a higher average pumping rate than the long-term annual average and is thus not given as an annual average.

Tab. 9.10 list the pumping volumes for 3 x different shafts each pumping from a different sub-basin which together make up the CR basin. For estimating the decant volume only the pumping figures for the DRD 6# and the SWV # are relevant since the Hercules # pumped water from the Far Eastern compartment of ERPM which has been hydraulically disconnected from the rest of the CR following a plugging programme (2005-2007).

With some 12.1 Ml/d of the water pumped at Hercules shaft being service water (Scott, 1995) the *natural ingress into the ERPM sub-void* can be estimated as the difference between the recent pumping average and the service water input resulting in some *3.6 Ml/d* (=15.7–12.1 Ml/d). This rate needs to be added to the pumping from the central sub-void to arrive at the total ingress over the entire CB.

The selected period displayed in Tab. 9.10 started in 1977 as at this point in time all deep level gold mining had stopped in the central sub-void and water had flooded the lowest part of all sub-voids to 24 level (1083 m below datum). The mine water was kept at more or less this level for the next 30 years through pumping via the SWV shaft. Since no service water was added during this time the pumping reflects largely the (natural and man-made) ingress. Although physically located on the ERPM mine lease area, the pumping chamber of this shaft is connected to the Rose Deep sub-void via a pipe, penetrating installed pressure plugs. As the water pressure from the Rose Deep side should not exceed a certain safety limit, plug pressure was measured continuously, guiding the pumping rate at SWV shaft. Following a methane gas explosion in October 2008 that caused a fatal accident, pumping from SWV shaft has stopped in November 2008 leaving the pumping chamber to be flooded. The water level at Rose Deep soon reached an uncontrollable inflow level, flooding most of the ERPM sub-void. The isolated part around the Far East ERPM compartment being isolated from all the other mining compartments is presently the only void being dry. (Labuschagne, 2011).

Apart from an average pumping rate given for the entire period (1977 to 2007), values for 2 x alternative periods are indicated (Tab. 9.10). The latter include the period from 1977 to 1995/7 during which DRD was still mining actively underground i.e. service water was included in the pumping volumes. The average pumping rate for DRD over this period is 18 Ml/d which is about 2 Ml/d above the rate indicated by Scott (1995). Although DRD stopped mining in 1995 pumping at DRD 6 # continued until 1999 keeping the water level in the DRD-RL sub-void at a constant level. When pumping from DRD 6 # stopped in 1999, the water level in the DRD sub-basin began to rise until it reached a holing in ***2002,*** via which the mine water decanted into the central sub-void. This means, that the water pumped from SWV shaft between 2002 and late 2008 to keep the water table constant and below the uncontrollable overflow into ERPM, reflects the ingress rate over the entire central basin from DRD to Rose Deep. This pumping rate together with the natural ingress estimated for the ERMP sub-void (3.6 Ml/d) is therefore a first order approximation for the expected maximum future decant rate. Pumping figures for SWV shaft that include the ingress contribution from the DRD/RL sub-basin are available for a 7 year period (2001-2007 with the annual average for 2005 missing) indicating a mean annual pumping rate of 37.4 Ml/d (Tab.

9.10). Together with the estimated ingress into the ERPM sub-basin of 3.6 Ml/d (which is not included in the pumping from SWV shaft) this suggests a total ingress into the entire Central Basin of some 41 Ml/d.

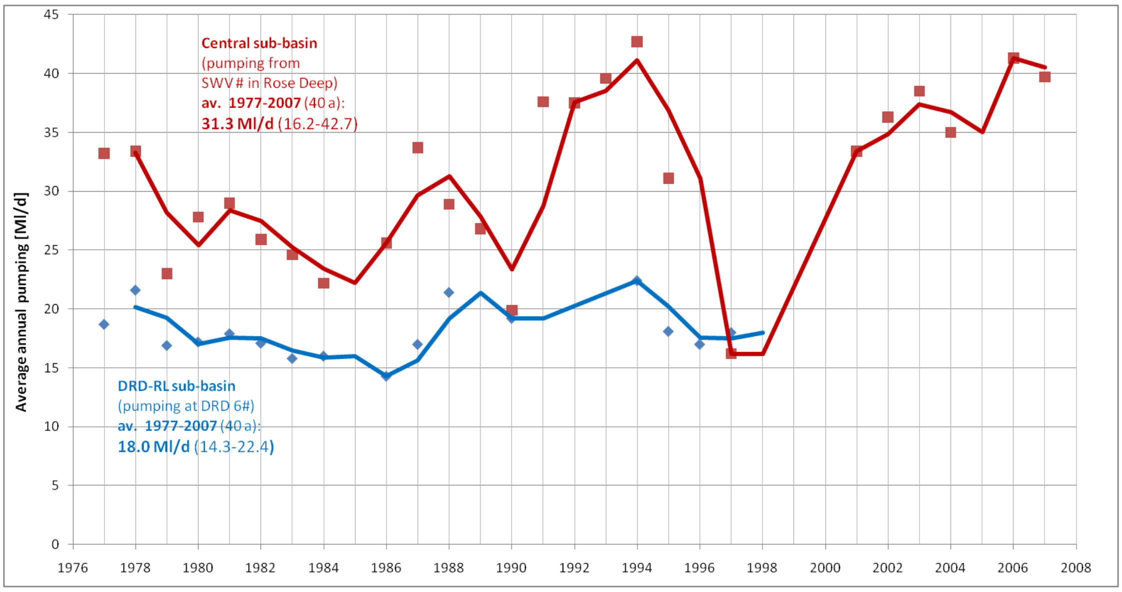

Fig. 9.17: Annual average pumping rate (squares and diamonds) and associated moving averages for the DRD-RL sub-basin (pumped at DRD 6#) and the central sub-void

The similar time series for annual average pumping volumes at the DRD-RL and the Central sub-basins depicted in Fig. 9.16 may point to a common factor controlling pumping rates. As over long periods of times the pumping rates do reflect ingress volumes, this common factor relates to varying intensities of ingress.

While the overall shape of the two graphs is similar, there is a pronounced difference for the period 1990 to 1994 when pumping from the Central sub-basin more than doubled while it increased only slightly at DRD-RL (Fig. 9.16). Since tailings reclamation started in the central area of the CB in the mid-1980s, chances are that this increase may be indicative of intensified tailings mining not only importing large volumes of water needed for the hydraulically mining of the old tailings into the area but also dumping the re-processed tailings together with process water underground.

With no service water for underground mining contributing to pumping, and no decant from either of the 2 x sub-basins to the west (DRD/ Rand Lease) and the east (ERPM), pumping for this period reflects the natural ingress rate into the central sub-basin being 31.3 Ml/d (Tab. 9.10). Comparing this to an average of 37.9 Ml/d pumped during 1952 and 1959 when all mines in the central sub-voids were still actively mining underground, the resulting difference of 6.6 Ml/d (= 17.4%) can be attributed to service water used in underground operations.

9.6 Dynamics and controls of mine flooding

9.6.1 Structural controls on the inter-mine water flow

The actual mine void consists of long-lived vertical structures such as shafts (including sub-vertical and ventilation shafts) connecting the surface to the underground workings and lateral components underground that provide access to the mined reefs through haulages, drives, cross cuts etc. The later structures are commonly designed for a life-span of around 40 years (Malan, 1999). All vertical and lateral access structures together account for approximately 10% of the total mine void volume, with the remaining volume made up by so-called ‘stoping areas’. Aiming to minimize the mining of non-paying rock, these stoping areas are just high enough for workers and machinery to access the reefs which often are only a few decimetres thick. The average stope height ranges from 1.1 m to 1.4 m may however widen to up to 4 m at so-called ‘pay-shoots’ where the reef horizon widens. Compared to the vertical and lateral access structures, stopes are less permanent and tend to close up (‘set’) over time owing to the large pressure of the overburden rock. The rate of closure generally increases with mining depths (i.e. rising height of the rock column and the associated pressure) and may finally reduce former stopes to barely recognisable mere ‘pencil lines’ over a period of just a few months in deeper parts of the mines. In the Witwatersrand mines rapid and nearly complete stope closure starts to occur from mining depths greater than 1500 m below surface. This process appears to be initiated by a combination of brittle and plastic deformation during mining, hence the need for stope support, followed by plastic deformation of the highly stressed rock in the form of bulk expansion. I.e., instead of brittle rock crumbling and falling from the hangingwall into the underlying void the quartzite gradually expands in a flow-like manner termed ‘creep’. This process of plastic rock deformation fills the original stope volume nearly completely, while it simultaneously reduces the hydraulic conductivity of the former stope area back to zero. Since stopes account for approx. 90% of the total mine void volume, at mining depths greater than 1500 m a significant part of the void volume is continuously lost already during active mining. Stope closure can be expected to be complete a few months after the cessation of active mining. Assuming an average depth of the Central Rand mine void of some 2800 mbs (where ore extraction starts right at the surface in contrast to other gold fields such as the Far West Rand where more than a kilometre of non-auriferous rock overlies the mined Au-reefs), some 60% of the total void is located below the critical depth (assuming a homogenous void shape over the full mining depth. Owing to the widening of the void in the upper most 50 m bs – to be explained later - this is a slight overestimate). With stopes accounting for 90% of the void volume the plastic filling of stopes is expected to have reduced the original void volume in the Central Rand by more than half (54 % = 243 Mm^3). Since most deep mines have ceased operations in the Central Rand several decades ago and the last mine (ERPM) 3 a ago, the reduction of the deeper void volume is assumed to be complete by now. This would result in a remaining void volume of 207 Mm^3 mainly located in the upper part of the void above the critical depth of 1500 mbs.

However, also the upper part of the void is likely to be affected by structural changes associated with the collapse of stoping areas as well as that of more permanent structures such as haulages and drives. Owing to the somewhat reduced pressure of the overburden, this is likely to happen not as plastic deformation but through mechanical collapse of the hanging wall in the form of rock falls etc. ('elastic setting'). While the associated filling reduces the hydraulic transmissivity in the underlying stopes, it is likely to increase transmissivity in the hanging wall above the stope owing to the formation of tension fractures and cracks caused by the elastic setting of the rock. It stands to reason that the total volume of cracks and fractures created in the hanging wall will be equal to the original stope volume filled, resulting in an overall unchanged total void volume in the upper part of the mine void (i.e. the part above the critical depth of 1500 m). The net effect in terms of the ability of the mine void to conduct water flow is likely to be negligible as reduced flow in the stopes are counteracted by increased flow in near-stope host rock fractures. The development of a so-called 'fracture-envelope' surrounding the mine void caused by explosions and mechanical mining may have further increased the transmissivity of host rocks. However, with the filling of the stopes and collapse of haulages and drives, the maximal flow speed is presumably reduced compared to conditions during active mining, as completely open channels no longer exist. With water moving mainly through cracks, fractures and fissures the mine void will finally resemble a fractured aquifer perhaps with a few preferential pathways developed along the less-complete closed access structures such as shafts, haulages and drives. This is also assumed to apply to the more permanent structures in the deeper part of the mine below the critical depth of 1500 mbs. Since all the access structures have been designed for a life span of 40 years it is to be expected that many of the lateral water pathways associated with haulages and drives are by now collapsed in those mines that closed 40 years ago or earlier (i.e. in or before 1971). Out of the 13 mines that created the Central Rand void this applies to 8 mines. Of the remaining 5 x mines 3 x closed between 1975 and 1977 rendering their youngest structures very close to the expected life-span of 40 a while the older ones have reached it long ago. Only the two mines at the extreme ends of the Central Rand void system, DRD in the west and ERPM in the east, have closed more recently allowing for rapid lateral flow in open channels (haulages) to continue for another 17 to 37 years (from now: 2011) respectively.

Owing to the above described changes in hydraulic properties, Scott (1995) replaced the term 'mine void' by 'mining aquifer' implying that the final void structures more resemble a confined, fractured rock aquifer than a network of vertical and lateral tubes. However, after Scott (1995), increasingly the term 'basins' were used to refer to the systems of interlinked mine-voids in the different gold fields of the Witwatersrand wrongly implying a rather homogenous underground reservoir in which water moves freely from one sub-void to the other[8]. The possibility of a drastic reduction of the mine void volume and associated changes in hydraulic properties – to our knowledge - has to date not been considered. If indeed applicable, it has however a number of consequences that are important for predicting future processes such as rates of rise of the mine water table as well as the final volume and water quality of the out-flowing mine water.

In the case of the Central Rand, in 1889, some 52 mining companies were registered. However, owing to the technological challenges in mining and extracting gold from the deeper, hard un-oxidised ore and the associated capital requirements, soon a consolidation process took place (sometimes reflected in the names of the new mines such as in 'Consolidated Main Reef') leaving only a few mines that could continue to extract ore at greater depths. For the first time these so-called 'deep' mines (which often added the distinguishing 'deep' to their names as in 'Durban Roodepoort *Deep*', 'Robinson *Deep*', 'City *Deep*', 'Geldenhuis *Deep*'; 'Rose *Deep*') used vertical and sub-vertical shafts to follow the steeply dipping ore to ever greater depths, finally reaching between 2000 m (Nourse Mines) and over 3400 m (ERPM) below surface. Some 85% of the total mine void in the Central Rand goldfield have been created by just 13 x mines (for the remaining 15% no detailed record could be found). Each of these companies created a mine void in its own right which, in this report is termed 'sub-void'. The volumes of the different sub-voids created by these deep mines are listed below in geographical order from Roodepoort in the east to Boksburg in the west including their operational period (the volume of void is given in million cubic metres as calculated from tonnage of milled ore given in Scott, 1995, the possible reduction of void volume trough plastic setting is not considered):

- DRD (1898-1994: 45.5 Mm^3; 1999: 53.6 Mm^3),
- Rand Leases (1936-1971: 25.7 Mm^3),
- Consolidated Main Reef; CMR (1889-1975: 31.2 Mm^3),
- Crown Mines (1897-1977: 78.8 Mm^3, then one of the world's largest mines, plus the sub-void of Langlaagte Estates; 1888 – 1946: 19,4 Mm^3; total: 98.7 Mm^3),
- Robinson Deep (1898-1966: 23.8 Mm^3),
- City Deep (1910-1976: 30.7 Mm^3, commonly lumped together with the sub-void of Nourse Mines;1896-1948: 12.9 Mm^3, total: 43.7 Mm^3),

[8] Sub-voids, in this report, are underground mine workings created by a gold mining company, which were later connected via haulages, boreholes, escape shafts etc. to neighbouring sub-voids forming a large, interconnected mine void.

- Simmer & Jack (1888-1964: 26.7 Mm^3 with Geldenhuis Deep; 1895-1947: 13.5 Mm^3 forming a sub-void (total: 40.2 Mm^2),
- Rose Deep (1897-1965: 18.4 Mm^3),
- ERPM (1894-1995: 68 Mm^3; Whymer (1999): 191.5 Mt; calculated for 2006: 87.5 Mm^3) assumed to be linked to the sub-void of the Witwatersrand Gold Mine west of Rose Deep, 1889-1953: 12.5 Mm^3, total: 100 Mm^3 = 21.4 % of total void volume in CR)

9.6.2 Characteristics of sub-basins

In many cases adjacent sub-voids have been hydraulically linked to each other through so-called ‘holings’ consisting of boundary haulages, escape routes, boreholes, ventilation structures, mined-through boundary pillars, etc. allowing water to flow from one sub-void to the next. The ease at which water moves across mine boundaries is known as ‘hydraulic interconnectivity’ and governs the time it takes for water table differences in two linked sub-voids to disappear (i.e. the 2 levels to equalize). For the 9 x sub-voids stretching from CMR in the east to Rose Deep such a high degree of hydraulic interconnectivity is assumed, that the water can ‘move freely’ (Scott, 1995) across void boundaries forming a single hydraulic entity termed ‘sub-basin’ (SWAMP 1996). For the Central Rand mine void SWAMP (1996) proposed the existence of three sub-basins (E to W):

(i) DRD-Rand Lease sub-basin

This sub-basin consists of the DRD and Rand Lease sub-voids which are, however, separated by a syenite dyke acting as boundary pillar that maintained a hydraulic head between the two sub-voids of 1180m (Scott 1995). It is supposedly only regarded as a sub-basin by SWAMP (1996) since a borehole through the syenite dyke (at 1413m bd) allows mine water from the Rand Lease void to flow into the DRD sub-void from where it was pumped to surface for an extended period of time. The DRD sub-basin is separated from the adjacent CMR sub-void in the E by an 18m-wide boundary pillar. This pillar is reported to have 2 x holings at 908 and 600 mbd potentially connecting it to the adjacent CMR sub-void in the east as soon as the water table on either side of the pillar rises above these levels.

(ii) Central sub-basin

The Central sub-basin consists of 9 x different sub-voids stretching from CMR to Rose Deep and includes the sub-voids of Crown Mines, Robinson Deep, City Deep and Nourse Mines, Simmer & Jack and Geldenhuis Deep resulting in a uniform water level of 1083 mbd (1994) over more than 20km from CMR to Simmer & Jack. One exception is Rose Deep where, in 1994, the water table was 87m lower owing to the collapse of boundary haulages caused by faulting (inferred from a sketch in Scott 1995 where an arrow indicates a wrong pillar). According to this sketch the Rose Deep sub-void will be linked to the reminder of the Central sub-basin at a level of approximately 700 m bd. The sketch also indicates that water pumped at the South West Vertical shaft (which is an ERPM shaft) is in fact not drawn from ERPM but from the Rose Deep sub-void. As mentioned before the western boundary of the Central sub-basin is formed by an 18m-wide pillar of unmined rock (quartzites) which, in 1994, was able to maintain an hydraulic head between Rand Leases and the Central sub-basin of 117 m (1200 vs. 1083 mbd). Less detailed information is available as to the nature of the eastern boundary to the ERPM/ Wits Gold sub-voids. From the sketch in Scott (1995) a boundary pillar is inferred which, however, is shown in such a way that it could only separate the Central sub-basin from the neighbouring ERPM sub-void to a level of approximately 1000 mbd. Once water at either side of the pillars exceeds this level the Central sub-basin would be linked to ERPM sub-void (except the plugged part around FEV shaft).

(iii) ERPM sub-basin and the plugging programme

This sub-basin consists largely of the sub-void of ERPM and – but this is somewhat uncertain and only inferred from other sources – the sub-void of the Witwatersrand GM. The ERPM part of the sub-basin is further sub-divided into smaller compartments (confusingly sometimes also referred to as ‘basins’). One of these compartments is called the ‘Central Compartment’ (ERPM EMPR, 2001) or ‘Hercules basin’ (Anonymous, 2006) named after the shaft that pumps water out of this compartment to surface. Up to a level of 2100 mbd this compartment is hydraulically separated from the rest of the ERPM sub-void. In order to ensure continued mining in the eastern parts of the ERPM sub-void (i.e. the South East Vertical and Far East Vertical basins) water was pumped into the ‘Central Compartment’ where the pillar maintained an hydraulic head of 1500 m over several decades while keeping the deepest part of the mine void dry. From here water was pumped further to the surface (at a rate of 16 Ml/d) via Hercules shaft and subsequently discharged into the Elsburg Spruit (Scott. 1995). The Far East Vertical compartment receives 1.6 Ml/d of ingress plus 3.5 Ml/d of services water pumped underground for mining purposes. In September 2004 the Central sub-void (measured at Rose Deep) was flooded to 24 level resulting in a considerable hydraulic head towards the SE-Vertical and Hercules basins, both only filled with water to 50 level (est. head of some 780 m). Due to the rise in water level in the central sub-void following the stopping of pumping at DRD, it was expected that water from the Central sub-void will cascade via an uncontrolled overflow 1092 mbd first into the Hercules compartment and the South East Vertical basin, and on 42 level, further into the Far East Vertical basin. This was expected to happen in July 2005 (Anonymous, 2006). As such decant would have jeopardised the active underground mining at ERPM still ongoing at the time a plugging programme was embarked on aimed at closing all hydraulic connections through which water from the Central sub-basin could have flown into the actively mined Far Eastern compartment at ERPM. At a total costs of R 29.1 million (of which the first and second project phases were paid for by government) 5 x new 22.5 m-long bulkhead plugs were installed at SE Vertical shaft at 42 level as well as 2 x existing plugs at 58 level and one at 68 level at the Far East Vertical shaft upgraded. The latter plugs vary in length from 21 to 40 m with the plug at 68 level required to withstand a hydraulic head of 2841 m (=28.41 MPa) (Anonymous, 2006). In order to prevent decant from the central sub-basin during the installation of plugs (Dec. 2005 to June 2007) pumping from the Rose Deep sub-void via SWV shaft continued. Associated costs were R1.5 m per month of which government (DME) covered R 1 m. Over a period of 19 months this totals another R 28.5 million bringing the total costs for the plugging programme to 56.7 million. Despite the considerable costs, much of it paid by public funds, in 2008 ERPM decided to stop underground mining following a gas explosion that killed a worker. Consequently pumping ceased from SWV shaft and all pumping equipment was subsequently flooded (Labuschagne, 2011).

Since then the water level in the central sub-void rises continuously prompting a number of frequently sensationalizing reports on the associated consequences for JHB and the environment. In this context it appears counterproductive that large sums and efforts went into a plugging programme whose only effect now appears to be that is has accelerated flooding of the Central sub-void due to hydraulically disconnecting the part of the void volume at ERPM (an estimated 10-15% of the total ERPM volume) which otherwise could have somewhat delayed the flooding. With a volume of 9000-13,000 Ml and an ingress rate of some 30 Ml/d the filling of this part of the now hydraulically isolated compartment could have delayed the decant by another 300 to 433 days.

Lateral intra-void water flow: In instances where the water level in one sub-basin reaches a holing linking it to another this will result in water flowing from the higher elevation in the ‘fuller’ (= decanting) sub-void to the lower elevation in the les full (= receiving) void. Depending on the rate of flow the holing allows this will (temporarily) slow down the rate of rise in the decanting sub-void and increase the rate in the receiving sub-void. Where the hydraulic conductivity of the connecting holing is equal or larger that the rate of ingress in the decanting sub-void this will result in a stagnant water table in the decanting void and relatively rapid increases in the receiving sub-void until the levels in both voids have equalized. Where the inter-void flow is limited through restricted hydraulic conductivity of the holing the water table in the decanting void will continue to rise albeit at a slower rate than before the decant. This type of interconnectivity-related equalisation has a profound impact on the dynamics of water levels in the different mine voids and needs to be understood before reliable predictions of future rates of mine water level rises can be made.

Based on the mine void model presented by Scott (1995) it is assumed that all sub-basins will eventually be hydraulically interlinked at a mine water level of 908 m bd. This is the level of the lowest holing through the 18m-wide boundary pillar that separates the Central sub-basin from the DRD sub-basin in the west. Well before this level is reached, the Central sub-basin and the ERPM sub-basin to the east have been linked via overflowing the separating boundary pillar that ends at around 1000 mbd. If the hydraulic conductivity of this holing is high enough to accommodate all of the resulting lateral water flow between the sub-basins then the water table across the entire Central Rand mine void should be at the same elevation (provided the latter is given in mbd and not in m bs as the surface elevation varies). In such a scenario a single low-lying outflow point with sufficiently high capacity to accommodate the total ingress volume (e.g. any point such as a shaft or river valley allowing some 30-90 Ml/d to pass through it) would be able to control the water level in the entire mine void.

Since it is known that the groundwater table in natural porous aquifers is a subdued reflection of the overlying topographic surface, it should be explored to what degree differences of topographic surface elevation impact on the water levels in the different sub-voids. In a scenario where the water in an inter-connected mine void (basin) flows towards a single decant point a hydraulic head from high-lying ingress areas towards the low-lying outflow point will still develop. According to the surface elevation profiles in the Central Rand, such an outflow point could be either a low lying open shaft (i.e. not capped) in the eastern part of ERPM that is directly connected to the mine void, or a river valley cutting through the void that is connected via fractures, faults or dykes to the void. In this case the outflow point would be at the extreme south-east corner of the mine void, drawing all the decanting water from the N and W. In such a scenario the water table across the entire Central basin would be flat, while gently sloping towards the south eastern corner of the mine void. The degree of this slope depends on the hydraulic interconnectivity of the different sub-voids and will be flatter the larger the hydraulic interconnectivity is in comparison to the ingress rate. For karst aquifers in the Far West Rand with nearly free flowing conditions in underground karst channels and conduits, a slope of 1:1250 was found (i.e. 1m water table difference over 1,25km). For the 44 km strike length of the Central basin this would result in a hydraulic head of some 35.2 m between the high lying ingress area at DRD in the W and the outflow point at ERPM in the SE.

However, since several low lying outflow points are scattered along the E-W profile, it is perhaps also possible that the mine water table will be controlled by more than one decant point even if the main outflow point is large enough to accommodate all flow (like a shaft for example). It is likely that at all locations where localised hydraulic gradients allow for mine water to flow from the void via fractures or other conduits to lower lying surface areas this will happen, even though a larger, lower lying outflow points exists elsewhere. Especially in low-lying former ingress areas where streams crossing the mined outcrop zone lost water to the underlying void, such a diffuse type of decant is likely to happen since the required hydraulic connection to the mine void already exists, and only the direction of flow is now reversed. This possibility needs to be considered when 'controlled-decant' scenarios are devised.

9.6.3 Hydraulic effects of the shallow mining zone of the MR outcrop

Once the shallow, oxidised ore was mined out through open-cast or trench mining supported by shallow incline shafts, many of the original mines did not have the capital to follow the much harder, un-oxidised reefs in depth. The zone affected by the early near-surface mining is approximately 70-120m wide and 30–50 m deep and follows the outcrops of the Main Reef and the Main Reef Leader ore-bodies over the entire strike length of approximately 44 km. With open pits and trenches created by surface diggings having subsequently been refilled with largely unconsolidated material such as mining residues and even municipal waste in places, much of the mined Main Reef outcrop zone shows exceptional high infiltration potential. Infiltration is further augmented by the valley-type shape of much the zone (especially near the CBD), which results in the collection of rainwater runoff from adjacent slope areas to the north and to the south. With these slopes being covered by settlements, stormwater run off into the outcrop zone in the centre of the valley is likely to be much higher than natural runoff as large portions in urban areas are impervious (roofs, paving), allowing most of the rainfall to be collected in the stormwater drainage systems instead of infiltrating into the soil. The former, in turn, may discharge much of the run off into nearby rivers that run parallel (i.e. E-W) with the valley resulting in increased stream losses and recharge to the underlying mine void.

Furthermore, the high infiltration capacity of the mined outcrop zone is also of concern for a number of man made major ingress sources which are located right in this zone, including slimes dams. Apart from continuously releasing highly polluted seepage, many of the SD are being reclaimed using hydraulic mining via high-pressure water canons which introduces large volumes of water into this high-infiltration area. Owing to the unconsolidated nature of the fill material in the outcrop zone in combination with mining-induced subsidence (collapse of old, near-surface stoping areas) and mining-induced tremors, sheer stress on underground reticulation systems and resulting leakage may be increased in the mine outcrop zone. Affected systems include pressured water pipes as well as non-pressured stormwater and sewage canals (many of which run preferentially along the valley bottoms relying on gravitational flow) which could well result in water losses exceeding that of other parts of the city. For metered pressurised pipes, that could perhaps be followed up with Rand Water and /or the City of JHB. It would also be helpful in terms of ingress prevention to have a map indicating the discharge points and receiving water bodies of the municipal stormwater drainage systems located in the outcrop zone.

The shape of the void is envisaged as being largely constant over its vertical profile as well as its lateral extent along the strike of the outcropping reef. Compared to the depths of 2000 m to >3400 m below surface and a length of over 44,000 m the average cumulative width of the void (some 4 m) is very small. This results in a void geometry that resembles a sheet-like shape inserted into the bedrock at an angle of 45°, dipping south, following the dip of the mined gold reefs. Generally, this would result in a more or less constant void volume to be filled by ingressing water. In the (hypothetical) absence of any other factor governing the flooding of the void and constant ingress volumes this would result mine water levels rising at a constant rate that could be used to predict the surfacing of mine water through linear extrapolation. However, the observed rates of rise in the different sub-voids are not always constant owing to a range of factors discussed below. For a 'no-intervention-scenario' where the mine water level is allowed to reach topographic decant level (i.e. is not artificially kept lower through pumping), slower rates of rise could be expected as the mine water enters the refilled surface mining area (some 50 m to 20 m below surface). With the width of this zone being between 20 to 30 times larger than that of the underlying (deep) void the rate of rise of the mine water level is expected to slow down considerably. This slow down would further be promoted by the increase in pore-space of the unconsolidated fill material compared to the 'fractured' mine aquifer and the fact that increasingly more ingress sources may be cut off by the rising mine water table reducing or even inversing hydraulic gradients that previously allowed for ingress.

9.6.4 Historical mine water levels in the CR

The best indirect indicator for the hydraulic interconnectivity of the various sub-voids is historically observed water levels which are depicted in Fig. 9.17 for the past 27 years.

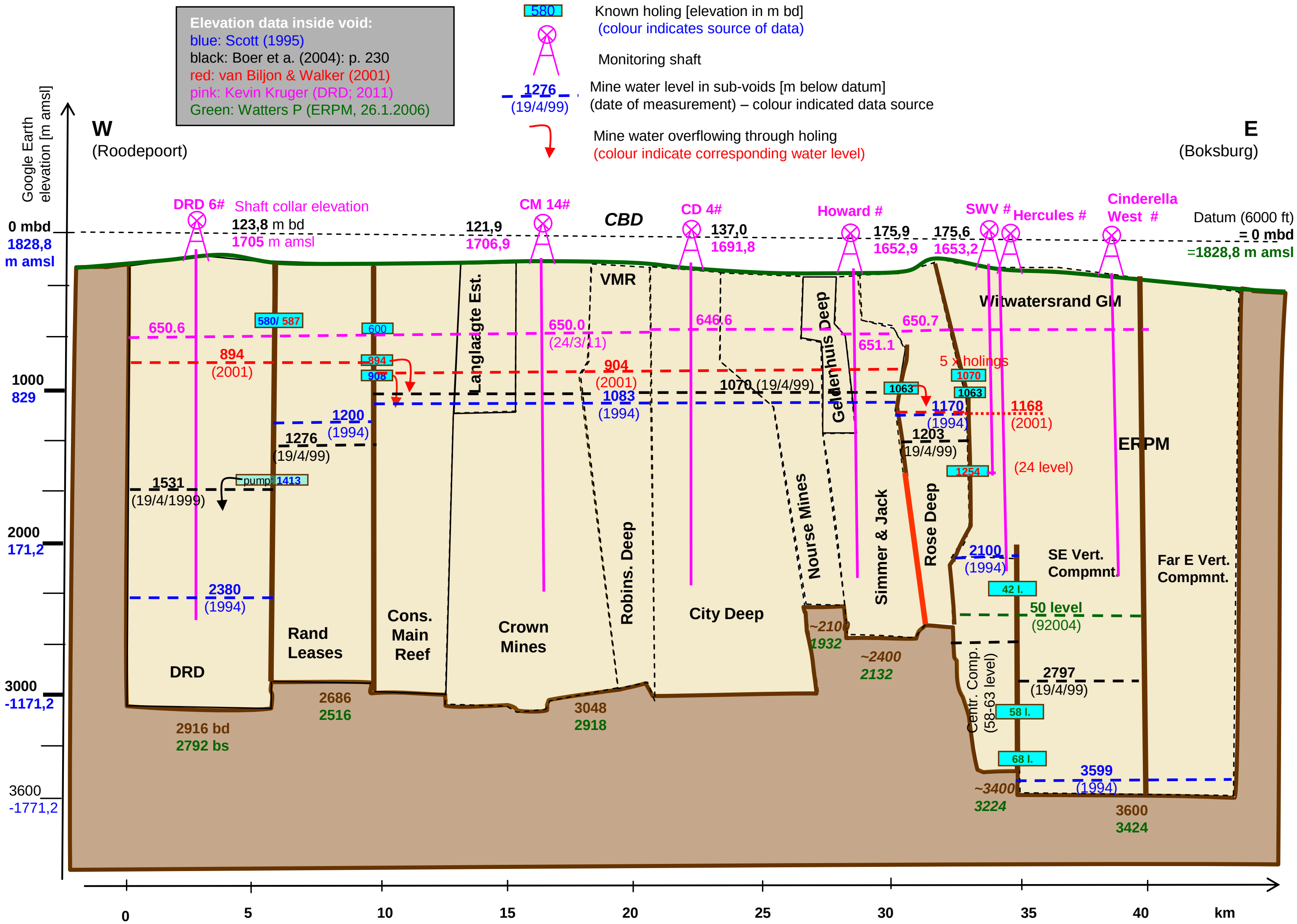

Fig. 9.18**:** Hydraulic links between sub-voids of the Central Rand and mine water levels for selected dates between 1994 and 2011 (based on various sources indicated in the figure)

Fig. 9.18 illustrates that the various sub-voids in the Central Rand are connected at different levels via so-called ‘holings’ (assumed to be a general terms for any conduit connecting two mines including boreholes, haulages, drives, etc.). However, the number of links indicated and their respective elevations differ between the different sources and therefore display some degree of uncertainty.

For determining whether or not a single water level elevation will be established across the different sub-voids, it is important to determine possible hydraulic links connecting the three major sub-basins i.e. DRD-Rand Lease in the west, the Central sub-basin (stretching from Consol. Main Reef to Rose Deep) and the ERPM sub-basin at the eastern border of the CR. Fig. 9.17 suggests that this can indeed the case. The DRD/RL sub-basin can be connected to the Central sub-basin through 3 x holings at various levels (984, 906 and 600 mbd) of which the lowest one (984 mbd) must have been reached sometime between April 1999 (when the mine water table at both sides of the holings according to Boer et al., 2004 was below this level) and 2001 when according to Van Biljon and Walker (2001) the water table in both sub-voids was above this holing (Fig. 9.17). The fact that the WL in the DRD sub-basin rose even higher to the level of the second holing (at 894 mbd) indicate that the lower holing could not accommodate all inflowing water entering the DRD sub-void.

A significant hydraulic head between Rose Deep and the remainder of the Central sub-void indicates that the transmissivity of the boundary pillar in-between is not sufficient to accommodate the total ingress received by the Central sub-void (some 9-14 Ml/d). According to Scott (1995) this is due to the collapse of haulages located in an intrusive and faulted boundary pillar. While, in 1994, the hydraulic head from the Central sub-void to Rose Deep was 87 m (based on Scott, 1995) this increased to 133 m in 1999 (Boer et al., 2004) and to 272 m in 2001 (904 vs. 1168 mbd, van Biljon and Walker, 2001). This increase occurred in spite of a holing at 1063 mbd (Boer et al., 2004), 157 m below the 2001 water table in the remainder of the central sub-void (Fig. 9.18).

In Tab. 9.11 the latest water level data for 5 x shafts monitored monthly by DRD are compiled.

The WL at the 5 x different shafts are all measured at the same day within 5 hours using a non-stretch metered steel cable with electric wire that closes circuit at contact with the mine water triggering a whistle. The steel cable is wound on a winch mounted on the back of a bakkie (Labuschagne, 2011).

Tab. 9.11: Elevation of mine water tables [**m below datum** and m below collar] in different sub-voids as measured at 5 x monitoring shafts between 23 July 2009 and 14. March 2011 (data source: Kruger, 2010/2011). Red: rate of rise calculated between 2 successive measurements; Blue: deviation of water tables from level measured at SWV shaft [m]; first column: nr. of days between two successive measurements

			Measuring shafts (from W to E)					
		shaft name	**DRD 6#**	**GRC 14#**	**CD 4#**	**Howard #**	**SW Vert #**	
		sub-void	DRD-RL	Crown Mines	City Deep	Simmer&Jack	Rose Deep	CM-RD
		shaft collar elevation [m amsl]	*1705,0*	*1706,9*	*1691,8*	*1652,93*	*1653,2*	**av.** rate of ise
date of measurement				121,9				
	23.7.09	**[m below datum]**	**778,9**	**928,6**	**936,3**	**938,7**	**938,7**	
46		[m below collar /surface]	655,1	806,7	799,3	762,8	763,1	
		Diff. to SWV shaft [m]	159,8	10,1	2,4	0,0	0	
	7.9.09	**[m below datum]**	**779,0**	**920,6**	**926,2**	**927,9**	**927,9**	
28		[m below collar /surface]	655,2	798,7	789,2	752,0	752,3	
		Diff. to SWV shaft [m]	148,9	7,3	1,7	0,0	0	
		Rate of rise [m/d]	0,00	0,17	0,22	0,23	0,23	**0,22**
	5.10.09	**[m below datum]**	**779,0**	**911,0**	**912,8**	**914,7**	**914,4**	
17		[m below collar /surface]	655,2	789,1	775,8	738,8	738,8	
		Diff. to SWV shaft [m]	135,4	3,4	1,6	-0,3	0	
			0,00	0,34	0,48	0,47	0,48	**0,44**
	22.10.09	**[m below datum]**	**779,0**	**905,1**	**904,6**	**906,5**	**906,2**	
26		[m below collar /surface]	655,2	783,2	767,6	730,7	730,6	
		Diff. to SWV shaft [m]	127,2	1,0	1,6	-0,4	0	
		Rate of rise [m/d]	0,00	0,34	0,48	0,48	0,48	**0,45**
	17.11.09	**[m below datum]**	**779,0**	**893,0**	**892,5**	**894,4**	**894,0**	
56		[m below collar /surface]	655,2	771,1	755,5	718,5	718,5	
		Diff. to SWV shaft [m]	115,0	1,0	1,5	-0,4	0	
		Rate of rise [m/d]	0,00	0,47	0,47	0,47	0,47	**0,47**
	12.1.10	**[m below datum]**	**779,0**	**838,6**	**838,0**	**839,9**	**839,6**	
20		[m below collar /surface]	655,2	716,6	701,0	664,1	664,0	
		Diff. to SWV shaft [m]	60,6	1,0	1,5	-0,4	0	
		Rate of rise [m/d]	0,00	0,97	0,97	0,97	0,97	**0,97**
	1.2.10	**[m below datum]**	**779,0**	**838,6**	**835,0**	**837,9**	**838,4**	
11		[m below collar /surface]	655,2	716,6	698,0	662,0	662,8	
		Diff. to SWV shaft [m]	59,4	-0,2	3,4	0,5	0	
		Rate of rise [m/d]	0,00	0,00	0,15	0,10	0,06	**0,08**
	12.2.10	**[m below datum]**	**779,0**	**838,6**	**835,0**	**831,1**	**830,9**	
17		[m below collar /surface]	655,2	716,6	698,0	655,2	655,3	
		Diff. to SWV shaft [m]	51,9	-7,7	-4,1	-0,2	0	
		Rate of rise [m/d]	0,00	0,00	0,00	0,62	0,68	**0,32**
	1.3.10	**[m below datum]**	**779,0**	**838,6**	**824,0**	**824,4**	**824,1**	
49		[m below collar /surface]	655,2	716,6	687,0	648,5	648,5	
		Diff. to SWV shaft [m]	45,1	-14,5	0,1	-0,3	0	
		Rate of rise [m/d]	0,00	0,00	0,65	0,39	0,40	**0,36**
	19.4.10	**[m below datum]**	**779,0**	**791,9**	**788,0**	**789,0**	**788,6**	
17		[m below collar /surface]	655,2	670,0	651,0	613,1	613,0	
		Diff. to SWV shaft [m]	9,6	-3,4	0,6	-0,4	0	
		Rate of rise [m/d]	0,00	0,95	0,73	0,72	0,72	**0,78**
	6.5.10	**[m below datum]**	**779,0**	**780,0**	**777,4**	**778,4**	**778,0**	
57		[m below collar /surface]	655,2	658,1	640,4	602,5	602,4	
		Diff. to SWV shaft [m]	-1,0	-2,1	0,6	-0,4	0	
		Rate of rise [m/d]	0,00	0,70	0,62	0,62	0,62	**0,64**
	2.7.10	**[m below datum]**	**747,0**	**748,0**	**745,4**	**746,4**	**746,0**	
40		[m below collar /surface]	623,2	626,1	608,4	570,5	570,4	
		Diff. to SWV shaft [m]	-1,0	-2,1	0,6	-0,4	0	
		Rate of rise [m/d]	0,56	0,56	0,56	0,56	0,56	**0,56**
	11.8.10	**[m below datum]**	**734,6**	**735,6**	**733,0**	**734,0**	**733,6**	
42		[m below collar /surface]	610,8	613,7	596,0	558,1	558,0	
		Diff. to SWV shaft [m]	-1,0	-2,1	0,6	-0,4	0	
		Rate of rise [m/d]	0,31	0,31	0,31	0,31	0,31	**0,31**
	22.9.10	**[m below datum]**	**719,8**	**719,8**	**718,0**	**721,0**	**720,6**	
20		[m below collar /surface]	596,0	597,9	581,0	545,1	545,0	
		Diff. to SWV shaft [m]	0,8	0,7	2,6	-0,4	0	
		Rate of rise [m/d]	0,35	0,38	0,36	0,31	0,31	**0,34**
	12.10.10	**[m below datum]**	**714,1**	**714,1**	**714,6**	**715,3**	**714,9**	
36		[m below collar /surface]	590,3	592,2	577,6	539,4	539,3	
		Diff. to SWV shaft [m]	0,8	0,7	0,3	-0,4	0	
		Rate of rise [m/d]	0,28	0,29	0,17	0,29	0,28	**0,26**
	17.11.10	**[m below datum]**	**698,8**	**701,8**	**703,0**	**703,0**	**702,6**	
34		[m below collar /surface]	575,0	579,9	566,0	527,1	527,0	
		Diff. to SWV shaft [m]	3,8	0,7	-0,4	-0,4	0	
		Rate of rise [m/d]	0,43	0,34	0,32	0,34	0,34	**0,35**
	21.12.10	**[m below datum]**	**691,4**	**691,1**	**688,2**	**692,3**	**691,9**	
22		[m below collar /surface]	567,6	569,2	551,2	516,4	516,3	
		Diff. to SWV shaft [m]	0,5	0,7	3,7	-0,4	0	
		Rate of rise [m/d]	0,22	0,31	0,44	0,31	0,31	**0,32**
	12.1.11	**[m below datum]**	**681,5**	**680,8**	**680,1**	**682,0**	**681,6**	
37		[m below collar /surface]	557,7	558,9	543,1	506,1	506,0	
		Diff. to SWV shaft [m]	0,1	0,7	1,5	-0,4	0	
		Rate of rise [m/d]	0,45	0,47	0,37	0,47	0,47	**0,44**
	18.2.11	**[m below datum]**	**663,9**	**663,2**	**660,2**	**664,4**	**664,0**	
34		[m below collar /surface]	540,1	541,3	523,2	488,5	488,4	
		Diff. to SWV shaft [m]	0,1	0,7	3,8	-0,4	0	
		Rate of rise [m/d]	0,48	0,48	0,54	0,48	0,48	**0,49**
	24.3.11	**[m below datum]**	**650,6**	**650,0**	**646,6**	**650,7**	**651,1**	
		[m below collar /surface]	526,8	528,0	509,6	474,8	475,5	
days		Diff. to SWV shaft [m]	0,5	1,1	4,5	0,4	0	
		Rate of rise [m/d]	0,39	0,39	0,40	0,40	0,38	**0,39**
		WL rise since 2.7.10 [m]	96,4	98,1	98,8	95,7	94,9	
		days since 2.7.2010	265	265	265	265	265	
		Av. rate (since 2.7.10) [m/d]	**0,36**	**0,37**	**0,37**	**0,36**	**0,36**	**0,37**
		days toreach surface at av. rate	1448	1427	1367	1315	1328	
		date of reaching surface	**11.3.15**	**17.2.15**	**19.12.14**	**29.10.14**	**11.11.14**	

The latest measurements (March 2011) listed in Tab. 9.11 indicate that the water table in Rose Deep (measured at SW Vertical shaft) is no longer below the central sub-void but even slightly (0.4 m) higher than in the adjacent void to the west (Simmer & Jack as measured at Howard shaft, right next to Rose Deep: 650.7 vs. 651.1 mbd). This suggests that the Rose Deep sub-void is now fully connected to the rest of the Central sub-basin.

The same is true for the DRD-RL sub-basin. This sub-basin displayed a constant water table until May 2010 as all water that ingressed into this sub-void was overflowing via the above mentioned holings into the central sub-basin (Fig. 9.19).

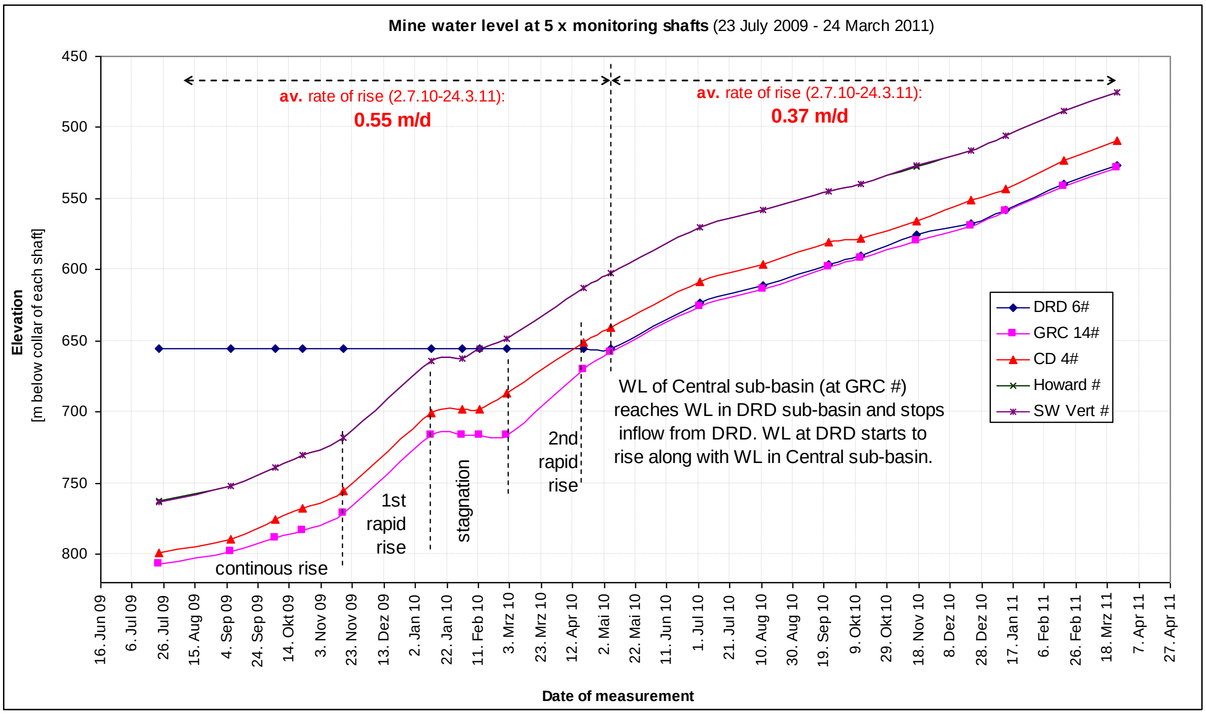

Fig. 9.19: Mine water levels in different sub-voids between July 2009 and February 2011 as measured at 5 x different monitoring shafts with associated average rates of rise before and after the WL at the Central sub-basin reached the WL in the DRD-RL sub-basin resulting in the formation of a single basin (based on data from DRD, Kruger, 2010/2011)

This stopped when the water table in the central sub-basin reached the water level at DRD. According to Tab. 9.11 this must have happened only 2-3 days after the 6th of May 2010 when the water level measured at CRG (representing the old Crown Mine sub-void) was only 1 m below the DRD level. At the next measurement (2 July 2010) water tables in both sub-basins were on more or less the same level. From then on the water table in the DRD-RL sub-basin started to rise at the same rate as at all other shafts indicating that both sub-basins have now been fully interconnected and reacts as a single basin (Fig. 9.19).

For the central sub-basin the joining with the DRD void resulted in a considerable slow down of the speed at which the water table recovers. The rate of rise in the Central sub-basin went down from an average of 55 cm/d over the 9 months before the basins joined to 37 cm/d in the 11 months thereafter (Fig. 9.18). This reduction is attributable to the fact that water ingressing into the DRD sub-void no longer runs into the central sub-basin and contributes there to the rise of the water table but now fills up void space in the DRD sub-basin itself which was not available for flooding as long as DRD water flowed into the central sub-basin. Hence the slow-down of the water table rise in the Central sub-basin is an indirect measure for the additional void volume that became available (regarding ingress as remaining constant). The observed reduction of the rate of rise from 55 to 37 cm/d (=18 cm/d) would represent an additional void volume of some 33%. With a total void volume of 79.3 Mm^3 (DRD: 53.6 + RL: 25.7 Mm^3) the proportion the DRD sub-basin occupies from the total volume in the Central Rand (467 Mm^3) is approximately 17%. That means that the reduction cannot only be attributed to the additional void volume that became available but to an additional factor. The only conceivable factor in this regard is a disproportionally small ingress volume at the DRD sub-basin compared to the central sub-void. As this results in the DRD sub-basin not being flooded as rapidly as the Central basin this would allow for water from the Central sub-basin to flow into the DRD-RL void space explaining the remaining 16% of the 33% drop in the flooding rate observed in the Central sub-basin. That means some 17% of the drop of the flooding rate in the Central sub-basin (i.e. about half of the 33% decrease) are explained by additional void space while the remaining 16% are attributable to a disproportionally small ingress volume at the DRD sub-basin.

This is somewhat unexpected as most previous studies assumed that much of the ingress into the Central Basin is generated at the high lying DRD and RL areas from where it will flow towards the low lying decant shaft in the east (e.g. Scott, 1995). It also implies that ingress is actually stronger in the central sub-basin. This is supported by the fact that since the basin joined the WL at CD 4# is consistently 1-5 m above the water level at DRD 6 # (with one exception).

Water table dynamics

Comparing the periods before and after the joining of the basins also shows the highly dynamic behaviour of the water table in the central sub-void which changed to a more steady rise after the basins joined.

Before the joining the water table in the Central sub-basin changed frequently between phases of steady increase, rapid rise and stagnation (Fig. 9.19). It is not clear what exactly caused these changing rates of water table recovery as pumping (as a possible factor controlling the water table) had already stopped in November 2010. While rapid increases may be caused by exceptionally high ingress, e.g. following intense rain events, stagnation periods are more difficult to explain as a minimum of ingress permanently enters the mine void causing some increase in the water level. Excluding possible monitoring errors the water table can thus only remain constant if the inflowing water is discharged into a lower lying empty void as observed at the DRD sub-basin before the WL in the Central sub-basin reached the same level.

This suggests that the stagnation period observed between mid January and March 2010 (i.e. during the wet season where ingress is relatively high, Fig. 9.18) can only be explained by the WL in the Central sub—void having reached a connecting link to such a void mid-January 2010. It appears that the water table stagnation is most pronounced in the west as the WL at the GRC shaft remained constant over 4 x consecutive measurements while at the second most westerly shaft (City Deep nr 4) stagnation lasted only for 3 x measurements being further reduced at Howard and SWV shafts in the east where WLs have been constant only for the first two measurements in this period. Thus, decant into a lower lying empty void must have occurred in the west of the Central sub-basin. As the WL in the DRD sub-basin also remained constant over this time (Fig. 9.18) it was perhaps some or other sub-void that had been hydraulically disconnected.

This however is highly speculative and it should perhaps also be explored why the period of stagnation is preceded and followed by phases of rapid rises which could indicate that changing measuring intervals or reporting errors may also be a possible explanation for the observed stagnation. The corresponding rates of rise for the different water tables in the monitored sub-voids are depicted in Fig. 9.20.

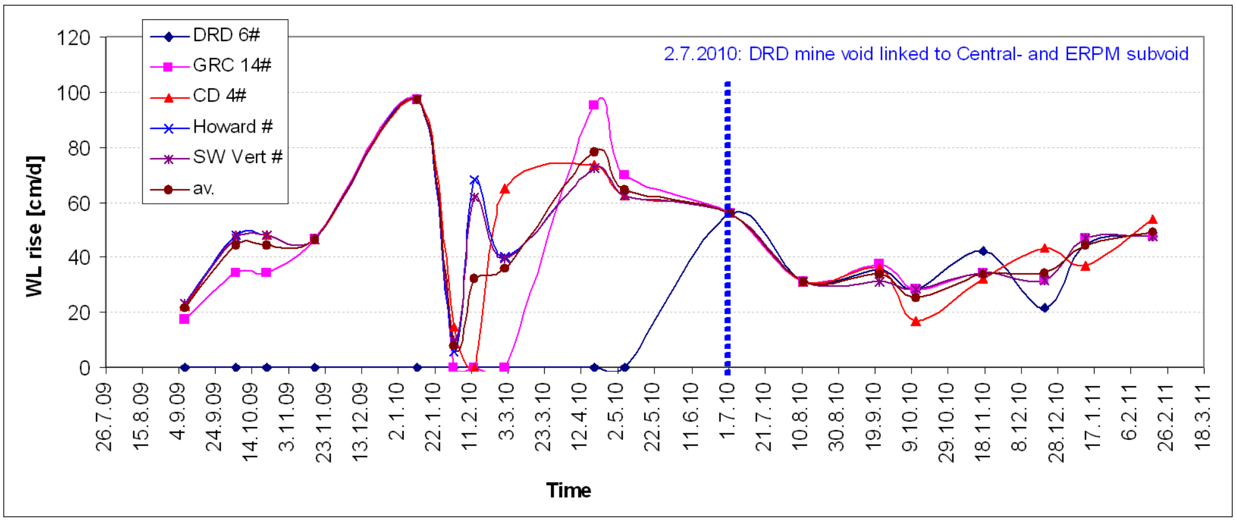

Fig. 9.20: Changes in the rate at which mine water tables in different sub-voids of the Central Rand rose between July 2009 and February 2011 based on water level measurements in 5 x monitoring shafts (based on data from DRD: Kruger, 2010/2011)

Fig. 9.20 illustrates that the amplitude of all rates clearly decreased after the DRD and central sub-basin were linked showing largely synchronised changes thereafter within a relatively narrow range around 40 cm per day.

A longer period of water level rise was available for Crown Mines no. 14 # which is now used by Gold Reef City as a monitoring shaft. The water levels were later also measured by DRD using the same shaft but their own instrument. The 2 different data sets are shown in Fig. 9.21.

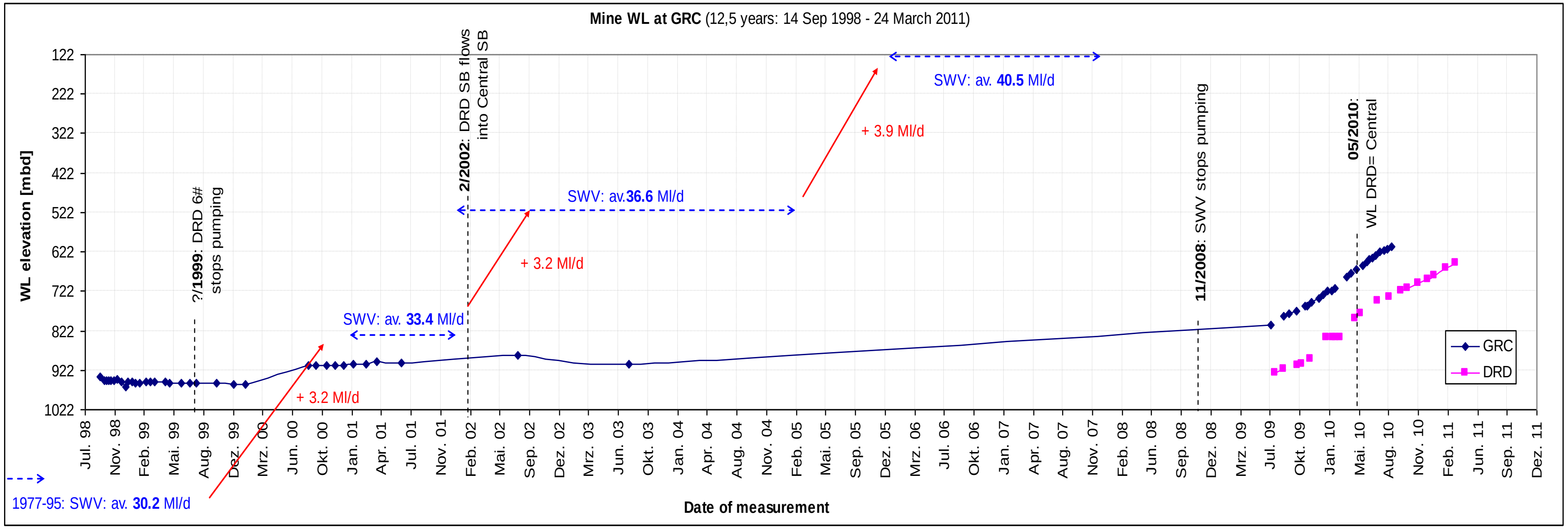

Fig. 9.21: Mine water level at Crown Mines 14 # (central sub-basin) between July 1998 and March 2011 measured by Gold Reef City (GRC) and DRD as well as corresponding pumping rates at the SWV shaft

Unfortunately large gaps exist in both data sets leaving an almost 6-year long period (September 2003 to June 2009) uncovered by data. While the straight line between the last measurement of GRC in 2003 and the first one in 2009 suggests a gradual increase this is probably not correct as pumping at SWV shaft only stopped in Nov. 2008. That means that the WL before this date has been kept more or less constant with consistent increases only commencing thereafter. Since both monitoring programmes only have data starting mid-2009 the first 6 months of rise after pumping stopped at SWV # are not covered.

Based on the GRC data the impact of the pumping cessation at DRD 6 # can be seen by a rise of the water table a couple of months later after water in the DRD sub-basin reached the level of the decant holing and started to overflow. However, continued pumping at SWV # kept the WL in the receiving Central sub-void constant until pumping stopped for good in late 2008. The increase in pumping necessary to keep the water level constant is shown in Fig. 9.21.

It appears that the rate of rise of the WL (gradient of a straight line connecting all data points) indicated by the two data sets is more or less identical. Different however is the starting point of the first measurement after pumping stopped which for CRG data is set more than 100 m above the starting point of the DRD data. This, in turn, has implications for predicting the date at which the WL will reach the surface that based on CRG data will be earlier than based on DRD measurements.

Based on the average rate that prevailed in the 9 x months after the WL in DRD-RL sub-basin was reached in May-July 2010 (2.7.2010-23-3-2011: 37 cm/d; n=9) the earliest date at which the mine water table could reach the surface at the lowest lying monitoring shaft (Howard shaft at Simmer & Jack at 1652.93 mamsl) would be October 2014, i.e. in 3.5 years from date of writing (April 2011) (Tab. 9.11). This is, however, only a hypothetical data as the mine water level will reach the surface earlier at the Cinderella West shaft at ERPM as lowest still open shaft fully connected to the Central sub-void.

The most recent measurement (24 March 2011) indicates that the water table in the DRD sub-void is currently nearly exactly on the same level as in Rose Deep located some 33 km further east (650.6 vs. 650,7 mbd respectively). This confirms that water moves (nearly) freely across all mine boundaries from DRD in the west to Rose Deep in the east.

While it is generally accepted that the lateral intra-void mine water flow is predominantly from West to East (i.e. from the higher lying grounds of DRD to the lower lying areas at ERPM) this currently does not always seem to be the case as the water level at City Deep (4 shaft) is 4.0 m higher than at DRD and 4.5 m higher than at Simmer &Jack (Howard shaft). An analysis of water table fluctuations between the 5 x monitoring shafts between 7/2009 and 2/2011 indicates that heads of up to 15 m temporarily formed between adjacent sub-voids. Such heads can only form in a sub-void if the hydraulic links to the adjacent voids are not large enough to instantly accommodate all ingress received by this void. This, in turn, can either be caused by a reduction of the hydraulic connectivity (e.g. through collapse of flooded haulages or continued bulk expansion) or by increasing ingress volumes in this particular sub-void.

The hydraulic head at City Deep no. 4 # which persists for the past 3 months (December 2010 - March 2011) is possibly indicates ingress volumes at this area which are above those received by the other sub-voids as well as a limited hydraulic connectivity.

The historic water levels in the different sub-voids (Fig. 9.18) also indicate that data from different sources are not always consistent with each other. In some instances more recent water levels are reported to be below older levels. This, however, could only be possible if existing hydraulic barriers between sub-voids with different water levels were subsequently pierced allowing for mine water to drain from the void with the higher water table into the less flooded void. While such instances may have occurred, triggered for example by failing pressure plugs, such events have never been explicitly mentioned by any of the consulted reports and competent persons. An example of a dropping water table is the Rose Deep sub-void where the WT in 1994 was 33 m higher than in April 1999. This also applies to the Rand Lease sub-void where the WT in 1994 was 76 m higher than in April 1999 (Fig. 9.18).

9.7 Prediction of the final water level in the flooded mine void

Based on the above discussed factors controlling the FMWL sufficient evidence could be gathered that the decant level will be controlled by the lowest lying open shaft that is hydraulically connected to Central Basin. Currently this is the Cinderella West (Ventilation shaft) located at ERPM in the eastern part of the Central Basin displaying a collar elevation of just below **1614 mamsl** (Fig. 9.22).

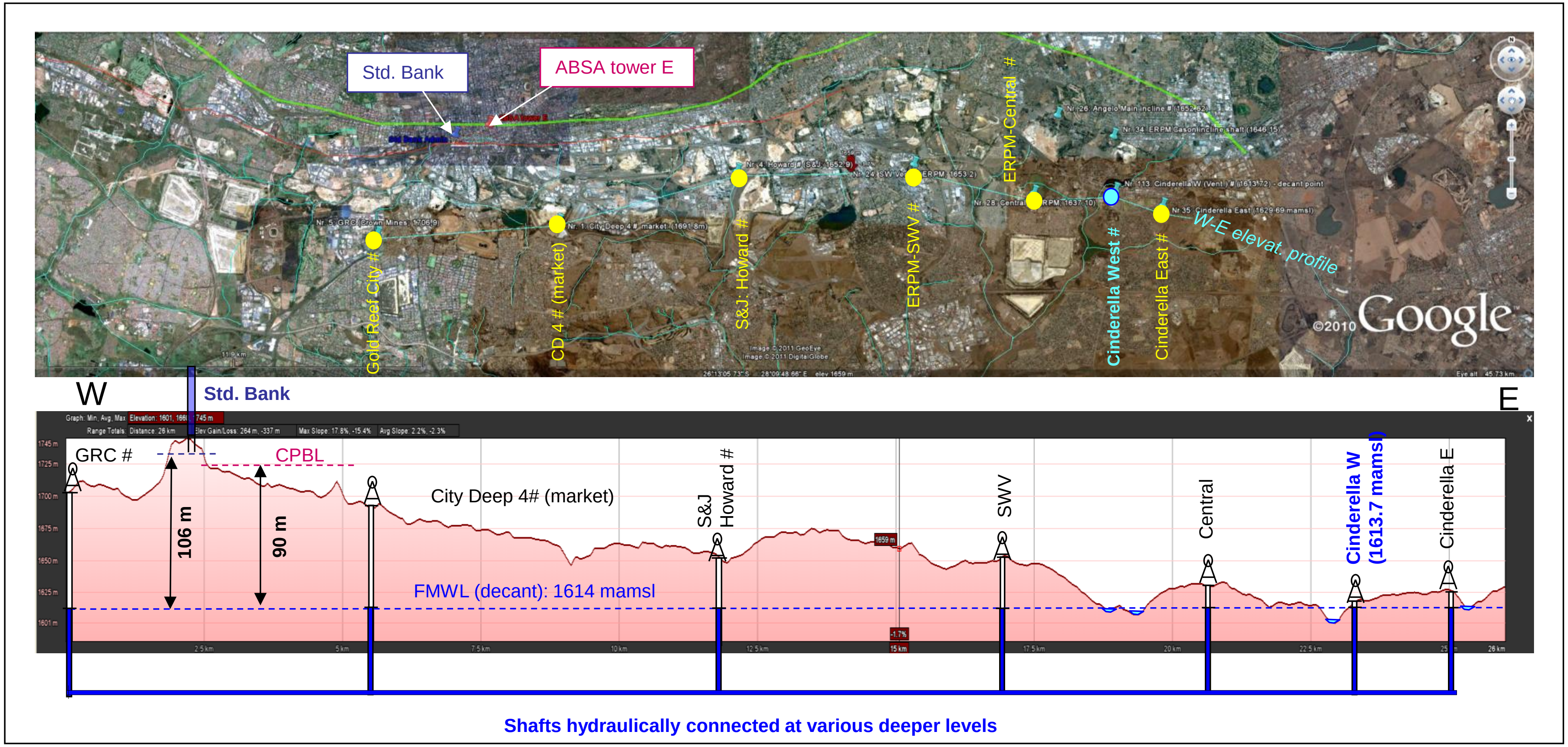

Fig. 9.22: Location and elevation of 7 x selected open shafts (3 x of which are monitoring shafts) in relation to the pile base levels of Standard Bank and ABSA tower East indicating a safety margin of 90 m and 106m respectively to the expected decant level at the lowest lying shafts (Cinderella West # at ERPM: 1613.7 mamsl).

Since the PBL of Standard Bank is 106 m above the mine water table, and the PBL of the ABSA tower E some 90 m, the flooding of basement structures of the two bank buildings (and all other related bank buildings indicated by ABSA) is most improbable as long as the decant shaft is kept open. Being large enough in diameter to accommodate all reasonably expected decant volumes, this shaft would be able to keep the FMWT at a constant level and thereby prevent any flooding risk for any building in the CBD even if no intervention of any kind to artificially control the water table is implemented.

However, located at a busy area near a major shopping centre and highways, it may not be feasible to use this shaft as a preferred decant point where water is collected and centrally treated before being released into nearby streams. Thus it cannot be excluded that in a 'do-nothing-scenario' (i.e. no intervention other than closing this shaft) the next lowest lying open shaft will be used as central decant and treatment point. As illustrated in Fig. 21 a number of other still open shafts exists which are well below the CPBL excluding a risk for basement flooding in the CBD. In addition to this, all unlined shafts (filled, capped or otherwise closed) would most probably act as safety valves as these shafts have not been lined allowing mine water to escape through cracks, fissures and fractures connecting the shafts to low lying surfaces areas. The same is true for faults and dykes along which stream water currently enters the mine void. Once the mine water table rises above these ingress points, (uncontrolled) discharge of mine water into these streams would prevent the water reaching the CPBL.

Based on latest monitoring data displaying similar elevations of the mine water table across the entire Central basin it can be assumed that the hydraulic interconnectivity of the different shafts and sub-voids is large enough to allow all ingressing water to laterally flow towards the lowest lying outflow point (decant shaft). Observed differences between adjacent monitoring shafts do not exceed 5 m and are believed to be caused by concentrated inflow of surface water into the City Deep sub-void. As a sustained head of approximately 4m was observed for this sub-void over the past 4 months (since the onset of the rainy season in December 2010) it may relate to above average stormwater run off preferentially entering this part of the Central Basin. The fact that the head is constant over the past couple of months also suggests that an hydraulic equilibrium is reached between ingress from surface and the distribution to neighbouring sub-voids through (obviously somewhat limited) hydraulic connections such as collapsed haulages.

With no intervention, mine water is expected to reach the collar elevation of Cinderella West shaft in 839 days. The last mine water measurement on 24 March 2011 indicated the mine water level being at 646.6 mbd (measured at CD 4 #) giving the date of daylighting in **mid September 2013.** This prediction is based on the average rate of rise of the mine water table of 37 cm per day observed since July 2010 when a uniform water table was established across the entire Central Basin. As this average rate was calculated over a period that was marked by exceptionally high rainfall (Dec. 2010 to March 2011) it is unlikely that the predicted recovery to surface be shortened, as the probability that even wetter periods would occur before the decant date is rather small. In fact, the opposite is more likely i.e. a slow-down in rise as the inclusion of a completely dry season (thus not covered by the calculated average) probably would result in a somewhat reduced average rate of rise.

Once the mine water table recovers to the surface at Cinderella W shaft, much of the decant will most probably occur at this shaft. However, it cannot be excluded that some mine water may also seep diffusely and uncontrollable into the environment where flooded parts of the mine void are above low-lying adjacent terrain. Such diffuse type of outflow is most likely to happen where transmissive features connect the flooded void with the receiving environment. In this regard the following areas are identified as being potentially at risk:

(1) Areas around old shafts along the Main Reef outcrop zone which are flooded to a level that allows the mine water to seep directly through the unlined walls of the shaft into the disturbed 40 m-deep zone where the reef was mined from surface.

(2) Identified ingress areas where streams crossings of faults, dykes and reef outcrop zones lying below the decant level

(3) Valleys and depressions in the natural relief located below the decant level which are connected to the flooded mine void via faults and dykes.

Fig. 9.23 suggests that areas where seepage from old unlined shaft could directly affect the highly disturbed outcrop zone of the MR are confined to 2 shafts in the low-lying eastern part of the MR. For this area it should be explored whether water saturation of possibly existing unconsolidated material could pose a potential geotechnical risk to structures such as slimes dams or buildings.

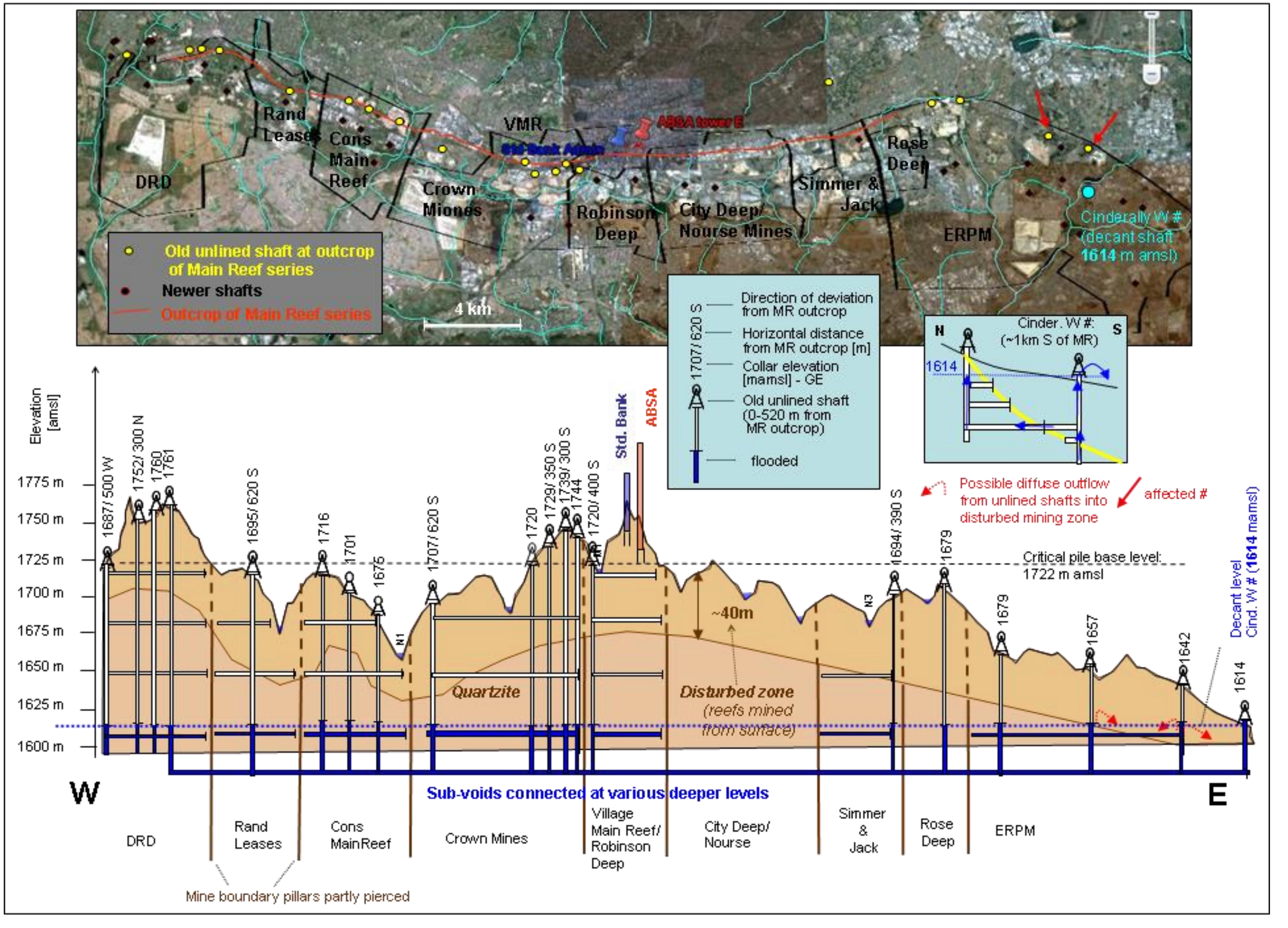

Fig. 9.23: Location and elevation of old unlined shafts at the MR outcrop along an E-W profile of the Central Rand in relation to the decant level at Cinderella West shaft at 1614 mamsl. Red arrows indicate possible seepage of mine water migrating from old unlined shafts into the disturbed outcrop zone where liquefying of unconsolidated material like sand could cause geotechnical stability problems.

Stream crossings of dykes, faults and outcrop zones which are below the decant level and thus possibly affected by diffuse mine water seepage are shown in Fig. 9.24.

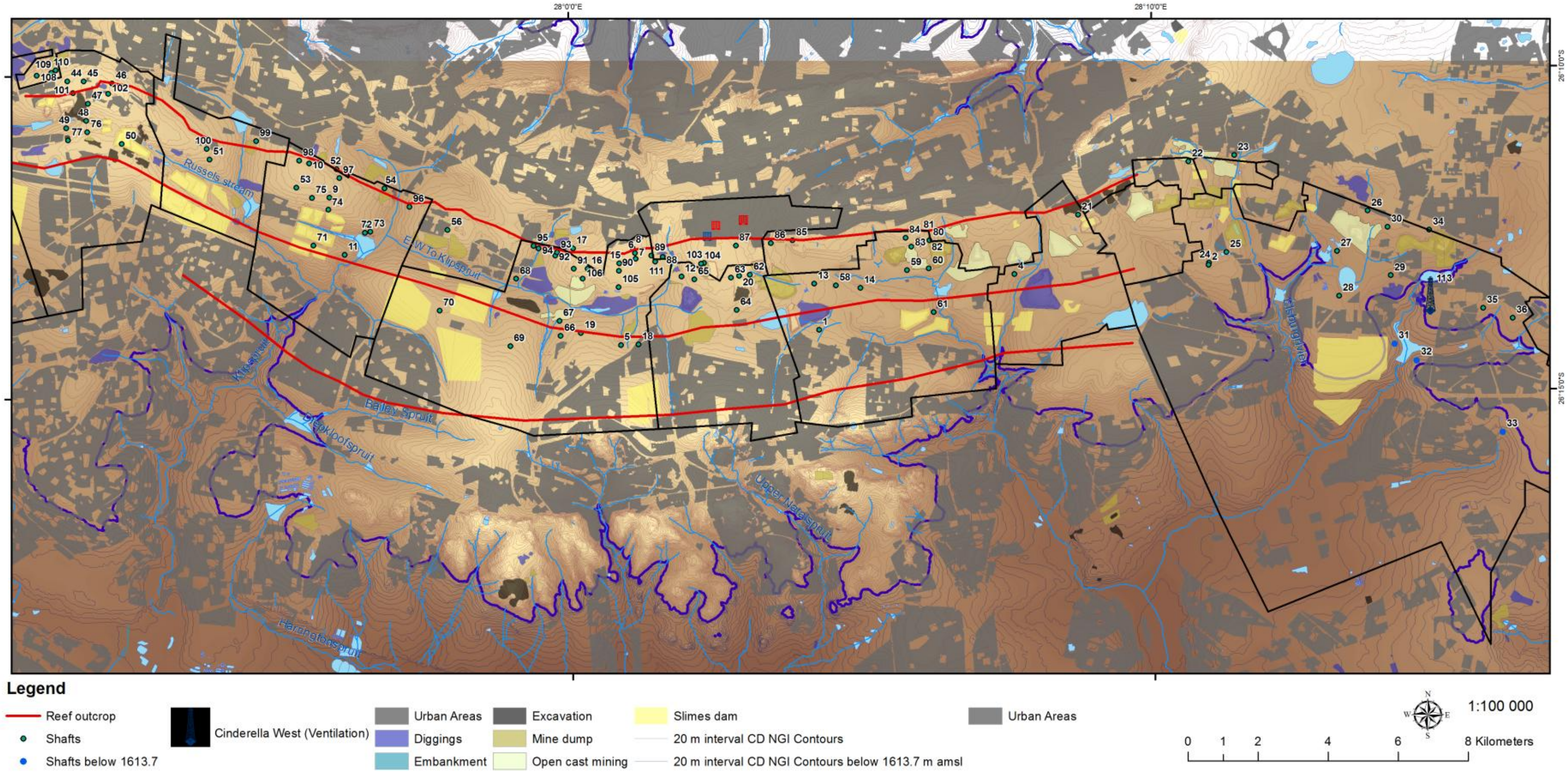

Fig. 9.24: Location and elevation of old unlined shafts at the MR outcrop along an E-W profile of the Central Rand in relation the piezometric surface and the critical pile base level. Scenario: no outflow through open shafts – old shafts serve as decant points

Fig. 9.24 indicates that of the known ingress points associated with streams crossing transmissive geological conduits 5 shafts lie below the decant level (2 shafts are associated with dykes and 3 shafts with faults).

Areas of the natural relief lying below the decant level are depicted in Fig. 9.25.

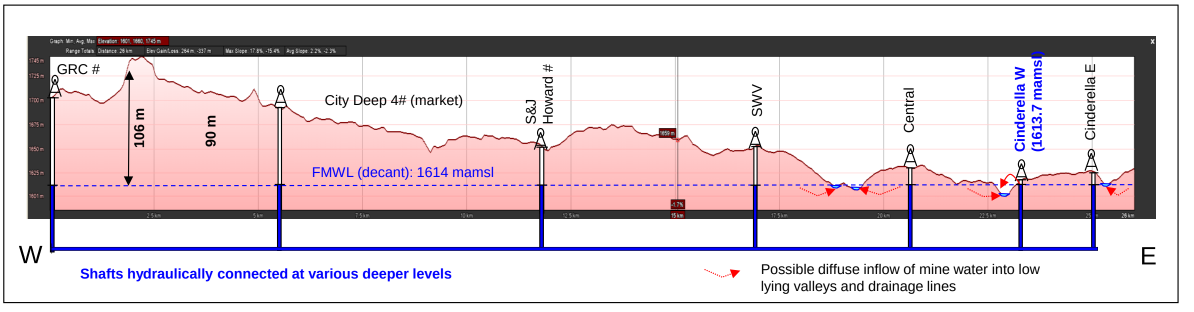

Fig. 9.25: E-W cross section along the monitoring shafts indicating valleys potentially affected by mine water seepage from the flooded void based on a decant level of 1614 mamsl.

Fig. 9.25 illustrates where low lying areas especially stream valley and drainage lines are potentially affected by diffuse mine water seepage from the flooded void.

10 Assessing risks associated with the mine void flooding

10.1 Basement flooding

Following from the findings discussed above no risk of basement flooding exists for any building in the CBD even if no intervention measures are taken.

Provided that the lowest lying shaft that has been identified as the decant point (ERPM's Cinderella West Ventilation shaft at 1614 mamsl) will be kept open, the final water level in the mine void will remain 90 m to over 100 m below the lowest basement structures of the identified 3 x key bank buildings. That leaves a considerable safety margin which can fully accommodate all possible data uncertainties.

In the case that this shaft could for some reason not serve as a decant point, 8 x other open shafts exists that could take over this function, all located well below the CPBL. In addition to this it is likely that a number of old shafts in the MR outcrop, even those which have closed, would serve as underground outflow points as water could diffusely seep through the unlined walls into low lying areas. Such diffuse seepage is also likely to happen at a range streams which previously acted as natural ingress points. Thus a range of additional 'safety valves' exist that render a rise of the mine water table close to the critical PBL most unlikely.

The most frequently cited key uncertainties relevant to the flooding risk relate to the following aspects:

(a) Accuracy of elevation data;

(b) Hydraulic connectivity between sub-voids in Central Basin,

(c) Hydraulic conductivity of potential decant shafts to the Central Basin;

(d) The extent of topography-related hydraulic heads;

(e) The volume of mine water decanting from the flooded mine void,

(f) The corrosiveness of the decanting mine water.

All of the above aspects have been investigated in the report, and where possible, were quantified. It was found that none of these uncertainties has the potential of rendering the above stated risk assessment invalid. Where assumptions had to be made, worst-case scenarios were always assumed in order to rather over- than underestimate the possible risks. The main findings regarding the key uncertainties are briefly summarised

(a) In build-up areas such as the CBD radar-based Google Earth data tends to overestimate elevations by up to 15 m compared to Lidar and CDNGI data. In open terrain, however, GE data were found to show the highest accuracy of the 3 data sets measured against high-confidence surveyed elevations. By using the lowest of all elevation data for determining the critical PBL, and Google Earth data for terrain profiles the associated uncertainty was built in as a safety margin. Thus the maximum deviation found does not need to be considered again when the decant level is compared against the CPBL leaving the full 90 m height difference a safety margin against possible flooding.

(b) Monthly measurements of the mine water level at 5 x monitoring shafts distributed across the Central Basin show since July 2010 a nearly identical elevation of the mine water table indicating that all previous disconnected sub-basins are now fully interconnected forming one single mine void aquifer or basin. This is also confirmed by nearly identical rates of rise observed at all monitoring shafts.

(c) In order to act as potential decant point only shafts that are still open and connected to the Central Basin have been considered. Information regarding both aspects is based on statements of the chief surveyor of DRDGold, Mr. Vivian Labuschagne, who unselfishly and kindly supported this project by providing high-confidence data and pertinent information. Based on this it was established that all shafts identified as potential decant points meet the required conditions, i.e. are indeed still open and sufficiently hydraulically connected to the Central Basin.

(d) Since groundwater to some degree reflects the elevation differences of the overlying topographic surface, the formation of water table elevation differences (termed ‘hydraulic heads’) along the over 40 km-long CR with relief differences of up to 160 m, has been proposed ranging from 60 m (Scott, 1995) to 85 m (Metago, 1999a-c). These differences appear to be based on natural fractured or porous aquifers, rather than the interconnected mine void which appears to have generally a higher lateral transmissivity more resembling a karst aquifer. For the latter, hydraulic gradients of 1:1250 have been measured which would already result in a much reduced hydraulic head. Based on the latest measurement it appears that the maximal heads building up along the flooded basin do not exceed 15 m. After the DRD and Central sub-basins joined the highest water level difference was below 5 m. Thus the bearing of hydraulic heads on the risk of basement flooding in the CBD to occur is negligible.

(e) Although estimates for the possible decant volume cover a wide range from 24 to 100 Ml/d indicating a large degree of uncertainty, even the maximum value of 100 Ml/d could still be easily accommodated by an open shaft. However, with increasing decant volumes the probability of uncontrolled, diffuse mine water seepage into the environment would rise. Based on a conceptual model, the latest pumping figures and analogy studies from the Western Basin, it is proposed that the annual average decant volume will probably not exceed 40 Ml/d and more realistically be around 25-30 Ml/d.

(f) Since nearly all high-rise buildings in the CBD (and elsewhere in the world) are equipped to deal with groundwater inflow through draining systems and pumps the actual risks associated with the flooding by mine water is the proposed corrosiveness that could compromise the stability of the underground steel and concrete support. This ability of mine water, acidic or otherwise, has to our knowledge not been conclusively proven. Despite the absence of evidence it was assumed that any contact of mine water with basement structures regardless for how long it occurs constitutes a risk. However, a conceptual model on the gradual closure of deep stoping areas as well as observations on quality of mine water decanting in the only completely flooded mine void (the WB) suggest that mine water which remains underground, i.e. not exposed to atmospheric oxygen, may not be or turn acidic and as such may not be able to corrode massive concrete structures.

10.2 Other possibly associated risks

10.2.1 Flooding-induced ground subsidence

This risk relates to ground sagging that may compromise the geotechnical stability of structures build on affected ground. For areas within and adjacent to the mined outcrop zone of the MR which are undermined by old shallow mine workings this risk was identified long ago amongst others by Brink (1979) citing the collapse of old mine tunnels as a major risks.

While much of the mined outcrop will not be affected as the FMWL will not reach this zone some low lying areas in the eastern part (on ERPM property) may be affected. Whether, however, the flooding of shallow mine workings will indeed increase the risk of their collapse is not clear.

Of greater concerns, in our view, is the possibility that unconsolidated fill material found in the highly disturbed outcrop zone may be liquidized when flooded, resulting in stability risks for structures on surface such as SDs or buildings. Where old mine dumps and SDs have been used as support structures for roads (like for the M2 highway near the CBD where SDs form the base of major off-ramps), this possibly associated risk should be investigated. This in particular, as failure of slimes dams in the MR outcrop zone did already occur in the recent past as reported by Scott (1995).

For low lying valleys which are used as pathways for major N-S running highways like the N1 in the west or the N3 in the east of the CR, the stability of support pillars should perhaps be ascertained.

10.2.2 Flooding-induced seismicity

The possibility of rising mine water levels triggering increased seismicity in the CR has been raised by scientists from the CSIR and been widely reported in the newspapers. The recent AMD report to the IMC claims that a clear cause –effect relationship has been established, apparently showing an increased frequency of low magnitude seismic events for the CR. Based on the provided diagram in the report, this conclusion was not visually evident to us. As no statistical data were provided to support the proposed relationship, it remains still doubtful as to what extent seismic activity is indeed directly linked to mine flooding. This even more so as increased seismic activity was also observed since February 2010 in the Northern Cape around the Augrabies National Park, which are most certainly unrelated to any mining activity. In order to prove a statistically significant link between flooding and measured seismic activity, possible impacts of regional events such as the earth quake swarms currently observed in the N- Cape need to be taken into account.

While there is no doubt that deep level mining is the major (and perhaps only) cause of seismic activity in the generally tectonically stable Kaapvaal Craton in which the Witwatersrand basin is located, effects of flooding are less clear. Durrheim et al. (2005) stated that the rewatering of mine voids could reduce sheer stress at faults and thus ease stress-related adjustments in the form of low intense seismic events. Therefore, the magnitude of seismic events triggered by flooding is unlikely to exceed the magnitude of events that occurred during active mining, where maximum values of 4-5.1 on the Richter scale have been recorded.

In analogy to increased seismicity following the filling of large dams, the sheer water pressure in the flooded void was suggested to cause seismic activity. In this regard it needs to be pointed out that the shape of the mine void bears little resemblance to a dam as 90% of the water in the flooded void is stored in very thin 2-6m-thick sheets, which dips at an angle at some 45 degrees, distributing the water pressure very differently to a full dam. As pointed out in this report much of the deep mine void is closed due to plastic expansion of the highly stressed rock, reducing the total void volume by some 75% which also reduces the associated pressure on the surrounding rock.

Durrheim et al. (2005) also stated that seismicity would gradually decrease once the mine void is filled. As the mine void is approximately 85% filled without any major or abnormal seismic event, the likelihood for a disastrous seismic event to occur over the next few years seems to be small.

10.2.3 Radon (Rn) exposure

Samples of mine water from the CR taken during active deep level mining indicated elevated levels of U reaching up to 600 µg/l (Metago 1999a-c). As a product of radioactive decay, the radioactive gas radon is produced which subsequently escapes from the water. Where mine water recovers to levels close to surface, the escaping gas may reach the surface and accumulate in buildings or shacks. As a known and leading cause of lung cancer in uranium miners, radon accumulation in inhabited rooms poses a serious health risks. This risk is particular high where mine water levels are close to surface and covering material porous enough to allow the gas to migrate through capping layers in less than 3.8 days (the half-life of the gas after which 50% of the original gas decayed further). As the probability of mine water seepage is particular high in the disturbed reef outcrop zones that are directly overlying the deeper mine void and filled with unconsolidated material that offers little resistance for the radioactive gas to filter through, these zone are to be investigated regarding possible Rn exposure of residents. In this regard informal settlements are of particular concern as most shacks have no sealed surface allowing for Rn to accumulate in the often poorly ventilated structures which often even lack windows.

Since shafts are open long/lasting structures through which air can easily move and act as preferred pathways for equalizing barometric pressure differences (ventilation shafts were indeed designed for that purpose) and are directly connected to the uraniferous mine water, escaping radon can easily get to surface and contaminate the surrounding air. As many old shafts have been covered at a later stage by mining residues, such possible radon concentration may not always be indicated visibly by remaining shafts structures. Thus a detailed survey of the location and radon potential of old shafts should be conducted to identify possible exposure hot spots to take the necessary steps to prevent health risks.

10.2.4. Pollution of surface water

Where untreated mine water seeps directly into surface water bodies such as streams and dams this may result in a deterioration of the in-stream water quality. This impact is likely to be more pronounced in standing water bodies with long turnover times than in fluvial systems where no accumulation of seepage is possible. To a lesser extent this also applies to shallow aquifers which may be polluted by escaping mine water.

The actual extent of the associated pollution will depend on the water quality of the receiving system. For many of the small streams which are typically found in the CR this is already very poor as large percentages of their small catchments are covered with large volumes of mining residues such as sand and rock dumps as well as slimes dams which permanently (i.e. throughout the year) release highly contaminated seepage that finds its way into adjacent streams. Based on U concentration ratios, Winde and Sandham (2004) estimated for the upper Natalspruit near Alberton that acidic tailings seepage accounts for nearly a quarter of the total stream flow (24 %). This portion may increase considerably during the dry winter season when the seepage-contaminated baseflow is the major or even only component feeding the stream. In streams like that, which is the majority of all streams that originate above or within the mining belt, it is unlikely that escaping mine water will cause any significant further deterioration. This also applies to dams and other water bodies fed by these streams. However, the total waste loads will in this case increase.

As much of the ingress water arriving at the mine is derived from contaminated surface water, the quality of the decant may not be very different from the water quality of these streams. In fact, in some instance receiving streams may have a worse quality than the decanting mine water, which in contrast to certain streams also receive a considerable proportion of clean (unpolluted) water as part of the ingress. This may, however, be different during the initial phase of the decant when old mine water is discarded in which pollutants may have been concentrated underground, as was observed in the Western Basin where high initial levels later approached typical concentration found in tailings seepage. However, it should be explored to what extent the early practice of filling the upper few hundred metres of the mine void with ash from steam engines may lead to some form of buffering (i.e. raising the low pH) as the addition of fly ash from coal-fired power plants is an established way of neutralising AMD.

In conditions where the polluted and therefore denser mine water is kept below unpolluted and thus the lighter groundwater that skims over the flooded void, the escaping water from the void may be of a better quality than in the receiving stream. Measurements in flooded mine voids at different depths suggest that such stratification indeed occurs (Stoch L., personal comm. 29 April 2011). The significant reduction of the exposure of un-mined ore to oxidised water associated with the plastic closure of nearly three quarters of all stopes further reduces the potential for continued mine water contamination.

Lastly, streams like the Klipriver or the upper Elsburg and Natalspruit into which mine waste water used to be discharge during active mining may in the past have experienced much worse quality than currently or under the conditions of a (in volume limited) diffuse decant in future. In this context it should be ascertained in what way waste process water from the 3 x metallurgical plants used for tailings reclamation (Crown Mines, City Deep and Knights GM) is disposed of and if perhaps waste water is discharged into nearby streams.

While the possible impact of diffusely decanting mine water on the receiving surface water may be less dramatic than frequently proposed, it should not be used as an excuse not to rehabilitate the already polluted streams. In doing so many of the contaminating ingress sources such as slimes dams will have to be removed with positive consequences also for the long-term quality of the decanting water.

10.2.5 Pollution of groundwater

Owing to high concentrations of sulphate and metals including uranium mine water is generally of poor quality and thus able to contaminate unpolluted groundwater. Where this occurs and the affected groundwater is used for irrigation of crops, live-stock watering or direct human consumption the degree of contamination needs to be assessed specifically with regard to uranium (U) which is commonly elevated in mine waters from Witwatersrand goldfields. U concentrations measured in mine water samples may reach concentrations of several hundreds of µg/l compared to the global natural background for freshwater of about 0.4 µg/l. As U is chemo- and radiotoxic heavy metal precautionary measures must be taken to prevent undue direct and indirect exposure of residents using this water.

Groundwater in the CR is stored in a secondary (fractured) aquifer that developed in the weathered zone of the quartzites and shales that underlie JHB. This weathering zone extents to a depth of approximately 100 m below the topographic surface and stores most water in the upper 40 m. Below a depth of 100 m no water bearing fractures occur. With an average depth to the groundwater table of 15 m below surface in JHB the actual aquifer is on average some 25 m thick and located from 15 m to 40 m below the surface. The contamination of groundwater by escaping mine water can only occur where (a) the groundwater table is located below the mine water table *and* (b) pathways such as faults, dykes or fractures hydraulically link the mine void to the aquifer. The latter can assumed to be the case for the highly disturbed outcrops zone especially of the Main Reef package for which depths of up to 40m was reached via surface mining. Where mine water is able to enter this zone, pollution of the up- or downstream groundwater is likely to occur. For the identified decant level of 1614 mamsl, such diffuse groundwater contamination is in principle possible in the low lying eastern part of the MR outcrop at ERPM. Recharge of the aquifer by streams crossing geological features linking the surface water to the underlying groundwater (e.g. fractures, weathered dykes, fault lines, bedding planes etc.) is also possible. However, as many rivers and streams are already polluted as discussed above, such contamination will not be exclusively associated with the flooding of the mine void.

10.2.6 Dolomite-related risks

Firstly, and perhaps most importantly with respect to proposed risks for the CBD it needs to be pointed out, despite claims of a national newspaper to the contrary (Fig. 10.1), that no dolomite whatsoever occurs anywhere underneath central JHB or the CR goldfield for that matter.

Fig. 10.1: Sketch depicting risks associated with dolomite allegedly underlying central Johannesburg (Rapport, 15 August 2010)

However, as an area approximately 5 km south of the CB is underlain by dolomites, and the Media may have misunderstood and thus misreported valid concerns of activists and scientists, dolomites and ground movement related issues in the context of AMD is briefly addressed. As mentioned, in a number of news-media reports (print and television) the flooding of the mine void has been associated with the risk of causing ground instability in dolomite, allegedly through accelerated dissolution of the rock through acidic mine water. While the rock largely consists of calcium-magnesium-carbonate that can indeed be dissolved, even by weak acids such as carbon-dioxide containing rainwater (a process that needs to continue over thousands of years to yield measurable dissolution voids called karst cavities) a rapid dissolution of dolomite by AMD, to our knowledge, has not been proven in practice. In fact, mining experiments in the FWR to utilise chips from the abundantly available dolomite that overlies the mine void in this goldfield to neutralising AMD failed, as the chips were soon armoured by covers of precipitated iron hydroxides preventing any significant dissolution of the dolomite chips.

Secondly, the karst-related ground instability such as dolines or sinkholes in dolomitic areas are not caused by current dissolution and formation of voids but by percolating water removing unconsolidated near surface material (soil) to pre-existing cavities. This was most dramatically observed in the FWR where the dewatering of the dolomitic aquifers overlying the underground mine workings lowered the groundwater table by several hundreds of meters in places. In many areas that resulted in pre-existing groundwater filled karst cavities falling dry, allowing infiltrating surface water to percolate through the covering soil and weathered rock material removing particles along the way to the karst cavities (also termed receptacles). This sub-surface erosion is enhanced in situations where the water table is lowered below an underground cavity connected to the overlying soil which now tends to concentrate flow of diffusely infiltrating water, much like a French drain, into the void. This concentration is the actual cause of sinkholes, as it allows washing down of lose fill material into the underlying karst receptacles until only a small arch of compacted soil at surface remains concealing the void underground. The sudden collapse of these arches leads to the formation of a sinkhole which in many cases had catastrophic consequences including loss of life. Thus the presence of compacted soil layers, which in the FWR have been mainly associated with ferricrete horizons, is a pre-requisite for the dangerous suddenness (and thus danger) with which sinkholes appear. Without such layers that withstand sub-surface erosion for a certain period of time, most sinkholes would not be catastrophic as their gradual formation could be observed from surface timely indicating impending danger.

As no such lowering of the groundwater table is caused by the flooding of the mine void (in fact the opposite is likely to happen) this specific risk mechanism type does not exist. Possible indirect impacts of mining on the stream water levels and associated groundwater table changes in the downstream dolomitic aquifer are regarded as marginal compared to impacts associated with the large-scale abstraction of dolomitic groundwater by irrigation farming in the area.

10.2.7 Damage to corporate and city image as well as business confidence

While very different in nature, to the risks discussed in the sections above may be useful to explore to what extent the continued negative publicity and sensationalizing reports damage the image of JHB, not only as a metropolitan area of global importance but also as a place where major business transactions e.g. at the JSE are conducted.

Continuous reporting on an imagined imminent environmental catastrophe threatening to “wipe out the city” through “South Africa’s own Tchernobyl” may perhaps assist in getting public attention, but may also harm the confidence of the international community that JHB is a place for doing business. With the new administration building of Standard Bank being frequently singled out, such reporting may not only adversely affect the corporate image of the affected company, but also result in a perhaps more direct and tangible loss of property value.

The Minister of the National Planning Commission, Mr. T. Manuel, suggested that part of the media hype may be driven by private interests supposedly relating to the pumping and treatment of AMD for which a major project has been proposed to government.

In view of the complete absence of many of the much publicised risks pointed out in this report (e.g. flooding and corrosion of basements in the CBD, dolomitic ground instability, massive pollution of rivers etc.), the aforementioned assessment of the Hon. Minister may well be correct. As many of the risks were reiterated in the latest and most important AMD report to the IMC and Cabinet, the question arises why a more sober analyses of the situation was not conducted to date, especially as other risks, identified in this report have been ignored (radon exposure, geotechnical risks to highways and buildings in the disturbed mining zone etc.).

This especially, as far-reaching decisions had to be based on a document which in many crucial aspects, such as the decant volume, associated geotechnical, environmental and risks, decant date, ingress sources and pathways, the ingress dependency on rainfall etc. appear to have left significant room for improvement.

11 Summary and conclusions

Following a request by Standard Bank and later ABSA to Prof. Winde to investigate the validity of media reports on an impending catastrophic flooding of their underground building structures by rising mine water rising underneath JHB, the Mine Water Research Group of the North-West University headed by Prof. Winde, in August 2010, agreed to undertake a desk-top study to assess possible associated risks.

The main objective of this study was to gather existing data and information in order to determine - with a high degree of certainty – the level to which the mine water table in the currently flooding mine void will ultimately rise and to what extent this would pose a threat to basements of the identified key bank buildings in the CBD of JHB. The risk can thus be quantified by using the height difference between the elevation of the lowest lying underground structure of the key bank buildings and the highest possible elevation of the final mine water table in the completely flooded void. Therefore, the risk assessment consists in essence of the determination of these 2 x elevations.

While the bank elevation is a fixed value whose determination is straightforward requiring only accurate data, the final mine water table needed to be relatively accurately predicted using conceptual models that quantify the factors controlling it. The overwhelming majority of data presented in this report is concerned with understanding and quantifying the factors that govern the elevation to which the mine water will finally rise. This was deemed necessary, to reduce the degree of uncertainty associated with assessing the flooding risk as much as possible.

The methodology underlying this report consists of the following four steps:

Step 1: Assessing elevation data accuracy: As accurate elevation data are of crucial importance all elevation data available for the study area were compiled and compared with each other to assess possible deviations and assess the accuracy of the different data sets. The following 3 x main data sets have been used in the project:

- Contours at 5 m interval as used in the official 1:50,000 topographic map series as provided by the Chief Directorate for National Geospatial Information (CD-NGI) claiming a vertical accuracy of 2.5 m, (short: 5 m CDNGI);
- Lidar data at 0.25 cm and 1 m vertical resolution obtained from the JHB and Ekhuruleni municipal areas respectively, which is based on an airborne remote sensing method using laser technology, (short Lidar);
- 1-m interval elevation data retrieved from Google Earth satellite imagery which is based on improved SRTM data, a satellite-based remote sensing technology using radar, (short GE);

In order to assess the accuracy of these data sets a range of high-confidence benchmark elevation data was used including the following:

- Spot heights such as trigonometric beacons retrieved from the 1:50,000 topographic map series for the study area (2001 and 2007 edition) provided by the CD-NGI with a vertical accuracy of 0.3 m;
- Shaft collar elevations provided by DRDGold for 5 x shafts currently used to monitor the water level in the mine void and a range of other low lying shafts at ERPM. Determined by ground-based trigonometric surveys by registered and licensed surveyors these elevations are the most accurate and reliable data available.

It was found that in open terrain such as the mining belt that runs along the mined reef outcrops Google Earth elevation data matched the surveyed shaft collar elevations best with an average deviation of only 20 cm. GE data also compared best to the CDNGI 2001 set of spot heights selected in the area. Thus, all elevation profiles for the mining belt are based on GE data. Lidar and 5 m CDNGI data each underestimated the elevation by 4-5 m.

For the densely built-up area in the CBD the radar based GE data tend to overestimate the elevation by 10-15 m. Lidar and CDGI 5m data are very similar to each other and compare best to the 2007 spot height set. In order to be risk conservative the lowest elevation indicated by the 3 x data sets for determining the surface elevation of the key bank buildings was used.

Step 2: Determining the critical level for the bank buildings: Although no evidence exists that mine water would indeed corrode the concrete basement structure of the bank buildings, for safety reasons it was assumed that any contact of mine water with even the lowest part of the underground support structures constitutes a risk. Thus the elevation of the deepest part of the underground building structures were used for determining the critical level. As concrete anchors (termed 'piles') extent below the bottom of the lowest basement level into the underlying bedrock, the base of these 10-m-long piles was used (termed 'Pile Base Level'). The lowest PBL of the three key bank buildings was termed Critical PBL (CPBL) and used as a benchmark to assess any flooding risk. As the lowest lying building, the CPBL refers to the ABSA tower East with an elevation of 1704 m amsl. However, the building with the highest exposure to possible mine flooding is the new administration building of Standard Bank which is located directly on the outcrop of the Main Reef package which constitutes the entrance to the underlying deep mine void. The PBL of Standard Bank's new admin building is 1720 m amsl.

Step 3: Determining the final water level in the flooded mine void: As this level is controlled by a number of partly unknown factors certain assumptions had to be made in order to arrive at a reliable prediction. The interactions between these factors are described via a conceptual model. The most important of these assumptions is that the water table in the flooded mine void will be controlled by the level of the lowest lying outflow point (also termed ‘decant point’) as long as this point can accommodate all ingressing water and that all individual mine voids that make up the Central Basin are hydraulically interconnected. The final mine water table (FMWT) is also referred to as ‘decant level’.

In order to identify possible outflow (decant) points to assess whether the above mentioned conditions are met is was necessary amongst other:

(a) to conceptualise the structure and hydraulic interconnectivity of the mine void system (termed ‘basin’);

(b) to determine from what sources and along which pathways water was entering the mine void and how this may change as the void increasingly fills up;

(c) to determine the volume of the ingressing water as a proxy for estimating the volume of the expected decant and whether that can be accommodated by the identified outflow points.

Based on the above considerations the final mine water table elevation (decant level) was determined at **1614 mamsl** identifying the lowest lying still open shaft (Cinderella West Ventilation shaft at ERPM) as the most probable decant point.

Step 4: Assess the risks associated with the flooding of the mine void: Based on the height difference between the CPBL at 1704 masml and the decant level at 1614 mamsl a **safety margin of 90 m** exist for the ABSA tower East and of **106 m** for the PBL of Standard Bank exist. Provided that the identified decant shaft is kept open and allowed to act as a decant point this means that no risk of flooding exists for the key bank building or any building in the CBD for that matter. In case the shaft cannot, for some or other reason, be used as a decant point another 8 alternative shafts exists which all are still open, connected to the mine void and located below the critical PBL albeit at somewhat lower safety margins. In view of the former a flooding risks for the basement structures of any building in the CBD can be excluded.

This assessment takes account of all uncertainties associated with the data on which the report is based.

The chosen decant scenario underlying this risk assessment does not take possible interventions into account which are aimed at keeping the mine water level artificially below the natural decant level as all such interventions would only further lower the risks of basement flooding.

In addition to the basement flooding risks a number of other possible risks were considered including the following:

(1) Flooding-induced ground subsidence in the mined reef outcrop zones;

(2) Flooding induced seismicity;

(3) Exposure to radon emanating from shafts and polluted mine water;

(4) Pollution of surface water;

(5) Pollution of groundwater;

(6) Dolomite-related risks; and

(7) Damage to corporate and city image and business confidence.

Of the above listed hazards the subsidence of ground in low lying parts of the mined outcrop zone due to unconsolidated fill material being liquefied by mine water and the exposure of residents to the radioactive gas radon emanating from shallow mine water tables through porous overburden or mine shafts are perhaps posing the most serious threat to the geotechnical stability of structures and the health of residents.

From a more economic point of view it is suggested that the continued, often sensationalizing reporting in the news media on the mine void flooding and associated consequences, may have caused damage not only to the image of JHB and Standard Bank but also to the general business confidence.

As many of the investigated aspects of the mine void flooding in the CR were also addressed in the latest AMD report to the IMC, and later Cabinet, findings presented bear some relevance to the decisions taken based on the AMD report. This may be of interest as findings in this report differ partly significantly from statements made in the AMD report. This includes aspects we regard as crucial for designing solutions to the decant problem such as the expected volume of water decanting from the mine void, its probable quality, sources which contribute to ingress before and after the decant, ingress area, and intra-void water flow.

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