From: Oswell, Mike
To: CWPSC
Subject: RE: Questions Taken on Notice - 31 January Hearing (CWPSC) - Anglo response
Date: Thursday, 16 February 2017 9:10:40 AM
Attachments: Anglo resp Q on N. Att - 1,2,3. - 15.02.17.pdf

G'day Marion / CWP Committee,

Please find attached, in this and further emails to follow, the Anglo American response to 'Questions on Notice' asked by the CWP Committee on 31 January 2017.

This response and the various attachments are provided in a series of emails due to: a) the confidential nature of one of the attachments; and b) the substantial file size of some of the attachments that would potentially exceed email file size capability if sent in the one email.

The total response comprises 5 files on 5 emails as below:

- The Anglo responses and attachments 1,2 and 3 – file name: *'Anglo resp Q on N. Att – 1,2,3 – 15.02.17.pdf'*
- Attachment 3 – confidential

- Attachments 4, 5, 6 and 7 – file name: *'Attach 4,5,6,7.pdf'*
- Attachment 8 – file name: *'Attach 8.pdf'*
- Attachment 9 – file name: *'Attach 9.pdf'*

All attachments are fully referenced in the first file – the Anglo responses and attachment 1,2 and 3.

Please advise if you would like hard copies of the response and attachments.

I trust these responses meet the expectations of the CWP Committee – please do not hesitate to contact me if you require further information.

Kind regards,

Mike

Mike Oswell
Head of Safety, Health and Environment
Australia and Canada

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14th February 2017

The Chair
Queensland Government's Coal Worker's Pneumoconiosis Select Committee
Parliament House
George Street
Brisbane QLD 4000

Attention: Mrs Jo-Ann Miller

Dear Mrs Miller

Please find attached the Anglo American Coal Australia responses to the Questions on Notice arising from Anglo American personnel appearances before the Committee.

I trust this is satisfactory and assists the Committee in its deliberations on this critical health issue.

Please do not hesitate to contact me if you require further information.

Yours sincerely

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COAL WORKERS' PNEUMOCONIOSIS INQUIRY
PUBLIC HEARING – 31 January 2017- Brisbane
QUESTIONS TAKEN ON NOTICE AND ADDITIONAL INFORMATION
Anglo Coal – Dr Bharath Belle, Mr Mike Oswell, Ms Liz Sanderson and Mr Jordan Taylor

Anglo American Coal Australia responses to Questions Taken on Notice

Questions Taken on Notice:

1. ***Mr McMILLAN:*** *I will get to the activities that you have undertaken in the last two years in some detail in a moment and I want to take you to some of the aspects that you have addressed in your submission. Focusing on the period prior to 2015, there was a requirement for you to periodically undertake dust exposure monitoring?...*
   Mr Oswell: *Yes.*
   Mr McMILLAN: *That is a requirement under the legislation?*
   Mr Oswell: *Yes.*
   Mr McMILLAN: *That had to be done at least every quarter, is that right?*
   Mr Oswell: *The frequency of sampling is determined in conjunction with the occupational hygiene service provider. There are various formula worked out as to how frequently the sampling should occur and the size of the sampling group within the similarly exposed group and so on and so on. I would take that on notice unless somebody else can help me out here, but I am assuming it would have been at least four times a year in the underground sense.*
   (P 15)

Response Summary:

Prior to 2015 Anglo American underground mines conducted dust exposure monitoring as a minimum on a quarterly basis. During the years 2010 to 2015 the dust monitoring campaigns typically occurred more frequently i.e. on a monthly or two monthly basis. The frequency and numbers of persons sampled in specific Similar Exposure Groups (SEGs) was / is determined by the professional occupational hygiene service provider. Specific occupational hygiene monitoring programs or campaigns and associated schedules are subsequently determined in consultation with site safety and health professionals.

In the six years 2010 to 2015 Moranbah North Mine conducted 59 personal dust monitoring campaigns, and Grasstree conducted 41 campaigns. In Grosvenor's first year of operation, 2015, they conducted 11 campaigns.

Response Details: - For additional detail - refer **Attachment 1**

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2. ***Mr McMILLAN:*** *Was respiratory protective equipment mandatory for any exposure group within the underground mining environment prior to 2015?*
   Mr Oswell: *To be 100 per cent accurate I will have to refer back to the producers* (procedures). *There are different procedures at different mines. The mandatory wearing of protection often commenced at certain points along longwalls and in development areas, but specifically I would have to come back to you on that at each of the mines: Moranbah, Grasstree.*
   CHAIR: *Can you take that on notice, please.*
   Mr Oswell: *Sure.*
   Mr McMILLAN: *I am particularly interested to understand the historical context. Your submission speaks obviously in the present tense, which is fine and we will get to that, but in terms of respiratory protective equipment I think your submission says that that is mandatory for longwall operators and those working in development at present?*
   Mr Oswell: *Correct.*
   Mr McMILLAN: *That is generally consistent with your understanding?*
   Mr Oswell: *Yes.*
   Mr McMILLAN: *But you are not able today to speak to when that mandatory requirement was imposed; is that right?*

Mr Oswell: *Correct. I will have to come back to you on that.*
(P 15)

Response Summary

Reviews of historical dust management and associated procedures at the Anglo American Coal Australia underground mines have revealed there have been mandating respiratory protection requirements specified in procedures for specific workers and specific operational areas / activities since the early development of site safety and health procedures. Moranbah North mine procedures date back to 2002, Grasstree to 2007 and Grosvenor to 2013 during its project development.

Appropriate extracts from relevant procedures are provided in **Attachment 2**.

Response Details: - Extracts from dust management procedures dealing with personal protective equipment requirements at the three mines are summarised in **Attachment 2.**

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3. ***Mr Oswell:*** *Each position in Anglo has an associated position description and the various requirements of the position and the competencies and qualifications that go with that position are detailed.*
   CHAIR: *Can you provide us with those position descriptions?*
   Mr Oswell: *That would be easiest, yes. There is a range of potential qualifications for positions, yes.*
   (PP 15-16)

Response Summary:

All employee positions within Anglo American - Coal Australia have documented role profiles that describe the reporting structure, key outputs and accountabilities and the required experience, skills and qualifications. These requirements vary between roles and are updated as the business needs change.

Response Details: Attachment 3 (in a separate file – refer below) contains examples of role profiles for:

- Head of Safety and Sustainable Development, Anglo American – Coal Australia & Canada
- Occupational Health Specialist, Anglo American – Coal Australia & Canada
- Safety, Health and Environment Manager, Anglo American Australian Mine Site

Please note the style of these role profiles vary essentially as a result of when the role profile was developed. Given the role profiles reflect Anglo American human resource personnel policies and processes, **it is requested that these documents remain confidential to the CWP committee.**

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4. ***Mr McMILLAN:*** *Could I ask you to take on notice collectively or even through Mr Hobson this question. What inquiries were made as to the experience of the proposed specialist who was engaged to read X-rays for the Grasstree workforce before his engagement and at what level was that engagement approved?*
   Mr Oswell: *Yes.*
   (P 22)

Response Summary

Question re - inquiries re the experience of the proposed specialist:

- Mr David Lawrence, Safety Health and Environment Manager for Grasstree Mine proposed the use of Dr Nigel Sommerfeld's services to undertake reading of Grasstree employee chest x-rays. In this regard, Dr Sommerfeld was requested to provide his CV and a further recommendation was sought from a thoracic physician on his experience with thoracic diseases. In addition, Dr Sommerfeld's qualifications were checked with Australian Health Practitioners Registration Association (AHPRA) to verify his qualifications. A summary of Dr Somerfeld's qualifications and experience obtained prior to his engagement are presented in **Attachment 4.**

Question re - what level was that (Dr Sommerfeld's) engagement approved – refer question 5 below.

Response Details: - A summary of Dr Sommerfeld's qualifications and experience are presented in **Attachment 4**

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5. ***Mr McMILLAN:** I want to break that down a little more, if I can. First of all, were you involved in the decision that your workforce needed to be offered the opportunity to have chest X-rays read by a second person other than the specialist radiologist who had already been engaged?*
***Mr Oswell:** There was a whole range of discussions around that time and certainly with our chief medical officer. That was how the process was established.*
***Mr McMILLAN:** At what level of the organisation was that decision ultimately taken?*
***Mr Oswell:** I will have to come back to you on that. I will have to review the various discussions around all of that, so I will take that on notice.*
(P 23)

Response Summary:

In November and December 2015, the extent of the issues and concerns pertaining to the identified cases of coal workers pneumoconiosis was escalating rapidly. Given the involvement or potential involvement of the three Anglo American underground mines and the potential scale of the emerging pneumoconiosis issue, a forum of senior underground site and corporate personnel was formed to consider the various issues and how best these should be addressed. The persons variously involved in this forum included the underground site General Managers / Site Senior Executives, the underground site Safety Health and Environment Managers, the Head of Underground Operations, the Coal Australia Head of Safety Health and Environment, the Coal Australia Occupational Health Specialist and the Coal Business Executive Head of Human Resources and Corporate Affairs.

On the 24th and 25 November 2015, discussions were held around the intended approach to proposed further chest x-ray programs for underground employees and the use of 'specialist' radiologists to read all x-rays. It should be noted that this discussion was based partly on the fact that Anglo American was not confident that all previously taken chest x-rays had in fact been read by a radiologist – particularly for employees at the Grasstree Mine.

As such, a decision was taken by the abovementioned forum that all chest x-rays for Anglo American Coal Australia employees must be read by 'specialist' radiologists[1]. At that time, the forum did not specify exactly who those 'specialist' radiologists should be (note this was before the DNR&M developed an 'approved' list of radiologists - nominated by the Royal Australian and New Zealand College of Radiologists).

The x-ray program and the meaning of "specialist radiologist" was discussed further by the above forum during teleconferences on the 15, 16 & 18 December 2015. Advice on the meaning of a "specialist radiologist" was also sought from the then Deputy Director General of the Department of Natural Resources and Mines - Mr Paul Harrison and further discussions on understanding specific thoracic radiology were held with Dr Rob McCartney. Notes from those discussion are available to the committee upon request.

However, as above, the ultimate decision to specifically engage Dr Sommerfeld rested with the General Manager / Senior Site Executive of Grasstree Mine on advice provided by the site Safety Health and Environment Manager (who had a medical background). The principle of ensuring all Coal Australia underground employee chest x-rays were read by 'specialist' radiologists was established by

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[1] Note – at this time it was generally assumed that all radiologists were well versed and capable in the reading of reading chest x-rays for even low levels of coal worker pneumoconiosis.

the abovementioned forum of senior site and corporate managers. This forum supported the engagement of Dr Sommerfeld for reading of Grasstree employee chest x-rays.

__Response Details:__ Notes from the outcomes of discussions of the abovementioned forum are available to the Committee upon request – **Attachment 5**

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6. ***Mr McMILLAN:*** *Can I ask you to take the following questions on notice arising from the line of questions I have just asked you?*
    a. *What evidence can you produce to satisfy the committee that the US specialist engaged for the purposes of the B reading was qualified and accredited by NIOSH?*
    b. *When was the offer made to the Grasstree workforce to have chest X-rays secondary read in the United States and how was that offer conveyed?*
    c. *Was any similar offer made to the workforce at Moranbah North and Grosvenor prior to the establishment of the DNRM B-reading process?*

    Mr Oswell: *As a matter of process, is somebody summarising these questions?*

    Mr McMILLAN: *We will send you a list of the questions.*
    (P 25)

__Response Summary:__

This question has three parts - these are responded to separately below:

a) Evidence that the US based B Readers selected for Anglo American Coal Australia were accredited by NIOSH includes the individual B reader certificates obtained through Dr Robert McCartney that noted the currency of the B reader accreditation and documents downloaded from the US National Institute for Occupational Safety and Health (NIOSH) web site showing the NIOSH certified listing of B readers. Also attached is the proposal by Dr McCartney to provide that B reader service. This service was arranged before senior managers at the Grasstree Mine and corporate Anglo American Coal Australia were aware that the DNRM had changed the process to disallow any B Readers other than those from University of Illinois, in particular - Dr Robert Cohen (Copies of the relevant documents, certificates and proposal by Dr McCartney are provided in **Attachment 6).**

In this regard, it should be noted that the original Department of Natural Resources and Mines (DNR&M) Fact Sheet on the 'Two Reader' process dated 27th July 2016 provided for the use of alternate B Readers in the US. Given the expected substantial chest x-ray reading load that would be experienced by Dr Cohen, it was agreed that alternate US B readers would be utilised by Anglo American Coal in the short term.

Subsequently, in early September 2016, Anglo American Coal Australia learnt that a further / updated Fact Sheet on the 'Two Reader' process had been developed by the DNR&M. This updated Fact Sheet no longer provided for coal mines to utilise their own US B reader process and mandated that all chest x-rays must be channelled via the DNR&M Health Services Unit to Dr Cohen[2]. As such, the alternate B Reader process that had been established for Anglo American Coal Australia was never utilised and Anglo America adopted the latest DNR&M 'Two Reader' process. The original and updated Fact Sheets are included in **Attachment 6.**

b) The offer made to Grasstree employees to have x-rays read a second time was provided by way of communication of a 'SHE Brief' that was provided to all crews at start of shift on 25 August 2016. This 'SHE Brief' is included in **Attachment 6.**

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[2] It should be noted that this updated DNR&M Fact Sheet on the Two Reader Process was only issued to Nominated Medical Advisers on the 14th August 2016. Hence, there was some delay before Anglo American became aware of the changes specified in the updated Fact Sheet.

c) Moranbah and Grosvenor workforces were offered the opportunity to have recent chest x-rays read in accordance with the updated Fact Sheet on the Two Reader process once Anglo American Coal Australia became aware of this updated process.

Response Details: For additional detail on a) and b) above - refer **Attachment 6.**

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7. ***Mr McMILLAN:*** *I am trying to get a picture for the committee of doctors who are already appointed as nominated medical advisers and then in 2016 under the oversight of your chief medical adviser were reappointed as nominated medical advisers—what actual experience those doctors have of the underground environment and the actual jobs that the workers they are assessing do and what is the basis of that experience. I would like some specific detail for the committee about that. If you are not in a position to answer that, I will ask you to take it on notice and give us a detailed explanation of what experience those doctors have and how you are satisfied that they are adequately able to understand the environment of the workers they are assessing.*
Mr Oswell: *Certainly.*
(PP 29-30)

Response Summary:

The Nominated Medical Advisors (NMAs) appointed have considerable experience in undertaking the medical examinations required of the Coal Mine Workers Health Scheme (CMWHS). **Attachment 7** contains a listing of the currently appointed / re-appointed local medical practitioners who are the NMAs for Anglo American Coal Australia underground sites. The table also lists:

- the average number of Coal Mine Worker Health Scheme medicals undertaken annually by each doctor;
- the date of their original appointment as NMAs for the mine; and
- the timing of most recent visit by the NMA to the relevant Coal Australia underground mine site. It is noted that first-hand, detailed knowledge of the underground environment does not necessarily enhance the professional or technical understanding of the medical parameters to be examined or measured during a health examination. However, it is preferable that NMAs visit the work environment to aid in understanding specific physical risks and tasks associated with job roles and hence, fitness for duty requirements,.

Additional information provided to and available for the NMAs include a summary of the Similarly Exposed Groups (SEGs), task descriptions associated with each SEG and information regarding historical dust and noise exposure.

Further, the NMA appointment letter now includes a requirement that each NMA visits the relevant underground site on at least an annual basis.

Finally, Anglo American Coal Australia has appointed a Chief Medical Officer - a Specialist Physician in Occupational and Environmental Medicine who is well acquainted with the underground mining environment. This Chief Medical Officer has contact with each NMA and is able to advise each of these NMAs on issues related to the sites requirements for fitness for duty and specific details relating to the medical surveillance processes within the CMWHS medical assessments.

Response Details: For the additional details noted above see **Attachment 7.**

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8. ***Mr McMILLAN:*** *Thank you. Moving now to the dust management section of your submission—and we are approaching the end, I assure you; I am grateful for your patience and that of the committee—I wanted to ask you about the establishment of dust committees at your underground mines. When did that initiative happen?*
Mr Oswell: *I cannot tell you exactly, Ben. I think that is a question for Tim Hobson tomorrow. He could tell you exactly when the Grasstree committee was formed.*
Mr McMILLAN: *Have committees been formed at the other underground operations as well?*

Mr Oswell: Yes. With regard to the timing of that, I will take that on notice. We will find out when they were established.
Mr McMILLAN: I will frame the question in this way, if you can take it on notice please: when were dust management committees established at each of Anglo's underground mines and how many times have each of those committees met since their establishment?
Mr Oswell: Yes.
(P 34)

Response Summary:

Dust Committees were established in each of the underground sites in 2015 – Moranbah originally in May 2015, Grosvenor in November 2015 and Grasstree in December 2015. Dust Committee meetings at each mine are conducted either on a fortnightly or monthly basis – the frequency varying according to the urgency of issues pending and/or the frequency of receipt of dust monitoring results. All site Dust Committees comprise a mix of management personnel and operational staff including workforce representatives.

Additionally, the Head of Underground Operations chairs a Dust Mitigation and Respiratory Protective Equipment review meeting on a fortnightly basis. Participants in this meeting include the underground site General Managers / SSEs, site Safety Health and Environment Managers, a number of internal technical specialists and external experts as required such as personnel from the CSIRO. This committee and the meetings commenced in June 2016.

Examples of minutes from the various Dust Committee meetings are available to the Committee upon request.

Response Details: N/A

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9. ***Mr McMILLAN:*** *I take it then that the position of dust champion was initiated at the same time as those committees as part of a suite of improvements?*
Mr Oswell: *The exact timing I am not sure but, in terms of the appointment of these dust champions as such, that was a term that was coined at Moranbah North. In fact, we are underselling ourselves. As you saw at Grasstree, there are in fact a number of dust champions, if you like, specifically from engineering, from production, from the safety side of things all leading the charge on coordinating and implementing all of the dust improvement initiatives, so there is a range. In fact, there is quite a structure at each of the mines about the various people who were directly involved in initiating and following through on those improvements.*
Mr McMILLAN: *Do I take it that both of those initiatives are part of the suite of efforts that have been made by Anglo since the reidentification or the rediagnosis of new CWP cases in the last two years?*
Mr Oswell: *Correct. I am unsure whether there were dust committees before that time, but I will find out.*
Mr McMILLAN: *Thank you…*
(P. 34)

Response Summary:

No records of specific Dust Committee styled meetings have been located prior to 2015 however, the processes for the workforce to raise concerns about dust (and other hazards / safety concerns) has always included hazard reporting processes, regular crew and shift based safety meetings, raising concerns with supervisors and Site Safety and Health Representatives.

Response Details: NA

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10. ***Mr McMILLAN:*** *In September 2015 a directive was issued requiring mandatory use of respiratory protective equipment for all personnel working on or entering the longwall until*

exposures had been reduced to acceptable levels. Wasn't respiratory equipment already required as mandatory for those working on the longwall face at Grasstree?
Mr Oswell: *We touched on that before. I am not sure whether it is a question on notice. I cannot off the top of my head recall the exact details of those procedures and when they changed.*
(P. 35)

Response Summary:

Refer response to Question 2.

Response Details: For detail refer **Attachment 2**

Additional Information 1:

CHAIR: *In relation to the comments that we made earlier about the directions[3], if you have any evidence of different mines inspectors advising you to do things in different ways would you please provide that to this committee because we would be very interested in that. We certainly understand the confusion that obviously exists. If you could do that that would be good. Counsel assisting, do you have anything?*
(P. 42)

Response Summary:

The dust management situation, various mine responses and the input of the Inspectorate unfolded rapidly in 2015 /2016 – this was a very dynamic situation that required multiple responses at a variety of levels.

During this period there were a number of dust related 'Directives' issued to Moranbah North and Grasstree Mine by a number of Mines Inspectors. Many of these dust related Directives were multifaceted requiring action or confirmation of actions on numerous fronts. The nature of the Directives included requirements regarding reviews of procedural / administrative controls and use of personal protective equipment; increased dust monitoring regimes and the identification and implementation of engineering controls.

During the time the sites were addressing the Directives, there were numerous discussions with Inspectors (primarily during site visits) seeking clarity around specific aspects of the Directives and what actions / evidence was required to close-out the Directive or a part of it. As such, it was probably not surprising there were differing views at different times between Mines Inspectors and senior site personnel regarding dust management improvement measures and the means of specific and satisfactory close-out of Directives. Some of the issues that arose included:

- The specific composition of the SEG associated with longwall operations that were to be the 'reference' group for compliance monitoring;
- The duration of the time period during which dust compliance had to be achieved for close-out of the directive;
- The definitions and interpretations of exceedances e.g. was it one person – one exceedance, the average result for a SEG over one sampling period; the Upper Confidence Limit (UCL) exceeding the Occupational Exposure Limit (OEL) etc;
- The specific nature of the sampling program – number of persons monitored per shift and the frequency of monitoring campaigns;
- The specific performance criteria required to achieve close-out of the Directives; and
- The relatively short time frame (in some cases) between formal meetings between Anglo and the Chief Inspector.

While there were differing views around these issues at different times, on site meetings with Inspectors and more formal meetings involving Anglo American site and corporate senior managers

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[3] Assumed to mean 'Directives'

and members of the Inspectorate and DNR&M (including the Chief Inspector and the Inspector of Mines – Occupational Hygiene) ultimately led to resolution and concurrence on these matters.

Response Details: **N/A**

Additional Information 2:

Mr McMILLAN: *Just a procedural matter. Through Anglo representatives a number of documents have been provided to the committee confidentially, including a briefing paper, a power point presentation and some documents that Dr Belle provided. I understand that there is the potential for Anglo to provide some or all of those documents in a form that can be published by the committee. I think that would be most useful for the committee's work. Through you, if I could invite Anglo to provide those documents in whatever form they are willing for them to be published that would be helpful.*
CHAIR: *That would be good. You would have to make that very clear. Also documents that you would like to be private to the committee could you make that clear as well. Thank you very much for being here today…*
(P. 43)

Response Summary:

Requested documents as below are provided in non-confidential / restricted format.

Response Details:

- 'QLD Select Committee Rockhampton Meeting (12 Dec 2016) Transcript Responses - Dr B. Belle, Anglo American' – **Attachment 8.**
- 'Position Paper – PDM 3700', 26 September 2016 (document) – Anglo American / Glencore – **Attachment 9.**
- 'Position Paper – PDM 3700', 28 September 2016 (Powerpoint) – Anglo American / Glencore **– Attachment 9.**

QoN Attachment 1

Coal Workers Pneumoconiosis Inquiry
Public Hearing - Tuesday 31 January 2017 Brisbane
Questions Taken on Notice and Additional Information
Anglo Coal

Dust Sampling Regimes Anglo Underground Mines 2010 to 2015

Site	Year	Approx. Frequency	# Monitoring Campaigns	# Samples Results
Moranbah	2010	Monthly	11	78
Grasstree	2010	Monthly	11	91
Moranbah	2011	2nd Monthly	6	120
Grasstree	2011	2nd Monthly	8	83
Moranbah	2012	Monthly	11	284
Grasstree	2012	Monthly	9	118
Moranbah	2013	Monthly	10	70
Grasstree	2013	2nd Monthly	6	97
Moranbah	2014	Monthly	10	71
Grasstree	2014	Quarterly	3	50
Moranbah	2015	Monthly	11	162
Grasstree	2015	Quarterly	4	97
Grosvenor	2015	Monthly	11	46

QoN Attachment 2

Coal Workers Pneumoconiosis Inquiry
Public Hearing - Tuesday 31 January 2017 Brisbane
Questions Taken on Notice and Additional Information
Anglo Coal

Question related to Mandatory Requirements for Respiratory Protection (P15)

Dates listed below are the publication dates of specific documents for each of the underground sites, Moranbah, Grosvenor and Grasstree – the document control process requires that the document number and document title generally stay the same but "version date" is at the time of the publication of the document that contains amendments.

Moranbah Site Documents # *50105 Procedures for the Use of Personal Protective Equipment* Versions dated from **25/05/2002 to 08/01/2016** state:

"Respiratory Protection

Approved respirators are mandatory in the following areas on the surface and underground:

- *Near coal transfer points and crushers where coal dust is generated;*
- *Coal cutting operations for the continuous miner and longwall shearer operation. Any person required working near or on the return side of the cutting operation. This practice must be kept to an absolute minimum;*
- *Machine operations which generate dust such as brushing roof or floor with a miner or road header or cleaning up with an LHD loader. Airborne dust must be reduced to an absolute minimum by the use of effective water spray systems and watering down dusty areas as required", and*

"Generally, the following types of respirators shall be used unless otherwise determined by risk assessment:

- *Dust respirators for naturally contaminated atmospheres, including drilling operations where dust suppression fails;*
- *Supplied–air helmets are provided for Longwall operators due to the time of exposure and the*
- *concentrations of dust present;"*

Additionally Document # *50108 Procedure for Management of Respirable Dust* Versions dated from **04/03/2002 to 01/10/2015** state:

"Respiratory Protective Equipment (RPE)

If there is no practical way of reducing respirable dusts to allowable levels at the longwall face, operators at the Longwall are to be provided with, and wear, RPE at all times. Supplied air helmets are the preferred form of RPE and will be made available to all persons working at the longwall."

Grosvenor Site Documents # *GRO-239- Personal Protective Equipment* Versions dated from **15/08/2013 to 27/09/2016** state:

Respiratory Protection Equipment (RPE)

"A RPE program shall be developed and implemented in each area where RPE is required to be carried or worn……

…….. Dusty environments shall, as a minimum, require the use of disposable respirators (dust masks) that are NIOSH approved for the type of environmental condition. In areas where a DUST PROTECTION sign is displayed, the wearing of RPE is mandatory. Notwithstanding the above, all

areas inbye of the last open cut through, all homotropal gate road returns and all mains returns are mandatory areas in which RPE must be worn."

Additionally Document *#GRO-297-HMP Management of Inhalable and Respirable Dust* Versions dated from **10/10/2013 to 20/09/2016** state:

> ***"Mandating the wearing of PPE*** during production and other dust generating support or maintenance tasks and ensuring those personnel are trained to fit and maintain the integrity of the respiratory protection equipment"

<u>Grasstree Site Documents</u> *# SOP.UGGT.065. - Personal Protective Equipment* Versions dated from **01/01/2007 to 01/08/2012** state:

> "**Respiratory Protection**
>
> *Respirators of the approved type, selected in compliance with AS1715 must be worn whenever dusts, fumes, gases, or other harmful atmospheres are present.....*
>
> *The rules for respiratory protection are as follows:*
>
> *Dust respirators for naturally contaminated atmospheres, including drilling operations where dust suppression fails. Supplied air helmets are required for Longwall operators due to the time of exposure and the concentrations of dust present.*
>
> ***"Approved respirators are mandatory in the following areas on the surface and underground:***
>
> *Coal cutting operations for the continuous miner and longwall shearer operation. Any person required, to work near or on the return side of the cutting operation**. This practice must be kept to an absolute minimum,*** and

Versions from **26/09/2014 to 15/09/2016** state:

"Approved respirators are mandatory in the following areas on the surface and underground:

> *Coal cutting operations for the continuous miner and longwall shearer operation. Any person required to work near or on the return side of the cutting operation*
> *Machine operations which generate dust such as brushing roof or floor with a miner or road header or cleaning up with a LHD loader. Airborne dust can be controlled by means of effective engineering controls (i.e. effective water spray systems and watering down dusty areas as required)."*

Additionally from Document #GSHMS008 - *Safety & Health Management System Respirable Dust / Silica* versions dated **20/04/2005 to 01/06/2016** state:

> *"**Respiratory Protective Equipment (RPE)***
>
> *Any area where the dust levels are above the concentration limits and reduction is not practicable; the area is to be signposted as a "Respiratory Protection Area". All persons in a "Respiratory Protection Area" shall wear a particulate respirator that meets the standard prescribed in AS 1715.*
>
> *Operators at the Longwall are to be provided with, and wear, supplied air helmets or suitable particulate respirators to P2 Standard.*
>
> *Where maintenance is carried out in a "Respiratory Protection Area" and the equipment or activity that causes the dust is shut down, the supervisor may cover the "Respiratory Protection Area" signs for the duration of the maintenance work.*

Copies of the relevant procedures can be provided to the Committee upon request.

QoN Attachment 5

Coal Workers Pneumoconiosis Inquiry
Public Hearing - Tuesday 31 January 2017 Brisbane
<u>Questions Taken on Notice and Additional Information</u>
Anglo Coal

The level of the organisation at which the decision to have 'specialist radiologists read all Coal Australia underground employee chest x-rays (P.23)

Notes summarising the outcomes of discussions of the Anglo American Coal Australia senior management forum that was addressing the various pneumoconiosis issues around November and December 2015, are available to the Committee upon request.

QoN Attachment 6

Coal Workers Pneumoconiosis Inquiry
Public Hearing - Tuesday 31 January 2017 Brisbane
<u>Questions Taken on Notice and Additional Information</u>
Anglo Coal

Anglo selection of US based B Readers / X-ray offers (P25)

1. Listing of B Readers from the NIOSH website;
2. Certificates provided by Dr Robert McCartney to note accreditation;
3. Proposal by Dr McCartney to provide a B Reader service;
4. DNR&M Fact Sheets on the 'Two Reader Process'; and
5. The Grasstree 'SHE Brief' providing information to the workforce regarding the second B Reader process

U.S. DEPARTMENT OF HEALTH AND HUMAN SERVICES
Public Health Service

Centers For Disease Control and Prevention
National Institute For Occupational Safety and Health
Appalachian Laboratory For Occupational Safety and Health

THIS IS TO CERTIFY THAT

Kenneth Charles Fortgang, M.D.

HAS SATISFACTORILY COMPLETED TRAINING/TESTING AND IS A DESIGNATED

B READER

AS STIPULATED IN CFR TITLE 42, PART 37.51.
FEDERAL MINE SAFETY AND HEALTH ACT OF 1977 AND ITS AMENDMENTS*
This Certification will remain in effect from **10/1/2014** until **9/30/2018**

David N Weissman, MD

DIRECTOR, DIVISION OF RESPIRATORY
DISEASE STUDIES, ALOSH/NIOSH

CDC
CENTERS FOR DISEASE CONTROL AND PREVENTION

*NIOSH does not regulate or monitor classification of chest images performed for non-NIOSH purposes.

U.S. DEPARTMENT OF HEALTH AND HUMAN SERVICES
Public Health Service

Centers For Disease Control and Prevention
National Institute For Occupational Safety and Health
Appalachian Laboratory For Occupational Safety and Health

THIS IS TO CERTIFY THAT

John Carroll DeMocker, M.D.

HAS SATISFACTORILY COMPLETED TRAINING/TESTING AND IS A DESIGNATED

B READER

AS STIPULATED IN CFR TITLE 42, PART 37.51,
FEDERAL MINE SAFETY AND HEALTH ACT OF 1977 AND ITS AMENDMENTS
This Certification will remain in effect from **5/1/2013** until **4/30/2017**

David N Weissman, MD
DIRECTOR, DIVISION OF RESPIRATORY
DISEASE STUDIES, ALOSH/NIOSH

CDC
CENTERS FOR DISEASE CONTROL AND PREVENTION

EXTRACT FROM:

The National Institute for Occupational Safety and Health (NIOSH)

Providing National and World Leadership to Prevent Workplace Illnesses and Injuries NIOSH

NIOSH Certified B Readers

Physicians from inside the United States that have who have demonstrated competence in applying the ILO classification by successfully completing the NIOSH B Reader examination within the last 4 years. Listing does not imply medical licensure.

Listed by Reader's Name

Name	Department	Street	City	State	Zip	Phone
ROGER A ABRAHAMS	MORGANTOWN PULMONARY ASSO	1265 PINEVIEW DR	MORGANTOWN	WV	26505	(304) 598-2336
DAVID RAPHAEL ABRAMOWITZ		12 STONY POINT RD	CHARLESTON	WV	25314	(304) 549-1414
KIM ALLYN ADCOCK	RADIOLOGY	11 GOLDEN EAGLE LANE	LITTLETON	CO	80127	(303) 641-4585
AFZAL UDDIN AHMED	PRINCETON COMMUNITY HOSP	12TH STREET EXTENSION	PRINCETON	WV	24740	(304) 487-7168
MARK JASON AKERS	RADIOLOGY INC	5221 US RT 60 EAST	HUNTINGTON	WV	25712	(304) 522-1550
MICHAEL SHEPARD ALEXANDER		699 PERKINS RD SE	VALDESE	NC	28690	(828) 879-2880
TIMOTHY EDWARD ALLEN	RADIOLOGY AND NUCLEAR MED	1303 SW FIRST AMERICAN PLACE	TOPEKA	KS	66604	(785) 234-3451

MICHAEL JOSEPH ALLINE	WEST JEFFERSON MED CENTER	1111 MED CTR BLVD,STE N108	MARRERO	LA	70072	(504) 349-1461
DANIEL ROBERT ALZHEIMER	CMMC	1334 TOBOGGAN SLIDE LANE	LEWISTOWN	MT	59457	(307) 751-6257
JUDITH KOREK AMOROSA	ROBERT WOOD JOHNSON U HOS	60 PROSPECT STREET	SOMERVILLE	NJ	08876	(732) 887-7140
HENRY A ANDERSON III	WI DIVISION OF PUB HEALTH	200 LAKEWOOD BOULEVARD	MADISON	WI	53704	(608) 241-1227
KENNETH CHARLES ANDERSON		1 WOODLEA LANE	LOUISVILLE	KY	40207	(502) 893-1684
GLEN RAY BAKER		1200 RAVENWOOD DRIVE	CORBIN	KY	40701	(606) 528-0141
STEPHEN GEORGE BASHEDA		812 WHITE OAK CIRCLE	PITTSBURGH	PA	15228	(412) 572-6168
JAMES KEVIN BENJAMIN	ALLEGANY IMAGING	PO BOX 3206	LAVALE	MD	215043206	(240) 964-1035
ROGER A BERG		21 WATCHUNG RD	SHORT HILLS	NJ	070783029	(973) 467-1180
RICHARD CARL BERNSTEIN		4902 CARLISLE PIKE #316	MECHANICSBURG	PA	17050	(570) 814-0183
GERALD ALAN BLACK		5132 MUIRFIELD DRIVE	FAYETTEVILLE	NY	13066	(315) 637-5471
DONALD A BREYER		926 FAIRWAY DRIVE	SONOMA	CA	95476	(707) 933-8327
BRUCE CHARLES BROUDY	LEXINGTON CLINIC	1225 SOUTH BROADWAY	LEXINGTON	KY	40504	(859) 229-0929
HAROLD DEAN CAIN		6309 SHADOW MOUNTAIN DRIVE	AUSTIN	TX	78731	(512) 653-2430
			HILTON HEAD			(843)

JAMES RICHARD CASTLE		37 SAVANNAH TRAIL	ISLAND	SC	29926	682-8223
KENNETH JOSEPH CAVORSI	MEDICAL IMAGING OF LEHIGH	3437 STURBRIDGE PLACE	ALLENTOWN	PA	18104	(215) 620-5162
RONALD R CHERRY	SWEETWATER HOSPITAL	304 WRIGHT ST	SWEETWATER	TN	37874	(865) 675-7187
JONATHAN HERO CHUNG	UNIVERSITY OF CHICAGO	DEPT. OF RAD. 5841 S. MARYLAND AVE.	CHICAGO	IL	60637	(720) 235-8674
WILLIAM D CLAPP	JH STROGEN HOSPITAL	1901 W HARRISON ST, ROOM 2815	CHICAGO	IL	60612	(312) 864-2915
ROBERT ANDREW COHEN	UNIVERSITY OF ILLINOIS	2121 WEST TAYLOR, SUITE 160	CHICAGO	IL	60612	(312) 413-5267
MARK S COLELLA	ALLEGHENY VALLEY HOSP	1301 CARLISLE STREET	NATRONA HEIGHTS	PA	15065	(724) 226-7863
PATRICK MCMILLAN CONOLEY	KELSEY-SEYBOLD CLINIC	4527 NENANA DRIVE	HOUSTON	TX	770353627	(713) 442-2527
KENNETH SUNGHO COOKE	ROOSEVELT HOSPITAL	1000 AMSTERDAM AVE	NEW YORK	NY	10027	(212) 523-4260
CHRISTIAN WARREN COX	NATIONAL JEWISH HOSPITAL	1400 JACKSON ST	DENVER	CO	80206	(303) 270-2810
JEFFREY ROBERT CRASS	NORFOLK GENERAL	917 BOBOLINK DRIVE	VIRGINIA BEACH	VA	23451	(757) 466-0089
COURTNEY CRIM		10912 ASHLAND MILL COURT	RALEIGH	NC	27617	(919) 749-8970
JAMES BRANDON CRUM	UNITED MEDICAL GROUP	545 ZIEGLER DR	PIKEVILLE	KY	41501	(606) 793-7654
RAYMOND THOMAS CUMMINS	CALIFORNIA PACIFIC MED CT	1 MEADOWOOD DR	LARKSPUR	CA	94939	(415) 924-5254
ANNE MCBRIDE CURTIS	YALE NEW HAVEN HOSPI	PO BOX 8042	NEW HAVEN	CT	[illegible]	(203)

						278-7213
AUREA S DE SOUZA	EASTERN NIAGARA RADIOLOGY	222 GENESEE STREET	BUFFALO	NY	14203	(716) 860-8416
GEORGE LUIS DELCLOS	RESP CONS OF HOUSTON	6550 FANNIN #2403	HOUSTON	TX	77030	(713) 790-6250
David Matthew DeLonga	NAVAL MED CTR PORTSMOUTH	6513 HARBOUR POINTE DRIVE	SUFFOLK	VA	23435	(850) 712-6586
CRAIG ALLEN DELORD		1614 BRAXTON CIRCLE	NEDERLAND	TX	77627	(409) 722-0359
JOHN CARROLL DEMOCKER		20 MERRY HILL LANE	PITTSFORD	NY	14534	(585) 248-9120
TERRENCE CONSTANT DEMOS	LOYOLA MED UNIV CTR	2160 S FIRST AVE	MAYWOOD	IL	60153	(708) 216-8625
KATHLEEN ANN DEPONTE	DIAGNOSTIC IMAGING ASSOCI	935 Virginia Avenue NW	NORTON	VA	24278	(276) 275-3742
FREDERICK MAST DULA JR		1109 SUMTER COURT	SALISBURY	NC	28144	(704) 213-6955
WILLIAM HARVEY DURHAM	FGH	131 PINEHILL DRIVE	HATTIESBURG	MS	39402	(601) 270-0560
MICHAEL J EISENBERG	RADIOLOGY ASSOC OF NV	2945 MONDAVI COURT	LAS VEGAS	NV	89117	(702) 243-0752
ORN ELIASSON		9106 PHILADELPHIA RD SUITE 208	BALTIMORE	MD	21237	(410) 391-0646
ILAN ALLAN FEINGOLD		120 GAVILAN AVENUE	CORAL GABLES	FL	33143	(305) 609-2889
DAVID EUGENE FINLAY	UT HEALTH SCIENCE CENTER	11937 US HWY 271	TYLER	TX	75708	(903) 877-7108
GREGORY JOHN FINO	ST CLAIR HOSPITAL	200 SUGARWOOD DRIVE	VENETIA	PA	15367	(412)

JONATHAN HAROLD FISH		33 BONITA AVE	PIEDMONT	CA	94611	(510) 681-8940
JOHN RANDOLPH FOREHAND		1 CLINIC DRIVE	RICHLANDS	VA	24641	(276) 964-1229
KENNETH CHARLES FORTGANG	PREMIER RADIOLOGY	1125 BEL AIR DR	HIGHLAND BEACH	FL	33487	(561) 756-4610
ALFRED FRANZBLAU	UNIVERSITY OF MICHIGAN	1415 WASHINGTON HEIGHTS	ANN ARBOR	MI	481092029	(734) 936-0758
ERIC JONATHAN FREEMAN		1318 WESTOVER AVE	NORFOLK	VA	23507	(757) 484-5900
ARNOLD C FRIEDMAN		12419 NORTH VIA TUSCANIA AVENUE	CLOVIS	CA	93619	(520) 247-5302
CARL R FUHRMAN	UPMC RADIOLOGY SUITE 201	200 LOTHROP STREET	PITTSBURGH	PA	15213	(412) 647-7288
JOSEPH IGNATIUS GAGLIONE		1241 WOODLAND AVENUE	MT PLEASANT	SC	29464	(843) 881-4020
DANIEL RAPHAEL GALE	LOWELL GENERAL HOSPITAL	160 PAUL REVERE ROAD	NEEDHAM	MA	02494	(978) 937-6240
MARK ELON GALE	LOWELL GENERAL HOSP.	190 PARKER ROAD	NEEDHAM	MA	02494	(781) 444-5955
RAMON JULIO GARCIA	SE TEXAS IMAGING LLP	1323 S 27TH STREET #700	NEDERLAND	TX	77627	(409) 729-5400
KAREN KODSI GARFIELD	MOUNT SINAI HEALTH SYSTEM	510 EAST 89TH STREET	NEW YORK CITY	NY	10128	(212) 523-4260
DOMINIC JOSEPH GAZIANO		3100 MACCORKLE AVE SE	CHARLESTON	WV	25304	(304) 346-1811
WARREN BRUCE GEFTER	HOSP OF THE UNIV OF PA	3400 SPRUCE STREET	PHILADELPHIA	PA	19104	(610) 574-3391

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Resile Pty Ltd
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22 August 2016

Jannah Dowe
Health and Safety Coordinator
Anglo American Grasstree

RE: CHEST X-RAY PROGRAM B READER

Dear Jannah

Thank you for your enquiry for assistance in obtaining chest x-ray images and facilitating B Reader services for coal mine workers under the Coal Mine Workers Health Assessment Scheme.

Background

In the last 6 months there have been identified cases of coal mine workers with pneumoconiosis discovered in Qld.

Australia's coal mining industry has reported few new cases of pneumoconiosis for more than 20 years. Mortality from coal workers' pneumoconiosis in official health statistics and the prevalence of pneumoconiosis in x-ray surveillance programs are lower in Australia than other parts of the world.

Given that we now have miners who have been working for many decades in the underground (and other higher exposure environments) it is distinctly possible that there will be more cases of CWP reflecting past exposures.

A review of the Coal Mine Workers Health Surveillance Scheme (Review of Respiratory Component of the Coal Mine Workers' Health Scheme for the Queensland Department of Natural Resources and Mines) has been completed and the recommendations published this includes the introduction of dual reading of chest x-rays.

B Reader Service

Resile will offer a complete managed process for chest x-ray B reading.

Anglo American will:

- Provide a list of names, contact details for reports and completed consent for the facilitation of B reader services (consent form will be supplied by Resile at the commencement of the service).

  or

- Provide a list of names and contact details and Resile will co-ordinate consent forms completion

Resile will:

- Ensure appropriate consent completed for process
- Contact the radiology company to obtain the digital film
- Facilitate file transmission to the B reader service
- Arrange the second reading by a B reader contracted to Resile
- Review the B reader report
- Provide a report to the employee and employer

B Reader Services

The Health Surveillance Unit has announced that they have developed an arrangement with B Readers in United States to facilitate this. Images (DICOM) will be sent to the HSU who will facilitate the second review. There has been no information on who will bear the cost of this service. The excepted turn around for this service will be 7-10 days as a minimum.

Resile have contracted a B Reader servi8cve to assist our clients who elect to use a private service. Reports will be available 24 hours after the receipt of the image by the service.

Fees

Service		Fee (GST exc)
B Reader Facilitation	Anglo to send through list and completed consent forms. Maintain database of employees requiring B Read of chest x-ray Obtain chest x-ray image Secure transmission of image to Resile's contracted B reader service	[redacted]
	or	
	Anglo to send through employee list Resile to contact individuals and facilitated consent completion Maintain database of employees requiring B Read of chest x-ray Obtain chest x-ray image Secure transmission of image to Resile's contracted B reader service	[redacted]
	Review chest x-ray B reader reports and provide report to employee and employer	[redacted]
	Teleconsultation and or further management if abnormal chest x-ray if required	[redacted]
B Reader Services	Second reading of chest x-rays by US based B Readers.	[redacted]

Resile

Drawing on over 25 years of experience, our team led by Dr Robert McCartney and Dr Robin "Sid" O'Toole assists organisations in all aspects of occupational health.

Our multi-disciplinary approach enables us to deliver quality health risk management advice to our clients. Our Occupational Physicians form the backbone of our service and are carefully chosen to ensure their experience and expertise is matched by approachable, dynamic and flexible professionalism.

We develop client relationships based on this expertise and accessibility. Then, with collaboration, we maintain a clear understanding of the clients' work environment, systems values and goals.

Our targeted services (to many sectors including transport, services, mining, energy, aviation and government) ensure outcomes through cost-effective and timely delivery.

We are proud to work with our clients to assist employers and employees:

- Mitigate their risks through control of exposures in the workplace environment
- Make informed, balanced and evidence-based decisions regarding fitness for duty
- Minimise the impact of work-related injuries and illnesses by ensuring best-practice management

As experts at the interface of health and work our goal is to partner with our clients to maximise the health, wellbeing and productivity of their workforce.

[redacted]

Terms and Conditions

Generally, prices will maintain at the same rate for the term of the financial year, but may be subject to change with 30 days written notice;

For all clinic appointments a clinical service fee of 75% will be charged for a nonattendance or where an appointment is cancelled with less than 24 hours notice;

Cance[redacted] will apply if cancellation of a site visit occurs within 1 week of the service delivery. A fee of 50% of the proposed cost for the scheduled trip will be administered.

Your[redacted]

[redacted]

Dr Rob McCartney

Two-reader process for chest x-rays

To ensure early detection of coal workers' pneumoconiosis (CWP), the Department of Natural Resources and Mines has implemented a two-reader process — effective immediately — under the Coal Mine Workers' Health Scheme.

This is a short-term interim measure that will be in place until a Queensland-based dual screening and adjudication process is established in partnership with medical practitioners and other key stakeholders.

This process is delivered by nominated medical advisers (NMAs) or examining medical officers under the supervision of the NMA (EMOs), radiologists listed on the Register of Clinical Radiologists for CWP Screening published by RANZCR, the Health Surveillance Unit (HSU) and US-based NIOSH accredited readers.

Two-reader process

The two-reader process commences when the NMA or EMO refers the coal mine worker for a chest x-ray (CXR).

1 CXR referral → 2 First CXR reading by Australian radiologist → 3 Second CXR reading by US-based NIOSH accredited reader → 4 Health Assessment Form Section 4 completed by NMA

1. **CXR referral**
   - NMA or EMO refers coal mine worker to radiologist clinic for CXR.
   - Consent form to be completed by coal mine worker. This is a new form that provides HSU with the required consent to organise a second reading by a NIOSH accredited reader.

2. **First CXR reading by Australian radiologist**
   - CXR is read to the ILO standard by a radiologist listed on the Register of Clinical Radiologists for CWP Screening published by RANZCR.
   - Radiologist completes the ILO classification form and returns it to either the NMA or EMO, together with the report and DICOM image.
     - If the first reading determines that opacities are visible and are consistent with CWP the recommendation is the individual is also referred for a high resolution CT scan immediately.
   - NMA or EMO provides CXR result (DICOM image and report), ILO classification form and Consent form to HSU.

3. **Second CXR reading under the US-based NIOSH system**
   - HSU dispatches CXR for a second reading; reports are returned to the NMA or EMO in 7 – 10 working days.
      - If the second reading determines that opacities are visible and consistent with CWP it is recommended that the NMA or EMO refer the individual for a CT scan (except where a CT scan has previously been conducted following the first CXR review). In some cases, on the advice of an appropriate medical specialist, further testing may be required.
      - If a CT scan was ordered, the NMA or EMO considers the results and advice from appropriate medical specialists.
   - NMA or EMO share the US CXR reading with the Australian radiologist.
   - NMA or EMO completes health assessment form section 3.

4. **Health assessment form section 4 completed by NMA**
   - NMA reviews all reports.
   - When the health assessment is complete, the NMA sends:
      - completed health assessment form and all medical reports (Australian CXR report; US CXR report; spirometry report; CT scan report if applicable) to HSU (all sections of the form must be completed; incomplete forms will be returned promptly)
      - a copy of section 4 form to the coal mine worker (does not include sections 1, 2 or 3)
      - section 4 of the completed health assessment form to the employer
      - NMA retains a copy of the health assessment data and forms.

Alternative process

Companies may wish to engage with US-based experts directly to read both the first and second CXRs. This would be considered appropriate provided the CXR is read by two NIOSH accredited readers and the health assessment form is returned as per step 4 above.

New and updated forms

Please note that the HSU has introduced two new forms and updates have been made to the health assessment form to support the two-reader process.

New ILO classification form
This form is to be completed by the radiologist conducting the CXR examination. A completed form must be attached to the final health assessment form.

New Consent form
This form must be completed by the coal mine worker to give approval for the second reading to be conducted under the US-based NIOSH system.

More information
To request a form, or for general enquiries, please call (07) 3818 5424 or email cmwhs@dnrm.qld.gov.au.

GRASSTREE SHE BRIEF

Title:

Relevant to:

Date:

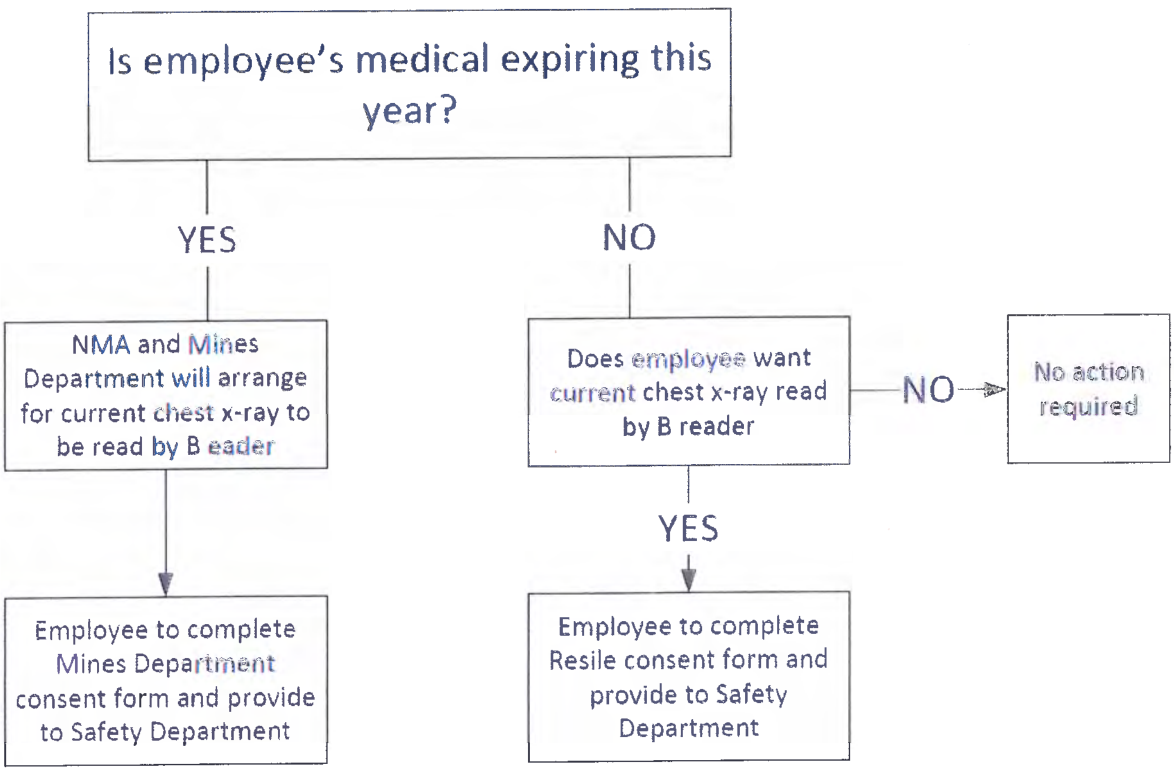
Changes to the Coal Mine Workers' Health Scheme

As an interim change the Department of Natural Resources and Mines has implemented a two-reader process under the Coal Mine Workers' Health Scheme, effective from 27 July 2016.

This means that all coal board medicals done after this date will be read to the ILO standard by both an Australian registered radiologist and also a US-based B reader.

Can I get my Recent Chest X-Ray read by a US Based B Reader?

Grasstree is offering to all employees (whose medical is expiring this year) the opportunity to have their recent chest x-ray read by an US-based B reader.



Administrators: Please remove this notice two weeks from date of issue

AngloAmerican

GRASSTREE SHE BRIEF

Title: 0461-2016
Chest X Rays – B Reader

Relevant to: All Employees

Date: 25 August 2016

NOTE: Employees who underwent subsequent investigations as a result of the findings from their recent chest x-ray (e.g. CT scan), there is no requirement for their chest x-ray to be dual read by a US-based B reader.

Composed by: Jannah Dowe
Authorised by: Clinton Barnes

Administrators: Please remove this notice two weeks from date of issue

QoN Attachment 7

Coal Workers Pneumoconiosis Inquiry
Public Hearing - Tuesday 31 January 2017 Brisbane
Questions Taken on Notice and Additional Information
Anglo Coal

Experience of NMA (PP29-30)

The table below provides a listing of the NMAs for Anglo Underground Sites

Site	Doctor	Original Date of Appointment	Has the NMA undertaken a site visit – most recent date or planned date	Approximate number of CMWHS medicals undertaken annually by that doctor
Moranbah	Dr Margaret Swenson	13/01/2016	Site visits are being planned to start again in March 2017	150
Moranbah	Dr Hengameh Jazebizadeh	28/07/2015	Site visits are being planned to start again in March. Last visit 05/06/2016	200
Moranbah	Dr Sally Trenorden	02/07/2014	Site visits are being planned to start again in March. Last visit 29/05/2016	50
Grosvenor	Dr Margaret Swenson	19/02/2016	Most recent visit February 2016. Further visit planned for Q1 2017	150
Grosvenor	Dr Francis Olopade	19/02/2016	Most recent visit February 2016. Further visit planned for Q1 2017	40
Grosvenor	Dr Sally Trenorden	19/02/2016	Most recent visit July 2016. Further visit planned for Q1 2017	50
Grosvenor	Dr Hengameh Jazebizadeh	19/02/2016	Most recent visit January 2016. Further visit planned for Q1 2017	200
Grasstree	Dr Chris Fenton	8/11/2012	Meetings with mine site staff held at medical clinic only – planned for Q2 2017	200
Grasstree	Dr Gale Ashby	18/02/2016	Most recent visit was on 14/11/15. Further visit planned for 2017	200

Additionally appended to this attachment is a sample of the appointment letter used by Anglo and signed by an NMA and stating the requirements of the NMA.

Please also note that the medical practice in Moranbah that Grosvenor and Moranbah used predominantly for NMA services was sold in January 2016 and a new owner and (some) new medical practitioners were engaged by the new owner at that time. As such, a number of 'new' NMAs for Grosvenor and Moranbah were appointed at that time.

AngloAmerican

Denise Cairns
Health Surveillance Unit
Department of Natural Resources and Mines
By email: denise.cairns@dnrm.qld.gov.au

18th February 2016

Dear Denise,

RE: Appointment of Nominated Medical Advisor

Anglo American Capcoal Management (Grasstree Mine) wishes to appoint Dr Chris Fenton to provide certain services to it as a Nominated Medical Advisor under the QLD Coal Mining Safety and Health Regulation 2001.

This agreement will commence on 18th February 2016 and will operate for a period of 24 months. Within this term, the parties may agree, in writing, to extend the term of the operation of the agreement if required. Unless terminated earlier in accordance with this agreement, the agreement will cease automatically on 17th February 2018.

Dr Chris Fenton will provide the following services to Anglo Coal Capcoal Management (Grasstree Mine) as a Nominated Medical Advisor under the Regulation s45(2):

1. Carry out health assessments in accordance with the Regulation for coal mine workers employed by or to be employed by Anglo Coal Capcoal Management (Grasstree Mine).

2. Provide to Anglo Coal Capcoal Management (Grasstree Mine) in a timely manner reports relating to health assessments carried out for coal mine workers employed by or to be employed by Anglo Coal Capcoal Management (Grasstree Mine).

3. Be available to discuss with Anglo Coal Capcoal Management (Grasstree Mine) health assessment reports relating to coal mine workers employed by or to be employed by Anglo Coal Capcoal Management (Grasstree Mine).

4. Be available to discuss and give advice about appropriate duties for a coal mine worker, subject to the discussions being held with and the advice being given to, representatives of Anglo Coal Capcoal Management (Grasstree Mine) and the relevant coal mine worker (or the worker's representative(s)).

5. Where a coal mine worker asks, discuss the worker's health assessment with another doctor nominated by the coal mine worker.

6. Carry out additional assessments for a coal mine worker if, having regard to a risk assessment carried out for a task for which the coal mine worker is to be employed, or is employed, the above mentioned doctors consider the person needs to be assessed in relation to the additional matters to achieve an acceptable level of risk.

7. Review health assessment reports for coal mine workers and carry out further health assessments having regard to health assessment reports which may be provided by other nominated medical advisors.

AngloAmerican

8. Keep records and maintain confidentiality in respect of such records in accordance with the Regulation.

9. Provide reports to nominated persons under the Regulation.

10. Otherwise comply with and fulfill the Requirements and obligations contained in the Regulation.

Additionally not stated in legislation Dr Chris Fenton agrees to the following:

1. Consider the potential health risks and exposure information provided by the site when determining the requirement for periodic or further or additional medical review of a coal mine worker.

2. Ensure when chest x-rays are required based on potential exposure risks in the workplace that all such x-ray are undertaken by registered radiological services and that it is clearly requested of the service that x-rays are reviewed, interpreted and reported against the ILO Standards for recognized diseases relevant to the potential exposure risks of the worker.

3. If reviewing, in the capacity of the mine's appointed NMA, a health assessment that has been undertaken by a doctor who is not an NMA appointed by Anglo Coal Capcoal Management (Grasstree Mine), ensure that the same conditions apply to all aspects of that assessment including x-rays and review of potential health risks and exposure information provided by the mine.

4. At least annually undertake a visit to Anglo Coal Capcoal Management (Grasstree Mine) at a time mutually agreeable to the Dr Chris Fenton and Anglo Coal Capcoal Management (Grasstree Mine).

Signed by:

....

Tim Hobson
Site Senior Executive/ General Manager
Anglo Coal Capcoal (Underground Operations)

Date of signature:

....

Dr Chris Fenton
Provider Number:
Practice Name:

Date of signature:

QoN Attachment 8

Coal Workers Pneumoconiosis Inquiry
Public Hearing - Tuesday 31 January 2017 Brisbane
<u>Questions Taken on Notice and Additional Information</u>
Anglo Coal

Additional Information 2 (P.43)

QLD Select Committee Rockhampton Meeting (12 December 2016) Transcript Responses – Dr B. Belle, Anglo American.

QLD SELECT COMMITTEE ROCKHAMPTON MEETING (12 Dec 2016) TRANSCRIPTS- RESPONSES

Dr. B Belle, Anglo American Coal

The following submission responds to requests made by the QLD Select Committee during its visit to Anglo American Grasstree mine on Dec 13th 2016. The requests were made in relation to issues reported to the Committee and captured in transcripts from recent Committee proceedings in Rockhampton (dated 12th Dec 2016). The issues raised are dealt with below under the following broad areas:

1. Respirable dust definition and $PM_{2.5}$.
2. Evolution coal dust exposure limits.
3. Opinions about no science behind current coal dust exposure limits.
4. Coal dust personal exposure monitoring leading practice.
5. PDM3700 mass based continuous monitor and PDR1000 light-scatter based technology.
6. Role of ventilation and optimal air velocity for dust management.

Conclusions are made with regards to each of the above areas. By way of a summary, these conclusions are detailed below and at the end of each section.

Conclusion 1: Respirable dust definition as in ISO (1995) with a D_{50} of 4 microns is valid and the use of $PM_{2.5}$ to describe respirable dust as per the 'Rockhampton transcripts' is incorrect in the context of determining CWP or silicosis risks. Further opportunity exists to amend the error in the AS2985 (2009) definition of respirable dust aligned to the ISO (1995) documentation.

Conclusion 2: The development of personal exposure limits for coal dust is based on scientific rigour and the opportunity exist for QLD mines to incorporate the influence of changes to the flow rate AS2985 (2009) on dust exposure limits aligned with the international sampling and harmonisation of exposure limits and assessment in Australian mines.

Conclusion 3: It is incorrect and misleading to suggest that the Australian dust exposure limits are without any scientific basis. However, opportunity exist for the QLD mines to incorporate the changes in sampling rate as in AS 2985 resulting in reduced measured dust results through a dedicated study and establish CWP statistics towards the development of Australian dose-response curve for CWP.

Conclusions 4: Without doubt, the current strategy of 'personal' sampling for exposure monitoring is valid and a leading practice for compliance determination – this cannot be achieved by static or area or other engineering dust sampling. However, using the PDM3700 mass based continuous dust monitor which is a legislated personal exposure compliance tool in the USA is a leading practice in managing the worker exposure to coal dust for its input to the medical surveillance. <u>Coal dust exposure limit is applicable only for 'personal sampling' and cannot be applied to evaluate the results from static or area or engineering sampling data.</u>

Conclusion 5: It is strongly recommended that the Select Committee recommends the adoption of the PDM 3700 mass based dust monitoring device for personal exposure monitoring as well as compliance determination purposes (as legislated in the USA (MSHA)) as a replacement for the traditional gravimetric dust monitors. Its approval for use in Australian mines as a compliance tool must be treated as a priority to benefit coal mine workers. In addition, it is strongly recommended that the error in the newly published (Jan 2017) Recognised Standard 14: Monitoring respirable dust in coal mines" that incorrectly notes that the results of TEOM monitor data are "indicative only" to be rectified and rather be applied as a leading practice 'compliance monitor' for effective dust exposure management.

Conclusion 6: The mine ventilation plays a significant role in managing the multiple hazards in underground workings in addition to dust management. The magnitude of airflow rate in typical longwalls <u>should not to be prescriptive</u> to underground operations without due considerations to the operational parameters and acknowledging the management other multiple fatality hazards present in the gassy and spontaneous combustion prone coal mines leading to multiple fatality risks.

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1. Respirable dust definition and $PM_{2.5}$

Respirable dust sampling is pivotal in estimating the 'dose' of individual coal worker exposure to dust and in deriving quantitative respiratory disease risks in epidemiological studies. Based on the past epidemiological knowledge (Orenstein, 1960), it has been established that the respirable dust particle size distribution is critical due to its potential health effects and quantifying the risks. Respirable dust refers to particles that settle deep within the lungs that are not ejected by exhaling, coughing, or expulsion by mucus. Since these particles are not captured with 100% efficiency on way to the deepest part of the lungs, respirable dust is defined in terms of size-selective sampling efficiency curves. This had led to internationally recognised respirable size-selective sampling widely known as the British Medical Research Council (BMRC) definition of the respirable dust fraction or Johannesburg curve with a median aerodynamic diameter of 5 μm collected with a 50 % efficiency (D_{50}).

These size-selective curves are actually lung penetration rates of dust particles that gravimetric dust sampling instruments attempt to replicate. Some of the new scientific evidence concerning the hazard from very small particles argued that it may not be appropriate to ignore a specific effect of these on worker's health. Therefore, in 1995, international standards organization (ISO) had recommended that the definition of respirable dust follow the convention described by Soderholm (1989, 1991) with a D_{50} of 4 μm. An international collaboration (ACGIH, 1985, ACGIH 1999, ISO 1995, CEN, 1993), for sampling harmonisation has led to the agreement on the definitions of health-related aerosol fractions in the workplace, defined as inhalable, thoracic and respirable curve (Figure 1). The new respirable size-selective curve is different from previous definitions used in the United States, South Africa, Australia and Europe and truly represents an international harmonization of the definition of respirable dust. This sampling harmonisation is aimed to eliminate confusion related to differences in dust exposure levels, exposure assessment and compliance determination methodologies. Since Y2004, Australia has adopted the ISO (1995) curve from the old size-selective curve (AS2985, 1987), similar to South Africa and USA that have adopted the harmonization curve in Y1998 and Y2016 respectively. Table 1 summarises the BMRC and ISO size-selective curves for dust sampling in mines (NIOSH, 1995; ISO1995).

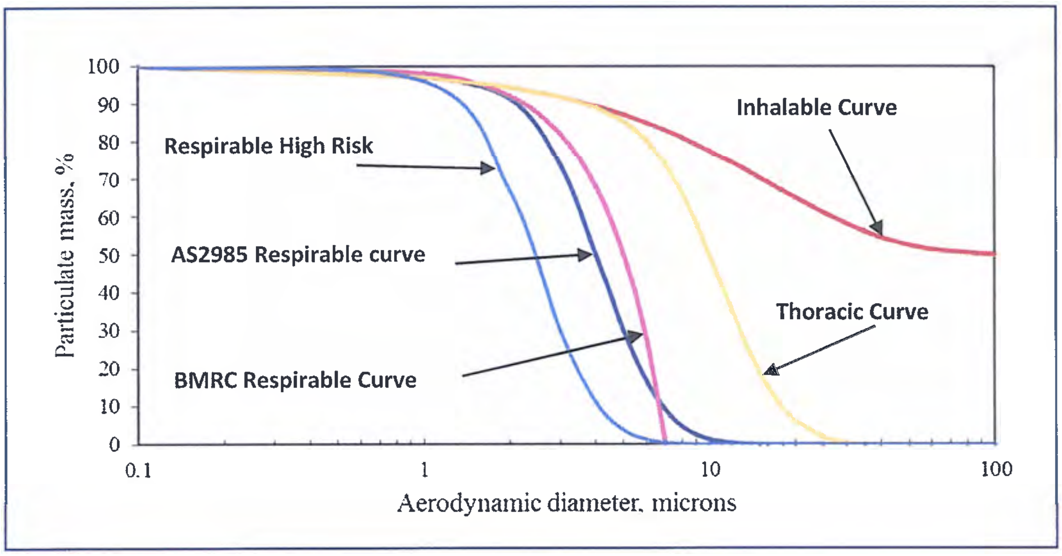

Figure 1: Respirable dust size-selective sampling curves including $PM_{2.5}$

Table 1. Collection efficiencies for size-selective sampling.

BMRC Curve		ISO Curve*	
Particle size, µm	% greater than stated size	Particle size, µm	% greater than stated size
0	100	0	100
1	98	1	97
2	92	2	91
3	82	3	74
4	68	4	50
5	50	5	30
6	28	6	17
7	0	7	9
		8	5
		10	1

*NIOSH criteria document (page 72); ISO 7708(1995)

Importantly, AS2985 (2004, 2009) defined the respirable dust using the inhalable convention with a "56%" efficiency at 4 µm (Table 1 of AS2985-2009, page 7) using the cyclones at 2.2 l/min flow rate. However, ISO7708 document states that the D_{50} of the size-selective curve to be 4 µm as the 'total airborne particles' convention. What is noteworthy here is that the AS2985 uses the 'inhalable convention [Table B.1-ISO7708]' against the globally accepted 'the total airborne particles convention [Table B.2-ISO7708]' in defining respirable convention. Conforming to the international definition consistently would therefore, removes the ambiguity of D_{50} value of 4.25 µm against the NIOSH/ACGIH standard of 4.0 µm when deciding upon the cyclones. It is worthy to note that to protect the target population of children, or the sick or inform (the "high risk" group), the corresponding D_{50} value is 2.41 µm or commonly termed as $PM_{2.5}$.

Conclusion 1: Respirable dust definition as in ISO (1995) with a D_{50} of 4 microns is valid and the use of $PM_{2.5}$ to describe respirable dust as per the 'Rockhampton transcripts' is incorrect in the context of determining CWP or silicosis risks. Further opportunity exists to amend the error in the AS2985 (2009) definition of respirable dust aligned to the ISO (1995) documentation.

2. Background and Evolution of Dust Occupational exposure limits (OELs).

OELs were first proposed by Emhurst Duckering (1910) in the UK, as a way of limiting exposure to dust, as "...The most scientific way of regulating a dusty trade would be to impose a limit on the amount of dust which may be allowed to contaminate the air breathed by the workpeople and to leave the manufacturer a completely free choice of methods by which this result may be attained..." Various countries around the world use different 'terms' to express the occupational exposure limits (OELs), including the use of 'safe limits' in the transcripts. For example, the term OELs as used by the Occupational Safety and Health Administration (OSHA) of USA or as in Australia or South Africa; Permissible Exposure Levels (PELs) as used by the Mine Safety and Health Administration (MSHA); TLVs as used by the American Conference for Governmental Industrial Hygienists (ACGIH).

To eliminate the health risk of exposure to substances, the coal dust exposure limits were developed, applied and promulgated through health studies originally carried out in the UK, the USA, South Africa and other European countries by subject matter experts. Of these exposure limits, TLVs being the most famous and influential guideline. TLVs is an abbreviation of Threshold Limit Values for hazardous substances which are developed and updated by the US based non-profit scientific association, called American Conference for Governmental industrial Hygienists (ACGIH) established in 1938. TLVs are based primarily on health considerations refer to airborne concentrations of substances and represent conditions to which it is believed that "nearly all workers" may be repeatedly exposed day after day without adverse health effects. TLV is a copyrighted trademark of the ACGIH and TLVs are not mandatory exposure standards but recommended to be used as a guideline. These limits are updated annually and reflect generally the current professional recommendations on workers' exposures to specific substances. Every TLV® is developed and based on the available, relevant, scientific data for that substance. It is possible that sometimes, the TLVs of few substances can be retracted based on new evidence (example, Diesel Particulate Matter [DPM]).

In the light of the past litigations against the ACGIH, the scientific body cautions that the TLVs are an expression of scientific opinion and are not consensus standards. Furthermore, the published TLVs are based solely on health factors and no consideration has been given to economic or technical feasibility. Some 13 countries (excluding USA), have adopted the TLVs of most substances as regulatory standards. The use of exposure limits and the monitoring methods and tools are explained in an another document titled "Misuse of Threshold Limit Values (TLVs) in Scientific Papers, Reports and Discussions-A Note" attached in the Appendix.

Conclusions 2: The development of personal exposure limits for coal dust is based on scientific rigour and the opportunity exist for QLD mines to incorporate the influence of changes to the flow rate AS2985 (2009) on dust exposure limits aligned with the international sampling and harmonisation of exposure limits and assessment in Australian mines.

3. Science behind current coal dust limits.

As indicated before, evolution of exposure limits published originally through ACGIH TLVs are based solely on health factors and no consideration has been given to economic or technical feasibility. In the case of Australia, there is no CWP dose-response curve that exists to understand the health risks for different exposure. On the other hand, the US exposure limit of 2 mg/m^3 was based on the medical surveillance and exposure data that was available originally from the UK research work (Jacobsen et al., 1970, 1971; Jacobsen, 1972). Following table summarizes the current coal dust exposure limits (Belle, 2004) used in various coal mining countries globally.

Table 2: Summary of coal dust OELs (Belle, 2004)

Country	Coal dust limit, mg/m^3
Australia	3.0/2.5 (Qld/NSW)
Belgium	10 / (% quartz + 2)
Brazil	8 / (% quartz + 2)
Finland	2.0
Germany	4.0
India	3.0
Italy	3.33
Netherlands	2.0
South Africa	2.0
UK*	4.0
USA	2.0 (Feb 1969) to 1.5 (Aug 2016)

Yugoslavia	4.0

**There are misleading references to UK coal dust personal exposure limit of 2 mg/m³. However, Section 249 of the Mines Regulations 2014 (HSE), the mine operators should aim to control exposure to respirable dust and RCS in coal mines to below 3 mg/m³ and 0.3 mg/m³, respectively, averaged over a 40-hour working week (referred to in the Regulations as the 'action levels'), and should take action if exposure exceeds these levels. The COSHH definition of a substance hazardous to health includes dust of any kind when present at a concentration in air equal to or greater than 10 mg.m-3 8-hour TWA of inhalable dust or 4 mg.m-3 8-hour TWA of respirable dust. It is also valuable to note that unlike in USA or South Africa, Australia carries out the inhalable dust sampling and its benefit and intended use in epidemiological study is worth clarifying.*

However, it is very important to note that the dust exposure results and limits are not to be compared without having a thorough understanding of the following elements, viz., dust monitors and their operation standard, dust type, sampling strategy, exposure assessment, compliance determination process and history and prevalence rates.

Although, USA coal dust limits have reduced from 2 mg/m³ to 1.5 mg/m³, the key measurement differences attributed to the reduction other than the dust control and greater understanding of CWP dose-response is due to following:

- USA adopting to the international harmonistaion respirable sampling curve with a D_{50} of 4 microns
- Use of the new size-selective curve and the sampling flow rate of 2.2 Lpm in coal mines.

On the other hand, Australia adopted the ISO (1995) respirable size-selective convention in 2003. This change meant that the sampling flow rate of the device used as in AS 2985 required to be operated at 2.2 Lpm instead of 1.9 Lpm. In addition, it would also mean that the sampling bias due to change in flow rate would result in reduced measured dust levels. Therefore, the NSW coal dust standard (NSW Government Gazette # 200) is set at 2.5 mg/m³ instead of the original 3.0 mg/m³. Similarly, silica dust limit was reduced to 0.12 mg/m³. However, one could not obtain the original dataset to verify the basis of NSW limit. On the other hand, other work in South Africa using the switch over to new size-selective curve would result in measured dust levels by 12% (Belle, 2003).

Conclusions 3: Therefore, it is incorrect and misleading to suggest that the Australian dust exposure limits are without any scientific basis. However, opportunity exist for the QLD mines to incorporate the changes in sampling rate as in AS 2985 resulting in reduced measured dust results through a dedicated study and establish CWP statistics towards the development of Australian dose-response curve for CWP. In addition, unlike in the USA or South Africa, the requirement of inhalable coal dust sampling in Australia and its benefit and intended use in epidemiological study is also worth clarifying.

4. Validity of current personal exposure monitoring as a leading practice:

Personal exposure monitoring is quintessential to medical surveillance and in the development of dose-response curves. Exposure monitoring is valuable to determine the validity of current safe dust exposure limits and developing future limits.

The assertions made in the 'transcripts' that personal exposure monitoring is invalid. What is acknowledged is that the engineering or static sampling have been used over 60 years to understand the efficiency of dust control system by the industry using various light-scatter based tools in a dynamic environment. Unlike USA and South Africa, Australia also measures inhalable dust and its use in epidemiological studies is yet to be known.

In this context, it is appropriate to provide several definitions relating to dust sampling techniques that is used in the mining industry. These definitions are provided below:

- **Personal Sampling:** Is a method of sample collection whereby the dust sample is taken from the breathing zone of a mine worker while performing occupational duties during a work shift. In this sampling method, the worker wears the sampling train (cyclone, pump, tube, sample filter) for the entire work shift. Personal sampling results are most commonly used as the exposure or dose element in the development of dose-response relationships;
- **Static or Area or Environmental Sampling:** Is a method of sample collection whereby the dust sample taken at a fixed location at the workplace in an environment or area of interest that is not mobile. The dust sample reflects the average concentration in the area of interest and does not reflect the exposure of any worker in that area;
- **Occupational Sampling:** An occupational sample is the dust sample taken during a work shift on individual workers who perform duties in a designated occupation - this terminology is used in US coal mines. This method of sampling measures the dust exposure for defined occupations as if one person performed the duties in that occupation for the whole working shift; and
- **Engineering Sampling**: An engineering sample is a dust sample taken at the Continuous Mining (CM) machine that is positioned at a consistent location when comparing the dust control systems. An engineering sample is the dust sample taken to characterize the emission source or suppression effectiveness of ventilation and dust control measures. The engineering sampler is switched on at the face area at the beginning of the shift while the cutting machine is standing and is switched off before leaving the face area at the end of the shift. It aims at evaluating both the management (administrative effectiveness) of the dust control system as well as effectiveness of the dust control system (engineering). The engineering sample is collected only while the engineering activity is taking place, i.e., duration of cutting operation.

The appropriateness of the current gravimetric personal exposure monitoring

Over the last two decades, changes have taken place in the domain of personal respirable dust exposure monitoring. Exposure monitoring and assessment is a complex system that requires clear understanding of the coal mining operation, monitoring practices, engineering controls, ventilation systems and dust generation dynamics. It is therefore increasingly necessary to measure the dust levels as accurately as practicable to assess the exposure, by using effective and practical sampling techniques. Prior to 1998, respirable dust samplers in all countries were operated at a flow rate of 1.9 L/min in agreement with the British Medical Research Council respirable convention (BMRC, 1952). Currently, personal respirable dust samples are collected in accordance with the new ISO/CEN/ACGIH respirable dust curve with a 50% cut point (d50) of 4 µm, which is global leading practice.

Dust sampling and exposure assessment is an element in the pathway to eliminate dust related lung-diseases. As required by regulations and directives, if properly carried out, exposure assessment can result in significant benefits to the mining industry. Various global dust studies have indicated that personal sampling provides the best estimate of worker exposures and of the temporal and spatial variability in those exposures for use in dose-response models. Leidel et al. (1977) recommended that, for accurate assessments, the personal exposure measurements must be taken within the worker's breathing zone. The inaccuracy incurred in using area sampling for measuring dust exposure of mining machine operators in US coal mines is well documented by Kissell and Sacks (2002) and Belle (2016). They recommended that the worker exposure is best assessed using 'personal sampling' rather than 'area or engineering sampling' techniques.

The following global dust studies reflect the extent of and the reasons identified for using personal sampling as the best methodology for personal exposure assessment:

- A comparative study of personal and fixed-point (area) samplers by Breslin, Page and Jankowski (1983) reported the coefficient of variation of measured mine dust concentration to be typically less than 20%;
- Listak et al (1999) concluded that there was little predictive correlation between fixed-location area samples on CMs to operator breathing zone samples. This US study noted that if the fixed-point dust level was 1.5 mg/m3, then the 95% confidence level predicted operator dust levels at the boom hinge point and in the operator breathing zone exposure could vary from zero to 2.6 mg/m3;
- Divers et al. (1982) conducted a three-shift dust study in a US coal mine operated using remote control machines. Their study showed that the mean ratio of respirable dust samples taken at the cab compared with the remote control operator location was 30.7 (i.e. the static sample result was 30.7 times greater than the personal sample);
- Kissell and Sacks (2002) have shown that a wide variation in dust levels between samplers located within a few feet (less than about 1.5m) of each other, i.e., fixed sampler was within 18 inches (45cm) to 30 inches (75cm) from the machine operator; and
- Belle (1998) has made similar observations when the engineering samplers and real-time monitors were positioned between the front two poles of the CM operator cabin. Despite the dust samplers located approximately 60 cm from each other, the dust cloud monitored by the instruments was different. Such variability in measured peak dust levels shows the complexity and validity of any conclusions that may be drawn from static or area sampling. In this specific example, the engineering sample dust level was 4.75 mg/m3. If the CM was to be operated under remote control, the CM operator would be standing in the fresh intake air with a measured dust level of 0.29 mg/m3. This clearly illustrates that a coal worker can be exposed to different dust concentration clouds and refutes the view that the dust exposure level within even a small area is fixed. This illustration gives an idea of the complex nature of sampling, analysis and interpretation of the dust concentration values obtained in the field for exposure assessment.

Table 3 below shows the results of four published US coal mine studies and the South African study, reinforcing the view that the fixed-location area samples cannot predict the personal dust exposure of CM operator (i.e., failing the NIOSH 25% accuracy criterion).

Table 3: Summary of mean ratio of fixed and personal sample (Source, Kissell and Sacks, 2010)

Published Study	No. of Mines	Mean ratio of Static or Area /Personal sample	Relative Standard Deviation
Kost and Saltsman, 1977	6	3.53	0.81
Divers et al. 1982	1	30.7	0.21
Kissell and Jankowski, 1993	5	4.15	0.45
Listak et al., 1999	5	3.07	0.59
Belle, 2016	8	7.19	2.41

Put simply, the results of the static or area sample measurement were very significantly greater (up to 30.7 times greater) than the personal sampling results. Hence, the static or area sample results do not provide an indication of personal dust exposure.

The position of the dust samplers used in the engineering area sampling is crucial in determining the effectiveness of dust control systems in longwall face or a CM face. This can easily be illustrated by positioning a sampler at different locations, e.g. sample location on the mobile shearer, various longwall shield locations, location of closer to the flight conveyor, CM cutter head, etc. Comparisons between dust sampling results therefore require consistent positioning of the samplers. Typically, engineering samples are used to identify failures of engineering controls and such sampling is not a common practice elsewhere in the world for routine and regulated sample collection requirements.

All of these sampling methodologies require the use of the monitors as prescribed in the statutory gravimetric sampling technique (AS2985) and it is merely location of the samplers to assess the dust loads, regardless of the worker position or personal exposure as there is no meaningful relationship that exist between engineering sampling and personal sampling. The global research studies have demonstrated that the fixed-location engineering sample (or any other similar sampling methodology with a different name), are unsuitable and cannot predict the personal shift dust exposure. The engineering samples can be used for evaluation purposes and should be measured at a consistent location at all times for comparison purposes. It would be beneficial rather to measure personal exposures that provides superior information towards exposure management of dust as well as 'dose' element in the medical surveillance results.

It is misleading to suggest that the current personal sampling and assessment practices are invalid as the dust sources and associated controls in various similar exposure groups (SEGs) are reflective of the personal exposures of workers. In fact, the new mass based PDM3700 real-time dust monitoring devices can be more effective to any current gravimetric sampling shortcomings for exposure management. The performance of ventilation and dust control system in a workplace can also be achieved by current operating alternatives such as on-board water and pressure flow monitoring devices, section intake and return real-time air velocity monitors in addition regulatory manual check-lists, change of blunt picks, start-up shift inspections, standard operating procedures.

Conclusions 4: Without doubt, the current strategy of 'personal' sampling for exposure monitoring is valid and a leading practice for compliance determination – this cannot be achieved by static or area or other engineering dust sampling. However, using the PDM3700 mass based continuous dust monitor which is a legislated compliance tool in the USA is a leading practice in managing the worker exposure to coal dust for its input to the medical surveillance. <u>Coal dust exposure limit is applicable only for 'personal sampling' and cannot be applied to evaluate the results from static or area or engineering sampling data.</u>

5. **PDM3700 (mass based) continuous monitor and PDR1000 (light-scatter based) continuous monitor**

Further to the comment on the reliability and its use for personal exposure monitoring as well compliance determination, following paragraphs provide the background and discussion (Belle, 2016, Appendix-A). Currently, in the realm of real-time dust monitoring, there are two technologies available:

1. Light scatter based dust monitor (developed since 1980s, e.g., PDR1000)
2. Mass based PDM3700 continuous personal dust monitor used as a legislative tool by the US regulator MSHA.

PDM3700 is a trade name of the mass based real-time continuous personal dust monitor (CPDM) that is used by the regulator (MSHA) in US coal mines. In the United States, MSHA's landmark respirable coal dust rule was promulgated on August 1 2014 resulting in reduced personal occupational exposure limit (OEL) for coal dust. On August 1, 2016, the overall respirable dust standard in coal mines was effectively reduced from the historic 2.0 to 1.5 mg/m^3 of air (MSHA, 2016). In addition, on February 1, 2016, the US mine regulator (MSHA), required US coal mine operators to use mass based continuous personal dust monitors (CPDMs) to assess worker occupational exposure to coal mine dust in underground coal mines. It is envisaged that the reporting of dust levels in real time will empower miners and operators to take immediate action in avoiding exposure to excessive airborne dust levels. The implementation of the CPDMs is through the use of the MSHA approved PDM3700 continuous mass based dust monitor using the Tapered Element Oscillating Microbalance (TEOM) principle.

The NIOSH developed mass based PDM3700 is well proven personal – real-time dust compliance monitoring tool that is currently used by the MSHA for compliance monitoring and has been introduced in Australian Anglo American coal mines in Y2016. The PDM3700 monitor not only provides an indication of the engineering control performance but can provide a valid regulatory personal dust sample for regulatory dust data submissions in various homogenous exposure groups (HEGs).

In contrast with the TEOM principle, passive light scattered real-time devices have been in use since 1980s to evaluate the effectiveness of ventilation and dust control systems as recorded in various USBM and MSHA dust studies (Williams and Timko, 1984; Page and Jankowski, 1984; Gero and Tomb, 1988). Historically, sources of variations in measured dust levels detected when using real-time monitors have been rationalized for parameters such as dust types, dust levels, monitor orientation, particle size, air velocity, and sensor contamination. It is often noted in these comparative studies that one of the major sources of variations in measured dust levels by the dust monitors could be the size distribution of the parent dust (Soderholm, 1989, Volkwein, 2002).

The conclusions from the above studies are similar in that the use of a real-time monitor as a stand-alone unit is not recommended for personal exposure assessment purposes but rather, more appropriately, for the identification of dust trends during a working shift. The most common sources of variability in the real-time monitoring can be attributed to dust levels, dust type, dust size, air velocity, monitor orientation and contamination of op-tics.

Conclusions from the research studies indicate that the reliability of stand-alone passive direct-reading light-scattering is inadequate due to their inherent sensitivity to airborne particulate matter other than dust. Despite this, their use is continuing. This is purely because there is no other alternative instrument that incorporates the traditional feature of mass-based continuous dust monitoring for the management of airborne dust in mines.

Conclusion 5: It is strongly recommended that the Select Committee recommends the adoption of the PDM 3700 mass based dust monitoring device for personal exposure monitoring as well as compliance determination purposes (as legislated in the USA (MSHA)) as a replacement for the traditional gravimetric dust monitors. Its approval for use in Australian mines as a compliance tool must be treated as a priority to benefit coal mine workers. In addition, it is strongly recommended that the error in the newly published Recognised Standard 14: Monitoring respirable dust in coal mines" that incorrectly notes that the results of TEOM monitor data are "indicative only" to be rectified and rather be applied as a leading practice 'dust compliance monitor' for effective exposure management.

6. Role of mine ventilation and longwall airflows in multiple hazard management including dust.

Mining hazards resulting from natural and mining conditions are generally managed by adequate mine ventilation that utilises air velocity as a fundamental and quintessential design parameter. Air velocity (or, more correctly, air speed- a scalar value) expressed in meters per second (m/s) indicates how rapidly the general body of an air current flows through a mine excavation (airway). Critical aspects that are considered in the design and planning of mine ventilation networks are air velocities and their localised direction in the working face, intake, return, tailgate, conveyor road, intake shafts, return shafts, main drifts, travel roads, haulage roads, longwall faces and stoping panels, last through roads (LTRs), overcasts, bleeder roads and air regulators. In turn, consideration of the excavation's cross sectional area (m^2) yields the air flow rate (m^3/s) through it and is also used in calculating the pressure differentials and hence the efficiency of mine ventilation systems.

There are two 'number values' to longwall airflows risen out of the CWP enquiries, as below:

1. *Ventilation optimisation: Whilst sufficient ventilation is essential for dust (and gas) dilution, too much of ventilation may promote the pickup of dust, dry-up exposed (fine) coals quickly and exaggerate dust contaminations. A ventilation volume* ***no more than 45 m^3/s*** *is recommended subject to gas emission levels and climate control (Dr. T Ren submission, Page 5, April 2016).*
2. *"...Too low velocity or too low a volume of air coming into the longwall face also has an adverse effect for the simple fact that it does not remove the dust. There is a sweet spot, which is usually anywhere* ***between 40 cubic metres per second and 60 cubic metres per second*** *depending on the size of the longwall—the height and the width and the length of the longwall—to actually dilute the gas and to remove the dust from the face..." (Rockhampton Transcripts, Page 12, Dec 2016)*

In order to understand the derivation of airflow at workings, following paragraphs provide a high level background. The methods considered to determine the minimum ventilation requirements are based on air velocities to meet the needs of various operational requirements, viz.:

1. to dilute dust or gas or other identified hazards encountered under normal mining operations
2. to dilute (and manage) dust, gases and particulates emitted by diesel engines to manage DPM and gases.
3. Provide adequate heat sink capacity for heat management
4. Provide a safe and viable blast re-entry time

The Australian longwall mining ventilation system is complex and dynamic that is able to provide the minimum airflow to extreme lengths such as longwall start-up to manage the gas, explosion and spontaneous combustion risks. In this regard, it is noteworthy that dust is one of many hazards in underground coal mines that require highly specialised, technical and complex management strategies to enable safe and successful coal mining activities. Other hazards inherent to underground coal mining activities include various toxic and flammable gases, geo-thermal heat, propensity for spontaneous combustion, diesel particulate matter (DPM), irrespirable atmospheres and re-entry after blasting activities, if applicable.

Given the typical high seam (4.0 meter) longwall operations experienced at Moranbah or Grosvenor and the low to medium seam heights experienced at Grasstree mines (approx. 3.0 meter), the longwall operation air velocities vary between 3.2 and 4.0m/s resulting in 75m^3/s and 50 m^3/s of air across these longwall faces respectively. These air velocity values suggest they are up to 10 times higher than the legal requirement of *'a ventilation current of an average velocity of at least 0.3m/s ....'* (Coal Mining Safety and Health Regulation 2001, p.219[1]) for a working face to manage various hazards. Therefore, the role of airflows supplied to each longwalls are not only to control the dust related hazards, but to manage the multiple fatality hazards such as frictional ignitions, sponcom fires and explosions. Any imbalance or prescriptive airflows differing to the operational practices may create additional hazards such as hazardous situation by oxygen egress into the goaf resulting in the risk of spontaneous

[1] From – Queensland Coal Mining Safety and Health Act 1999, Coal Mining Safety and Health Regulation 2001.

combustion of coal and undesirable increases in tailgate gas levels creating an explosive goaf gas fringe in the longwall tailgate.

Legislated Air Velocities

The following paragraph summarise air velocity requirements in the ventilation codes of practice (COP) and legislation of a number of countries (Belle, 2013). These requirements illustrate that manual and or electronic means of real-time velocity monitoring devices would enable to provide assurance needed on meeting those compliance requirements.

- The Queensland mine safety legislation requires that the Principal Hazard Management Pan (PHMP) must ensure that the ventilating air provided for the mine is of sufficient volume, velocity and quality to remove atmospheric contaminants from mining operations and maintain a healthy atmosphere at the mine during working hours. Also, it must ensure that the effective working temperature requirements are met. Effective temperatures are determined using measured wet bulb and dry bulb temperatures and air velocity. (Coal Mining Safety and Health Regulation 2001, Regulation 343-345)
- Also, in terms of the same legislation, controlled ventilation for a working place in each standing working place that is on the intake side of a working place and in each working place in an ERZ1 (Explosion Risk Zone 1) must provide for a ventilation current of an average velocity of at least 0.3 m/s measured across the cross-sectional area of the roadway in the working place. (Coal Mining Safety and Health Regulation 2001, Regulation 343-345)
- In addition, Safe Work Australia mine safety legislation requires that in areas of the mine where persons work and travel, the ventilation system provides an average air velocity of at least 0.3 m/s measured across the work or travel area (Model Work Health and Safety (Mines) Regulations 2011 Section 649).
- The prescribed Chinese ventilation regulations stipulate minimum ventilation volume per person (4 m3/min/person); decline travel airway velocity limit of 8.0 m/s; and, depending on location or activity, a minimum ventilation velocity of 0.25-0.50 m/sec aimed to attain a minimum diesel emission dilution factor of 0.06 m3/s/kW.
- US regulation 30 CFR 75.350(b) limits belt air velocity to 5.08 m/s; 30 CFR 75.327(b) limits air velocity in trolley haulage entries to 1.27 m/s provided the methane content can be maintained below 1%.
- In South Africa, with the change of legislation from the Minerals Act to the Mine Health and Safety Act in 1996, the prescribed minimum working face air velocity of 0.25 m/s and air quantity of 0.15m3/s/m2 of development heading face was removed and replaced with a risk-based process that ensures the mine operator would perform a risk assessment to determine the minimum air velocities and quantities that would be required to ensure that hazards and pollutants are controlled.
- In a very similar way, Ontario legislation does not stipulate any air velocity requirements (minima nor maxima) but hinges air requirements on the attainment of adequate and

stipulated time-weighted exposures for carbon monoxide, radon daughters and diesel particulate matter.

- Polish regulations (§ 190. 1) suggests that air velocity in areas with methane presence, except chambers, cannot be lower than 0.3 m/s, but if there is electricity cable it cannot be lower than 1 m/s. With the application of stoppings, air velocity can be lower if gas concentrations are correct. In addition, air velocity cannot exceed 5 m/s for mine workings (e.g. longwall), 8 m/s for transport drifts (but maximum of 10 m/s) and 12 m/s in intake shafts with cages (Wrona, 2013).

Maximum Air Velocity – Dust Dispersion

Amongst various air velocity design factors in underground ventilation design, another commonly quoted design air velocity is 4.0 m/s in conveyor road, face areas and intake airways. The paragraph below provides clarification to the origin to the design standard. Reinhardt (1972) showed that over a range of air velocities between 0.3 to 2.6 m/s, approximately 40 % of the coarse dust would settle out of the air within the first 30 m of the return airway and that 70 to 90 % would have settled out within 300 m of the face. For finer dust particles the values for were 0 to 20 %, and 35 to 60 % respectively. Settling and entrainment of coarse dust (greater than 10 microns) is highly dependent on velocity. At lower velocities it settles out of the air readily but on the other hand, it is more readily entrained at high velocities. The NCB report (1978) quotes a Polish study (Gruszka et al) that provided the impact of air velocity on dust levels for various dust fractions (Figure 2). On the one hand increase in air velocity reduces the dust level by increased dilution. On the other hand, increased air velocity may result in greater dust pick up by the air stream.

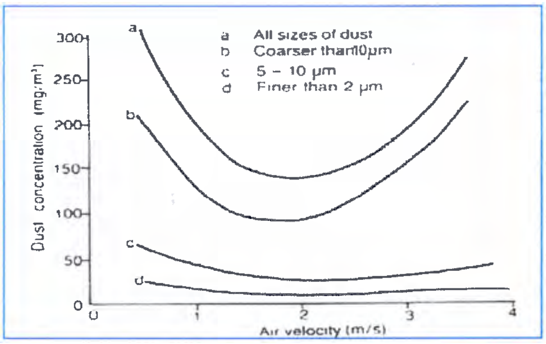

Figure 2: Relationship between dust concentration and air velocity for different particle sizes after Gruszka et al (in MRDE, 1980).

Another reference in relation to air velocity with respect to the dust was found in a UK recirculation study (MRDE, 1980) that recommends future research for ascertaining the relationship between air velocity and dust levels. The MRDE study noted that control of respirable dust levels can be achieved with face air speeds up to 3.5 to 4.0 m/s. Furthermore the study noted that the coarse dust pick up is known to be more susceptible to air velocity than is fine dust. The study concluded that air velocities in excess of about 2.5 m/s would result in increases of coarse dust concentration even with efficient filtration.

In overall, a research study by Ford (1976) suggested a recommended face air velocity below 4.0 m/s because coarse dust becomes intolerable to workmen at this velocity. The basis for this value is to manage the physical discomfort caused by large dust particles (Figure 3) striking the skin later suggested by McPherson (1984).

Currently, the question relating to the appropriateness of the 4.0 to 6.0 m/s limit for conveyor roads or intake roads is less debated or questioned. The question is possibly around which criterion, or number of these, determine the final selection: is it based on recent empirical data or studies relating to increased daily production rates or speed of conveyor belts or the likelihood of workers walking through that particular travel road on a regular basis? Furthermore, one cannot find any evidence of the data source that justified the 4.0 m/s velocity limit based on airborne dust considerations. What was missing in most of the design expert review documents (leading or misleading) was the rationale behind these air velocities and reference to 'no go' values.

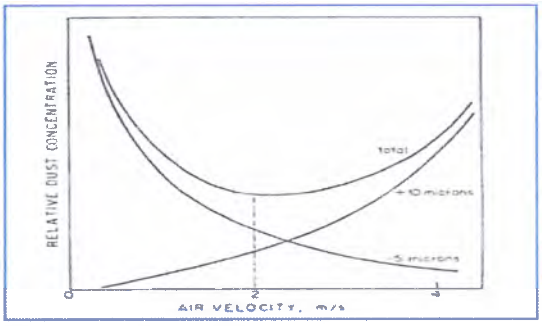

Figure 3: Source of intake velocity dust limit (McPherson, 1984).

Regardless of above, our experience in Qld mines suggest that most of the air velocities based on the mine specific longwall return airways suggest that they are less than 4 m/s. Figure 4 provides the longwall face velocity contours of measured longwall face air velocity data, viz., Chock 15 (top Left), Chock 75 (top middle), Chock 115 with shearer present (bottom Left) and Chock 135 (bottom right).

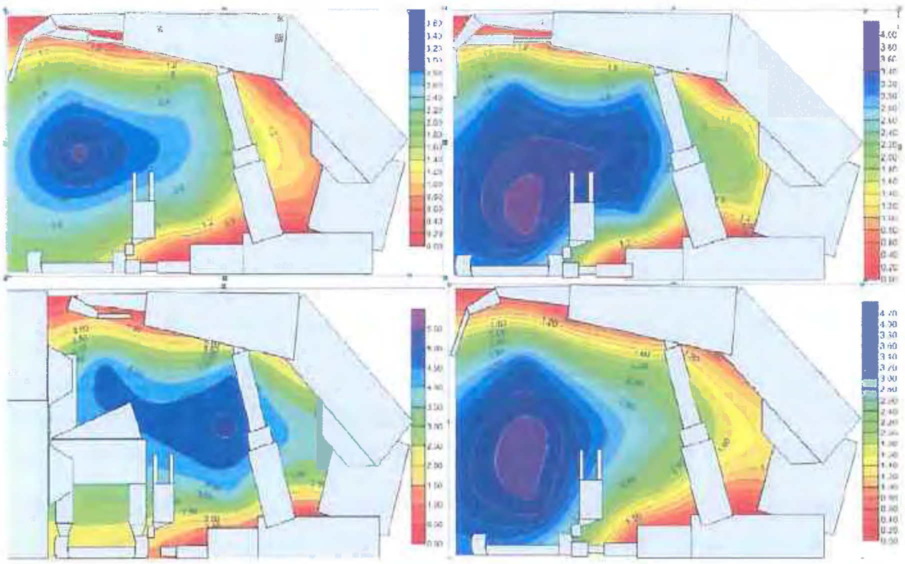

Figure 4: Isovels along the longwall face maingate (top left), mid gate (top right), shearer (bottom left) and tailgate (bottom right) locations (Belle, 2013).

As seen from these profiles based on air velocity measurements, these velocity contours can provide both a visual depiction of the air flow pattern and also a means of quantifying airflow. These profiles are useful to understand the possible location or presence of gas as well as possible scenarios for ventilation to leak into the goaf. In the context of this enquiry, 'prescribed air velocities or airflows" in a longwall face is a challenging and restrictive proposal to coal mining operations as the location of the measurement is complex and impractical to achieve, let alone its compliance determination. It is also important to note that the respirable dust is airborne and any additional airflow also dilutes the dust concentration, other than its influence of deposited dust being re-entrained.

Conclusion 6: The mine ventilation plays a significant role in managing the multiple hazards in underground workings in addition to dust management. The magnitude of airflow rate in typical longwalls <u>should not to be prescriptive</u> to underground operations without due considerations to the operational parameters and acknowledging the management other multiple fatality hazards present in the gassy and spontaneous combustion prone coal mines leading to multiple fatality risks.

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QLD SELECT COMMITTEE ROCKHAMPTON MEETING TRANSCRIPTS- RESPONSES

Dr. B Belle, Anglo American Coal

The following attachments are to be read in conjunction with the response submissions to the request made by the QLD Select Committee during their visit to Anglo American Grasstree mine on Dec 13th 2016 on Rock Hampton transcripts in relation to the following broad areas:

1. Respirable dust definition and $PM_{2.5}$.
2. Evolution exposure limits
3. Science behind current coal dust exposure limits.
4. Exposure monitoring leading practice.
5. PDM3700 mass based continuous monitor and PDR1000 light-scatter based technology.
6. Role of ventilation and optimal air velocity for dust management.

Appendix-A

1. Belle, B., 2004, International harmonisation Sampling Curve (ISO/CEN/ACGIH): Background and its influence on dust measurement and exposure assessment in the South African mining industry, Journal of the Mine Ventilation Society of South Africa, April/June 2004, pp55-58.
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TECHNICAL PAPER

International Harmonisation Sampling Curve (ISO/CEN/ACGIH):

Background and its influence on dust measurement and exposure assessment in the South African mining industry

By B. K. Belle, South Africa

Abstract

In the late last century there was a call for global harmonisation of size-selective respirable sampling of dust at workplaces. The impact of such switchover has not been widely publicised or had few investigations. The influence of switching over to the new curve has automatic influence on measured dust levels and occupational exposure limits. In South Africa, the switchover to the new international harmonisation curve has already been incorporated into the new airborne pollutant guidelines of the Department of Minerals and Energy Affairs (DME) and mines have adopted the new respirable curve. Lack of information on the newly adopted curve has resulted in further confusions such as claims of 'increase' in measured dust levels due to the switch over.

This paper attempts to clarify the misgivings through a field study carried out in an underground coal mine. The results suggest that switching over to the new size-selective curve (ISO-CEN-ACGIH) using the locally made Higgins-Dewell type cyclone results in a decrease in measured dust levels by about 11% on average at the current compliance limit of 2 mg/m³. It appears that this will have an influence on the analysed quartz content of the dust samples as the analytical methods depend on the particle size distribution of the collected dust samples. By switching over to the new harmonisation curve in gold mines would probably result in higher estimated quartz levels due to the collection of fine dust particles than heretofore using XRD or IR techniques.

1. Introduction

Dust sampling is pivotal in estimating the 'dose' of dust exposure and in deriving dose-response curves in epidemiological studies. Dose can be measured by dust sampling but it is not an accurate reflection of the "true dose." This indicates that the dose received by different groups of miners may not be completely characterised by their exposures. This can be attributed firstly, to a diverse mine work force in terms of race, gender, body size, and secondly, miner lung dose depends on breathing rate, particle size, solubility and mouth versus nose breathing.

After the research in the 1950s, it was accepted that dangerous particles are those with particle sizes smaller than 5.0 μm in diameter. This lead to the size-selective sampling curve widely known as British Medical Research Council (BMRC) curve or Johannesburg curve. These curves are actually lung penetration rates of dust particles that instruments attempt to replicate. Some of the recent scientific evidence concerning the hazard from very small particles argues that it may not be appropriate to ignore a specific effect of these on worker's health. Proposed international conventions for respirable size selective sampling (Soderholm, 1989, 1991) for international harmonisation to some extent precisely measures smaller particles than the BMRC curve. Therefore, adapting this curve and its impacts in South Africa are not known and are addressed in this paper.

2. Background

The primary purpose of dust sampling is therefore to characterise (with regard to mass and size) the environment of miners to evaluate their dust exposure. Other reasons include evaluating the effectiveness of engineering controls and changes in dust levels as a result of process changes, and finally as a measure of dose in epidemiological studies. The mass of respirable dust inhaled can be determined by sampling.

The measurement of dust in mines worldwide is usually carried out through various sampling instruments. The collected dust sample is expressed as a mass of dust (mg) per cubic meter (m^3) of air and generally referred to as "dust concentration" in the air. Over the years, various types of dust sampling instruments have been evolved so as the various size-selective sampling curves and the occupational exposure levels (OELs).

OELs provide the necessary guidance for planning, engineering, monitoring and controlling the hazard and work practices for effective control of exposure to substances. There are wide variations in the exposure limits as defined by regulatory and research authorities or scientific associations. The exposure limits set by regulatory authorities of countries worldwide need not be the same, and must not be compared directly with each other because of the differences in each country's exposure measurement, control and assessment strategies. Moving from one size-selective sampling to the other has some basic implications such as using OELs for compliance monitoring and dose-response estimation. Therefore, international harmonisation in dust sampling may avoid all confusions.

In principle, widely available different cyclones or dust samplers require to follow the specified size-selective sampling curves such as BMRC curve or the ACGIH curve or the new ISO/CEN/ACGIH curve. The performance of cyclones is typically described in terms of the 50% (or median) cut-point or D50. The median cut-point reflects the size of dust that the cyclone collects with 50% efficiency. The cut points are defined in relation to the particle penetration into the gas exchange region of the lung. The D50 of the BMRC Curve is 5 μm while the D50 of the new ISO/CEN/ACGIH curve is 4 μm (ACGIH, 1985, Soderholm, 1989, ACGIH 1999, ISO 1995, CEN, 1993). Figure 1 shows the two different size-selective curves that the dust samplers need to follow.

From the curve and as demonstrated below, we notice that a cyclone with a 5 μm cut-point will ideally collect higher mass of dust than that with a 4 μm cut-point:

$$M_{D50} = \rho \times \pi \times \frac{(D50)^3}{6} \qquad (1)$$

where,

M_{D50} = Mass of the dust particle (cut-point) in mg

ρ = Density of the dust particle in mg/m^3

D50 = Aerodynamic diameter of the cut-point in μm.

From the above equation (1), calculated mass of the quartz dust particle with cut-points of 5 μm and 4.0 μm are 0.000000173 mg and 0.000000088 mg respectively. Therefore, the respirable dust collected using different size selective curves will result in different dust masses. While the OEL is set in accordance with the specific size selective curve in mind, the comparison of measured dust mass collected using a different sampling curve and comparing it to the OEL would be incorrect. A study by Kenny et al., (1996) suggested that switching over to new size-selective curve using Higgins-Dewell type samplers would result in apparent decrease in measured levels by about 20 % on average.

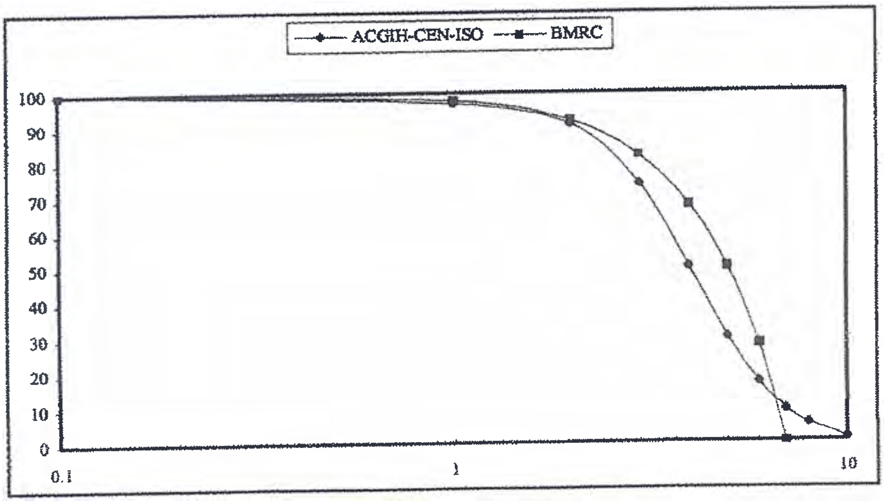

Figure 1: *Respirable dust sampling or size-selective curves*

3. Dust sampling

The paragraph and table below emphasise the importance of adhering to accepted sampling procedures for any given sampling instrument. In most of the South African underground mines, dust samplers (both mine operator and DME) were operated at 1.9 L/min in agreement with the BMRC respirable convention (BMRC, 1952). However, according to the new ISO/CEN/ACGIH respirable dust curve, the recommended flow rate of the dust samplers was 2.2 L/min (Kenny, Baldwin and Maynand, 1998). As a matter of interest, measurement of the size-selection characteristics of the South African cyclones confirmed that they are similar to the Higgins-Dewell designs commonly used in the UK and Europe, and hence for sampling according to the new ISO/CEN/ACGIH respirable convention with a 50% cut-point (D50) of 4 μm. Table 1 summarises the BMRC and ISO/CEN/ACGIH size-selective curves for dust sampling in mines.

BMRC Curve		ISO/CEN/ACGIH Curve	
Particle size μm	Particle mass %	Particle size μm	Particle mass %
0	100	0.1	100
1	98	1	97
2	92	2	91
3	82	3	74
4	68	4	50
5	50	5	30
6	28	6	17
7	0	7	9
		8	5
		10	1

Table 1. *Size-selective curves.*

The switch over to the international harmonisation curve has already been incorporated into the new airborne pollutant guidelines (SAMOHP, 2002) of the Department of Minerals and Energy Affairs (DME) and mines have adopted the new respirable curve. Lack of information on the newly adopted curve has resulted in further confusions such as claims of 'increase' in measured dust levels due to switch over.

4. Data collection

4.1 Dust Measurement

To date in South Africa, there is no scientific study either underground or in the laboratory on systematic comparison on quantifying the influence of switching over to the new international curve on measured dust levels. In this study, dust samples were collected replicating conditions encountered during the actual production shift using BMRC and ISO/CEN/ACGIH size-selective criteria. Personal and area dust samples were collected in a bord-and-pillar continuous minor (CM) section. The personal samplers were worn in the breathing zone and samples were collected at the section intake and nearest to the face area. The area samples were collected at the CM operator position, section intake and in the section return airway.

The objective of the study was to quantify the effect of switching over to new size-selective criteria on measured dust levels under dynamic conditions and its implications in dust exposure assessment.

4.2 Test samplers

For all tests, the locally manufactured and DME-approved 10 mm plastic cyclone (GME-G05) was used. The study involved a total of 5 shifts of measurements representing the actual underground production conditions. Out of the 21 pairs of samples, five pair-wise samples were rejected as one of the pumps of the pairs failed.

The dust-monitoring set-up contained two dust samplers, positioned side by side for personal (left and right lapel) and area sampling. Individual pair-wise SA cyclones were operated at 1.9 L/min and 2.2 L/min according to the BMRC and the new ACGIH/CEN/ISO size-selective curves respectively. The sampler operated at 2.2 L/min of air selectively collects the fraction of airborne respirable dust less than 10 μm particles on a pre-weighed filter disc. Similarly, the sampler operated at 1.9 L/min of air selectively collects the fraction of airborne respirable dust less than 7.0 μm particles on a pre-weighed filter disc. Filters from the samplers were weighed on an analytical electronic balance with readable 0.0001 mg. The procedure for determining the particulate mass was followed as per the DME guidelines (DME, 1997). Pumps were calibrated with 3-digits after the decimal using digital Gillibrator. The flow rates of the pumps were measured before and after the shift.

5. Results and discussions

From the underground measurements, a total of 16 pair-wise sample data was obtained. The data contained five personal sampling data and 11 area-sampling data. The flow rate data of the pumps before and after the shifts operating at 1.9 L/min and 2.2 L/min are summarised in Table 2.

From the analysis it was noted that the average measured flow rate using BMRC size-selective curve was 1.891 L/min. Similarly, the average measured flow rate using ISO-CEN-ACGIH size-selective curve was 2.202 L/min. The average sampling period for the pair wise sample data was 294 minutes representing the actual production shift. The measured dust levels using BMRC and ISO-CEN-ACGIH size selective curves is summarised in Table 3 and plotted in Figure 2.

Test #	Sample #	Flow Rate, Lpm Before	Flow Rate, Lpm After	Sampling Time Minutes
1	1	1.904	1.968	226
	2	2.201	2.187	226
	3	2.200	2.243	236
	4	1.904	1.907	236
	5	1.902	1.587	272
	6	2.206	2.195	272
2	7	2.201	2.187	347
	8	1.904	1.968	347
	9	2.200	2.010	348
	10	1.907	1.899	348
	11	1.902	1.587	360
	12	2.206	2.195	360
3	13	2.208	2.205	311
	14	1.906	1.906	311
	15	1.902	1.926	310
	16	2.206	2.223	310
	17	1.904	1.906	310
	18	2.204	2.211	310
4	19	2.205	2.211	361
	20	1.906	1.906	361
	21	1.903	1.926	361
	22	2.205	2.223	361
	23	2.200	2.208	340
	24	1.906	1.906	340
5	25	2.200	2.222	236
	26	1.906	1.929	236
	27	2.204	2.242	222
	28	1.904	1.925	222
	29	1.903	1.883	236
	30	2.205	2.239	236
	31	2.208	2.212	224
	32	1.906	1.904	224

Table 2. Pump Flow Rate Data Before and After Sampling.

Sample	Ratio of BMRC and ISO/CEN/ACGIH Dust Level
Personal	1.1102
Area	1.1055
Area	1.3233
Personal	1.3136
Area	1.0443
Area	1.2258
Personal	1.0666
Area	0.9979
Area	1.1635
Personal	1.2282
Area	1.4287
Area	1.1016
Personal	1.2503
Area	1.4575
Area	1.1231
Area	1.0255

Table 3. Ratio of measured dust levels using two size-selective curves.

From the results it was noted that when the cyclone operated in accordance with the BMRC curve, the average measured dust level for the sampling period was 5.323 mg/m³ (16 samples). Similarly, when the cyclone operated in accordance with the new ISO/CEN/ACGIH curve, the average measured dust level for the sampling period was 4.604 mg/m³ (16 samples).Personal dust samples were contaminated due to stone dusting in the section.

Overall, from the underground measurements, it was noted that by switching over to the new size-selective criteria, there is a 13.5% reduction in measured dust values. Statistical analysis on the pair-wise data indicates that there is a significant difference (p=0.001) between the measured dust levels between the two size-selective criteria. From the linear regression plot of the data, it can be inferred that there is a reduction in measured respirable dust levels by approximately 11.47 % at the current coal dust compliance limit.

6. Conclusions

The underground study has demonstrated that by switching over to the new ACGIH/ISO/CEN size-selective curve from the old BMRC curve would result in the reduction in measured respirable coal dust levels by approximately 11.47% at the current compliance limit of 2 mg/m³.

The impact of the 'switch over' on occupational exposure limit (OEL) values needs to be addressed in detail with all the relevant stakeholders. It appears that this will have an influence on the analysed quartz content of the dust samples in gold mines as the analytical methods depend on the particle size distribution of the dust. By switching over to the new harmonisation curve in gold mines would probably result in higher estimated quartz levels due to the collection of fine dust particles than heretofore and analysing using XRD or IR techniques. A systematic comparative study in gold mines may give clear indications on the measured silica levels by switching over to the international harmonisation respirable curve. Finally, the paper reminds the careful handling of dust exposure data in deriving the 'dose' for the dose-response studies in future and compliance determination.

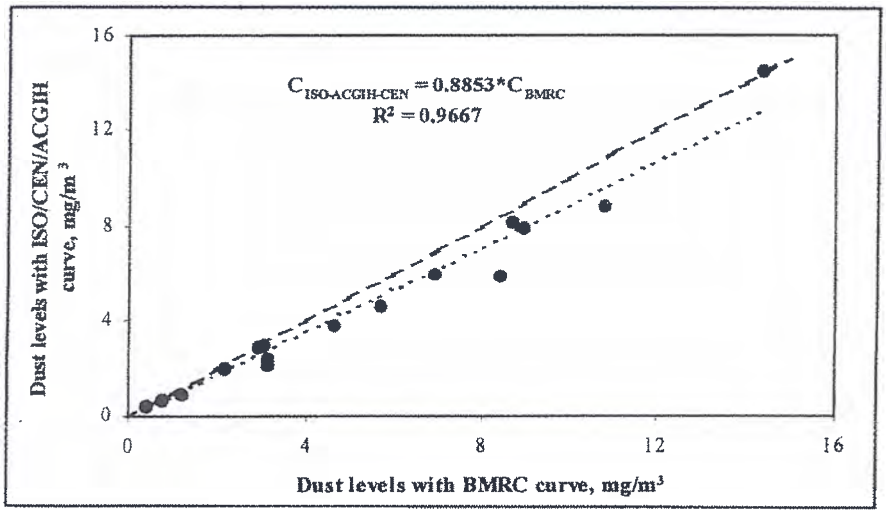

Figure 2. Relationship between measured dust levels using BMRC and ISO/CEN/ACGIH respirable curves.

7. Acknowledgements

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Comparison of three side-by-side real-time dust monitors in a duct using average and peak display dust levels as parameters of performance evaluation

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ABSTRACT: In South Africa, the monitoring of dust in the mines is a requirement in terms of Section 12.2 and 12.3 of the Mine Health and Safety Act (MHSA) of 1996. In order to ascertain the magnitude and range of dust levels and to react when an unhealthy dust exposure occurs, a real-time personal monitoring instrument for mineworkers is undoubtedly required. This paper discusses a comparative study of the three real-time (PDR units) monitors in a duct using coal and sandstone dust. The Higgins-Dewell (HD) and Dorr-Oliver (DO) type cyclone operated in accordance with the international size-selective curve were used as 'true samplers.' The average and peak display levels recorded by the three PDR units positioned randomly side by side, in the duct were analyzed using statistical techniques. The results of the study have showed that the dust levels measured with the three PDR units were not significantly different to the HD sampler data. Interestingly, the results showed significant differences in measured dust levels between HD and DO cyclones positioned side-by-side. The implication of this finding is that the majority of real-time monitors (e.g., Tapered Element Oscillating Microbalance (TEOM)) use these as a 'reference sampler.' This means that, based on measured differences found between the two cyclones, the introduction of TEOM for legal monitoring purposes may create ambiguity in its current state, i.e., agreement on the use of 'true cyclone.' The study demonstrated that, if the DO cyclone were used in the TEOM, it would measure significantly lower dust levels than the HD cyclone. Therefore, consensus on a 'true sampler for use in real-time monitors' must be established in the mining industry.

Pair-wise t-test analyses were performed to compare the three PDR units using the average and peak recorded level. The study indicated that when peak value is used to evaluate the performance between instruments, resulted in different inferences on the recorded levels when compared with the average value. The implication of this is that in practice, the random selection and use of a real-time monitor for engineering dust control application may be in favor or against the seriousness of the dust problem. Although the recorded levels show the differences in dust levels, ANOVA results showed the contrary: dust type, monitoring units or position were not the sources of variation in the measured average and peak dust levels between the three PDR units. Light scattering monitors depend solely on air movement to move the dust particles into the sensing zone. It is unknown, if the particle charges have any specific effect in terms of their movement towards the sensing chamber that could have contributed to the recorded differences. It is proposed that, for real-time monitor evaluation, the use of 'peak display' level may ascertain the probable sources of variations. The intention of this paper is not to suggest that the peak levels should be used in place of average levels for exposure monitoring, rather an evaluation parameter in understanding of variations experienced by researchers.

Keywords: Peak dust levels, real-time monitor, coal dust, silica dust, evaluation, mining

1 INTRODUCTION

In South Africa, the monitoring of dust in the mines is a requirement in terms of Section 12.2 and 12.3 of the Mine Health and Safety Act (MHSA) of 1996. In order to ascertain the magnitude and range of dust levels and to react when an unhealthy dust exposure occurs, a real-time dust monitor for mineworkers is undoubtedly required. Mainly in the USA, the need for the development of a real-time continuous respirable dust monitor has resulted in a new product based on Tapered Element Oscillating Microbalance (TEOM) principle. In South Africa, the quest for real-time monitoring has resulted in a number of research projects that have focused on the issues pertaining to the assessment of dust hazards in mining operations

(Unsted, 1997; Biffi et al., 2000). The research work has shown that the use of direct-reading light-scattering instruments is not reliable due to their inherent sensitivity to particulate matter other than dust. Against this background, the search for an improved or an alternative instrument capable of measuring dust more accurately and reliably is continuing. Therefore, any new information on the real-time dust monitoring techniques or their performance evaluation would be beneficial to the mining industry worldwide.

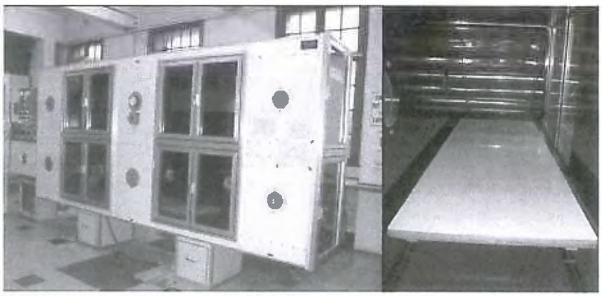

Figure 1. Pictorial view of the Polley duct with a rectangular rolling sampling table (right).

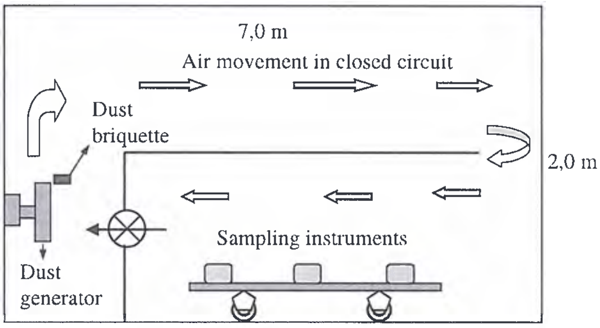

Figure 2. Line diagram of the Polley dust duct operation.

2 REAL TIME MONITORS

Direct-reading instruments or real-time monitors based on light scattering are available to estimate exposure to dust in underground mines. Real-time direct-reading instruments for mine dust have been used worldwide for routine engineering control and risk assessment purposes over two decades due to their added benefits when compared with the gravimetric samplers. All the available real-time monitors are calibrated using 'mono-disperse' particles (Arizona road dust). However, each monitor to be used underground requires a user-determined 'correction factor' obtained from a side-by-side gravimetric size-selective sampler, evaluated with 'poly-disperse' mine specific dust. There is no 'absolute correction factor' available for an individual real-time monitor. The 'correction factor' changes with the history of the sampling data obtained in side-by-side comparisons of the real-time monitor and the type of gravimetric size-selective sampler used.

Direct reading instrument evaluation is not new to the mining industry. Various studies (Williams and Timko, 1984; Page and Jankowski, 1984; Gero and Tomb, 1988; Tsai et al., 1996, Baldwin et al., 1997; Tarkington et al., 1997; Thorpe and Walsh 2002) have evaluated different types of real-time monitors for their usage as personal or area monitors. The conclusions from these studies are similar in terms of their recommendations on the usage of the real-time monitors, but with varying degree of certainty. It is well known that the use of a real-time monitor as a stand-alone unit is not recommended for personal exposure assessment purposes but rather it is more suited to the identification of dust trends during a working shift. Currently, there is no consensus standard on the selection of a suitable real-time instrument for use to the industry. Field trials using real-time dust monitors used in conjunction with the visualization system revealed that its response could vary significantly from one day to the next (Thorpe and Walsh, 2002). The sources of variability of the real-time monitors can be attributed to dust levels, dust type, dust size, air velocity, monitor orientation and contamination of optics, etc. This paper investigate the results of three real-time monitors positioned side-by-side in a duct and evaluates the instruments based on the average and peak measured dust levels as parameters of evaluation.

3 LABORATORY STUDY

This section of the paper discusses the laboratory evaluation of three real-time monitors (PDR) positioned side by side along with gravimetric samplers in a laboratory known as the Polley duct (Figure 1).

3.1 *Polley duct*

The Polley duct consists of a closed-circuit duct and two dust generators (Figure 2). The closed-circuit duct consists of two sections: a horizontal section and a vertical section. The horizontal section is the main section and measures 7.0 m long by 2.0 m high by 0.7 m wide.

The air flows along the horizontal channel into and along the top half of the large horizontal section. It returns along the bottom half of the large horizontal section through a flow-straightening section and flows along the lower, small horizontal channel into, and upwards in the vertical section to close the circuit. The duct also has other auxiliary parts such as time relays, two fans to circulate the air and a third to exhaust the dust-laden air through a filter to atmosphere, and a dust briquette press.

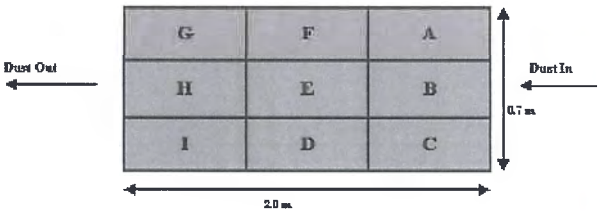

Figure 3. Instrument positions on the sampling table.

3.2 *Test instruments and methodology*

The three real-time monitors that were used for evaluation purposes were commonly known as PDR or MIE DataRam (USA). The units operate on forward light-scattering particle detection principle, which relies on ambient air movement to introduce particles into the sensing chamber. The PDR real-time monitor displays dust level in mg/m^3 in addition to TWA, Max, Min, STEL and sampling time on the display readout. The instrument has a preliminary Intrinsically Safe (IS) certificate obtained from the South African Bureau of Standards (Grupping, 2001).

For the evaluation purposes of real-time monitors, Higgins-Dewell (HD) and Dorr-Oliver Cyclones were used as these are the commonly used size-selective devices used worldwide. It is assumed that HD cyclone and DO cyclone or sampler gave zero or negligible errors and is a representative sample of 'true' measured dust level in the chamber. The HD cyclone was operated at 2.2 L/min and DO cyclone was operated at 1.7 L/min in terms of new international harmonization (ISO/ACGIH/CEN) size-selection curve. For each test, the samplers were positioned side by side inside the lower chamber of the duct (Figure 3). Each of the three real-time monitors was randomly positioned on locations D or E or F, while HD and DO cyclones were positioned at location A or C.

A low air velocity ($\sim$0.8 m/s) in the chamber was maintained consistently for all the tests. For the study, the instruments were exposed to two types of dust, viz. coal and sandstone briquette dust. The quartz content of the sandstone briquette dust was 50.63%. The real-time monitors were calibrated (zeroing) using an airtight polythene bag supplied by the manufacturer after each test. The test chamber did not have any instruments to measure the size distribution of the airborne dust in real-time. The detailed experimental procedures are discussed elsewhere (Belle, 2002). The cyclone inlets faced the direction of the airflow in order to avoid the effect of nozzle inlet orientation on sampler performance. New Gillian constant volume flow pumps were used and were calibrated to the nearest ml per minute flow rate, using a digital Gillibrator. The gravimetric sampler dust levels were determined in accordance with the established Department of Minerals and Energy procedures (DME, 1997). A photographic view of the real-time monitors and their positions during tests in the chamber is shown in Figure 4.

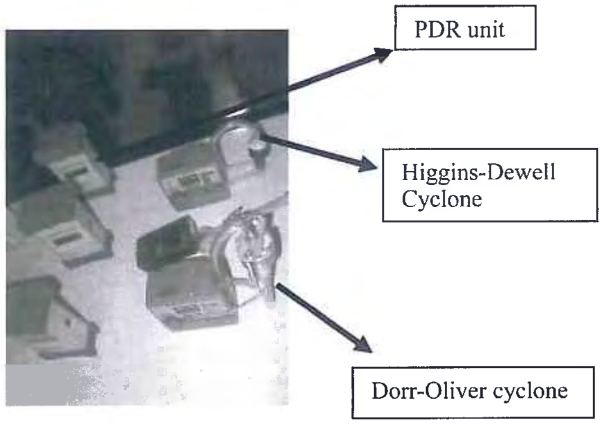

Figure 4. Pictorial view of test real-time monitors (left) and gravimetric samplers (right).

4 LABORATORY RESULTS

Initially tests were carried out to confirm that the dust levels across the Polley duct were uniform. Both DO and HD cyclones were positioned side-by-side in the dust chamber and tests were conducted for both coal and sandstone dust. Preliminary results indicated that there is no significant difference in the measured dust levels across the chamber. For example, the measured coal dust levels at positions B and H were 4.94 mg/m^3 and 4.91 mg/m^3, respectively. Similarly, the measured sandstone dust levels at positions B and H were 33.35 mg/m^3 and 31.90 mg/m^3 respectively. Figure 5 shows the respirable dust levels obtained in the side-by-side comparisons of similar types of cyclones. The correlation coefficient (r) between the two side-by-side DO cyclones was 0.993. Similarly, the correlation coefficient (r) between the two side-by-side HD cyclones was 0.998. A combined plot of the two data ($r = 0.998$) indicates a strong linear relationship between the two side-by-side cyclones. The two data sets of dust values showed that concentration across the chamber was uniform during the test conditions.

Figure 6 show the relationship between the measured dust levels using the DO cyclone and the HD cyclone positioned randomly, side-by-side in the test chamber. From the plot and the regression equations it is noted that the DO cyclone measured approximately 16% less respirable dust than the HD cyclone for a personal coal dust compliance limit of 2 mg/m^3.

The implications of this finding is that majority of the real-time monitors use cyclones as a 'reference sampler' operated in accordance with the accepted size-selective curve. The newly developed Tapered Element Oscillating Microbalance (TEOM) real-time

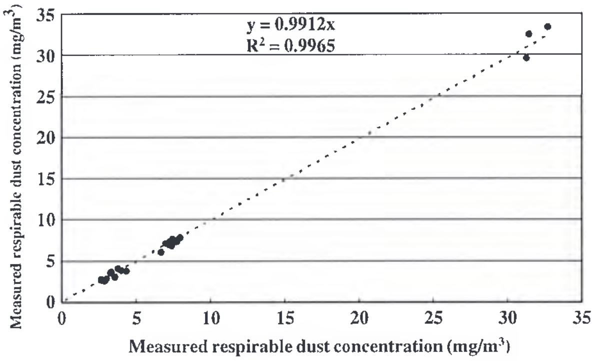

Figure 5. Combined data of two side-by-side cyclones (HD and DO).

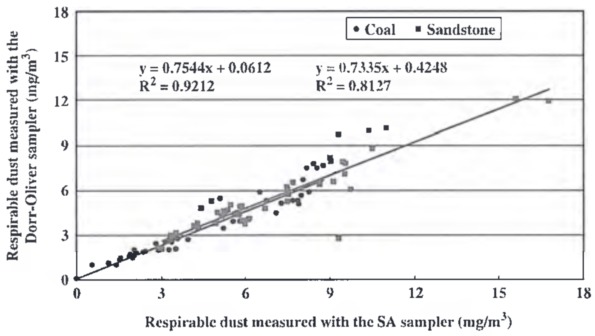

Figure 6. Relationship between measured dust levels using side-by-side DO and HD cyclones in the test chamber.

monitor uses the HD cyclone for its operation. This means that, based on measured differences found between the two cyclones for the US or SA industry, the introduction of TEOM for legal monitoring purposes may create ambiguity at its current state, i.e., agreement on the use of 'true cyclone' in real-time monitors. This study did not evaluate any imprecision of either HD or DO cyclones. Also, from South African experience, by switching over to the new size-selective curve, the measured coal dust levels were 11% lower than before at 2.0 mg/m^3 personal exposure limit (Belle, 2005). Currently there are no changes to the personal exposure limit due to the change over to the new size-selective curve. For this paper the HD sampler was used as a reference sampler.

4.1 *PDR results*

The results of the variation between the dust levels measured by three PDR dust monitors positioned side-by-side in conjunction with the gravimetric samplers are discussed below. Figure 7 show the relationship between gravimetric and real-time monitors for two different dust types. The solid line represents 1:1 relationship between gravimetric and real-time monitors. The coefficient of variation (CV) of the mean correction factors is in an increasing order for

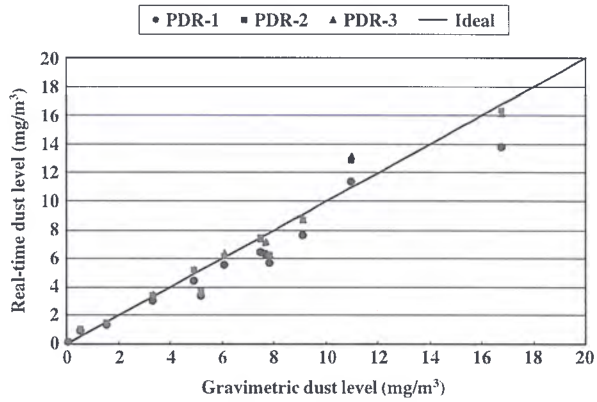

Figure 7. Relationship between measured gravimetric and recorded real-time monitor dust levels.

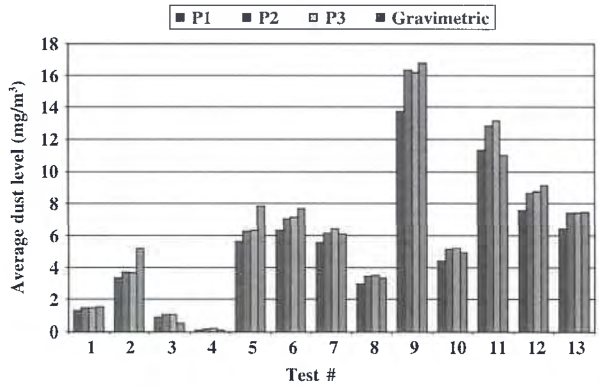

Figure 8. Relationship between average dust levels recorded by PDR units and gravimetric sampler.

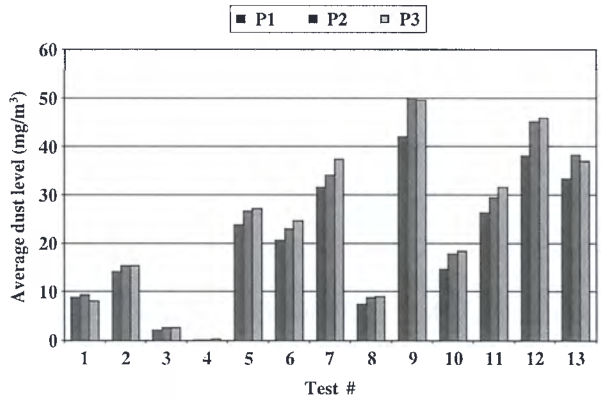

Figure 9. Peak real-time levels recorded by PDR units for coal and sandstone dust.

PDR instruments, P1 (8%), P2 (14%) and P3 (17%). The lower the CV of the mean correction factor, the more linear the response of the monitor.

Figures 8 and 9 show the average and peak (maximum) dust levels recorded by the PDR instruments positioned side-by-side, randomly, in the test chamber for two dust types respectively. Tables 1 and 2 show the summary statistics of the average and peak respirable

Table 1. Average and Peak dust levels using Coal Dust.

	Average dust levels, mg/m^3			
Test #	P1	P2	P3	HD*
45	1.30	1.50	1.48	1.56
46	3.34	3.70	3.67	5.22
47	0.91	1.05	1.07	0.55
49	0.13	0.17	0.19	0.09
50	5.63	6.25	6.30	7.86
51	6.32	7.04	7.19	7.69
	Peak dust levels, mg/m^3			
Test #	P1	P2	P3	HD*
45	8.80	9.29	8.09	1.56
46	14.2	15.33	15.32	5.22
47	2.18	2.72	2.63	0.55
49	0.23	0.26	0.33	0.09
50	23.84	26.62	27.18	7.86
51	20.57	22.96	24.77	7.69

* Higgins-Dewell gravimetric value

Table 2. Average and Peak dust levels using Sandstone Dust.

	Average dust levels, mg/m^3			
Test #	P1	P2	P3	HD*
45	5.56	6.15	6.44	6.11
46	2.99	3.43	3.50	3.35
47	13.78	16.37	16.18	16.77
49	4.43	5.14	5.22	4.94
50	11.30	12.83	13.17	10.99
51	7.61	8.65	8.77	9.12
52	6.41	7.45	7.41	7.49
	Peak dust levels, mg/m^3			
Test #	P1	P2	P3	HD*
45	31.51	34.02	6.44	6.11
46	7.33	8.78	8.95	3.35
47	41.99	49.95	49.61	16.77
49	14.69	17.87	18.27	4.94
50	26.36	29.54	31.59	10.99
51	38.20	45.15	45.91	9.12
52	33.33	38.35	36.99	7.49

* Higgins-Dewell gravimetric value

dust values obtained from the side-by-side comparison of the PDR monitors using coal and sandstone dust respectively.

From the data it is noted that the average measured levels by the units P1, P2, P3 and gravimetric sampler were 2.94 mg/m^3, 3.29 mg/m^3, 3.32 mg/m^3, and 3.83 mg/m^3, respectively for coal dust. From the sandstone dust, it is noted that the average measured dust levels by the units P1, P2, P3 and gravimetric sampler were 7.44 mg/m^3, 8.57 mg/m^3, 8.67 mg/m^3, and 8.40 mg/m^3, respectively. Using the combined data, the average measured levels by the units P1, P2, P3 and gravimetric samplers were 5.36 mg/m^3, 6.13 mg/m^3, 6.19 mg/m^3, and 6.29 mg/m^3, respectively.

Similarly, peak dust levels recorded by the three real-time units were compared. From the data it is noted that the average of peak recorded dust levels recorded by the units P1, P2, and P3 were 11.64 mg/m^3, 12.86 mg/m^3, and 13.05 mg/m^3, respectively for coal dust. From the sandstone dust data, it is noted that the average of peak dust levels recorded by the units P1, P2, and P3 were 27.63 mg/m^3, 31.95 mg/m^3, and 32.69 respectively. Using the combined data, the average of peak levels recorded by the units P1, P2, and P3 were 20.25 mg/m^3, 23.14 mg/m^3, and 23.65 respectively.

From the plots and tables it is observed that the average and peak display dust levels recorded by the three PDR units positioned side-by-side would differ when the instruments were exposed to the same dust cloud with inherently the same size characteristics (the dust source, dust generation and airborne mechanism). Comparison of the PDR and the HD sampler dust levels indicate that there is no statistically significant difference in measured levels for both dust types for all three units (high p-value). It appears that the difference between recorded levels by the real-time monitors is slightly pronounced when peak value is used as a performance indicator.

5 STATISTICAL ANALYSES

This section of the paper discusses the analyses of the data using appropriate statistical techniques. A paired t-test was performed on the set of real-time pair dust data to determine whether there was a statistical difference in the ratio of dust levels measured between real-time monitor and HD cyclone. A paired t-test of hypotheses was developed to compare the concentration level ratios (mean and peak) measured with three real-time monitors (P1, P2 and P3) and HD cyclone. The null and alternative hypotheses for the sample pairs tested were:

$$H_0 : CR_{P1} = CR_{P2}$$
$$H_a : CR_{P1} \neq CR_{P2}$$

For example, in the paired t-test, hypothesis H_0 states that the average dust ratios between two real-time monitors (P1 and P2) are equal. On the other hand, the alternative hypothesis states that the two real-time monitors, in fact, measure different average dust levels. The results of the paired t-test statistical analyses

Table 3. Results of paired t-test (average transformed values).

Pair	Dust	#	T-value	p-value	Hypothesis
P1-P2	Coal	6	−4.46	0.007	Reject
P1-P3	Coal	6	−3.59	0.016	Reject
P2-P3	Coal	6	−1.24	0.270	Accept
P1-P2	Sandstone	7	−16.30	0.000	Reject
P1-P3	Sandstone	7	−48.01	0.000	Reject
P2-P3	Sandstone	7	−2.07	0.083	Accept
P1-P2	Combined	13	−9.18	0.000	Reject
P1-P3	Combined	13	−7.68	0.000	Reject
P2-P3	Combined	13	−2.22	0.046	Reject

Table 4. Results of paired t-test (peak transformed values).

Pair	Dust	#	T-value	p-value	Hypothesis
P1-P2	Coal	6	−4.71	0.005	Reject
P1-P3	Coal	6	−2.42	0.060	Accept
P2-P3	Coal	6	−0.54	0.613	Accept
P1-P2	Sandstone	7	−9.42	0.000	Reject
P1-P3	Sandstone	7	−12.98	0.000	Reject
P2-P3	Sandstone	7	−1.53	0.176	Accept
P1-P2	Combined	13	−9.20	0.000	Reject
P1-P3	Combined	13	−5.93	0.000	Reject
P2-P3	Combined	13	−1.09	0.295	Accept

(for average and peak data) are given in Tables 3 and 4. For the analyses, a cut-off p-value of 0.05 was used (95% confidence level).

Using the average recorded level as a performance evaluation parameter (Table 3), a large p-value (>0.05) is observed suggesting that the measured mean concentration ratios are consistent with the null hypothesis. That is, the dust levels recorded by P2 and P3 are not affected at the 95% level of confidence for coal and sandstone dust. For the combined data, there was a significant difference between the recorded dust (coal and sandstone) levels between all the three units, viz., P1 and P2; P1 and P3 (p-value of 0.000); and P2 and P3 (p-value < 0.05).

Similarly, using the peak recorded level as a performance evaluation parameter (Table 4), a large p-value (>0.05) is observed suggesting that the measured peak and mean concentration ratios are consistent with the null hypothesis. That is, the dust levels recorded by P2 and P3 are not affected at the 95% level of confidence for both types of dusts. For the combined data, there was a significant difference between recorded dust (coal and sandstone) levels between the units, viz., P1 and P2; P1 and P3 (p-value of 0.000). The study indicates that, when peak value is used as a parameter to evaluate the performance between monitors, different inferences could be drawn on recorded dust levels by the three PDR units when compared with the average value.

5.1 *ANOVA*

Upon noting the differences between the recorded levels by the three real-time units, an analyses of variance (ANOVA) were performed. Typical sources for these variation are sampling type (active or passive), dust types, monitor orientation, size distributions of dust, air velocity, sensor contamination etc. The measured dust concentration ratio between the real-time dust monitors' data (average and peak) and the reference HD sampler data was used to perform an ANOVA using MINITAB 13.2 statistical software.

For this study, the sources of variation quantified were the influence of dust type (coal and sandstone), monitoring units (P1, P2 and P3) and the position of the dust-monitoring units in the dust chamber. Essentially, the measured dust concentration ratio data that were used for the analysis were in the form of CA_{ijk} (mg/m^3) and CP_{ijk} (mg/m^3) with the following definitions:

CA = Ratio between real-time dust monitor and average dust level measured using HD cyclone

CP = Ratio between real-time peak value and average HD cyclone

$i =$ dust type (DT), $i = 0$ is a coal dust, $i = 1$ is a sandstone dust

$j =$ monitoring unit (MU), $j = 0$ is unit-P1, $j = 1$ is unit-P2, and $j = 2$ is unit-P3

$k =$ unit position (UP), $k = 0$, 1, and 2 indicate the sampling positions (randomly selected) across the dust chamber respectively.

The results of the analysis of variance (ANOVA) on the average and peak concentration ratio data are summarized in Tables 5 and 6.

The ANOVA tables give for each term in the model, the degrees of freedom, the sums of squares (SS), the adjusted mean squares (MS), the F-statistic from the adjusted mean squares and its p-value.

In the ANOVA table some p values were less than 0.05, indicating that these factors are significant in influencing the concentration values. From the ANOVA results using average and peak level data, the conclusions are summarized hereafter. The effect of dust type on the dust concentration ratio between the real-time monitors positioned side-by-side is significant (p-value of 0.017). There is slight evidence (p-value of 0.096) of the effect of unit position on the measured dust levels when the units are exposed to the same dust cloud using average value as the reference parameter of evaluation. However, the dust-monitor's performance is not significantly affected by the position of the monitoring unit within the chamber or dust type for peak dust data (p-value >0.50). As we note from the table, the interactions between the main factors do not have any significance on the measured dust

Table 5. Results of ANOVA for average values.

Sources of variation	Df	SS	MS	F-value	Pr > F
Monitoring Unit (MU)	3	0.39	0.13	1.07	0.359
Dust Type (DT)	1	0.35	0.82	6.59	0.017
Unit Position (UP)	2	0.27	0.32	2.57	0.096
MU * DT	2	0.03	0.11	0.89	0.425
MU * UP	4	0.57	0.18	1.44	0.251
DT * UP	2	0.62	0.31	2.51	0.102
High order Interactions	25	3.11	0.12		
Total	38	5.31			

Table 6. Results of ANOVA for peak values.

Sources of variation	Df	SS	MS	F-value	Pr > F
Monitoring Unit (MU)	3	2.26	0.26	0.21	0.816
Dust Type (DT)	1	0.06	0.45	0.35	0.560
Unit Position (UP)	2	2.68	0.41	0.32	0.727
MU * DT	2	0.16	1.11	0.88	0.429
MU * UP	4	7.52	2.77	2.17	0.102
DT * UP	2	5.70	2.85	2.23	0.128
High order Interactions	25	31.9	1.28		
Total	38	50.33			

levels for both average and peak data. Overall, the ANOVA conclusively indicated that the factors such as dust type, monitoring unit or its position are not the sources of variation in the measured average and peak dust levels between the three PDR units.

6 DISCUSSIONS

The following paragraphs discusses the results of the study in light of the use of appropriate 'true reference sampler' for real-time monitor and probable unknown sources of variation in measured levels between three PDR units.

6.1 *Use of appropriate reference sampler*

Using cyclones as 'true reference' samplers for real-time monitor evaluation is not new to the mining industry (Kissell et al., 2002). The conclusions from the historic real-time studies are similar in terms of their recommendations on its usage. In past decades, researchers used the MRE 113a, which followed the Johannesburg curve, as a benchmark 'true sampler.' In general, the correction factors of the real-time monitors could be explained by the size-dependent light-scattering characteristics of the sensors with respect to any of the respirable size-selective sampling conventions and reference samplers.

The implications of the findings on the differences in measured levels between HD and DO cyclones considered to be 'true samplers' is that the majority of real-time monitors use them as a 'reference sampler.' The newly developed Tapered Element Oscillating Microbalance (TEOM) real-time monitor uses the HD cyclone for its operation as a real-time monitor. This means that, based on measured differences found between the two cyclones, the introduction of TEOM for legal monitoring purposes may create ambiguity in its current state, i.e., agreement on the use of 'true cyclone'. The study has demonstrated that, if the DO cyclone were used in the TEOM, it would measure significantly lower dust levels than the HD cyclone (although the HD cyclone is beneficial in terms of its sensitivity to higher flow rates). Therefore, owing to the differences observed in this study, the need for a consensus on a 'true sampler for usage in real-time monitors' which operates according to the proposed new international size-selective curve exist in the mining industry. Furthermore, from the South African experience, by switching over to the new size-selective curve from Johannesburg curve using the HD cyclone, the measured coal dust levels were 11% lower than before at 2.0 mg/m^3 (Belle, 2004). Currently there are no changes proposed to the personal coal dust exposure limit due to the change over to the new size-selective curve.

6.2 *Sources of variation between PDR units*

The results of the study have showed that the dust levels measured with the three PDR units were not significantly different to the HD sampler data. Historically, sources of variations in measured dust levels in real-time monitors have been evaluated for parameters such as dust types, dust levels, monitor orientation, particle size, air velocity, and sensor contamination. In this study, sources of variations evaluated in recorded levels between three PDR units were dust type, monitoring unit and monitor position. Although the recorded levels show the differences in dust levels, dust type or monitoring units or position were not the sources of variation. Therefore, probable sources, which is not known or understood, may provide answers to differences in measured levels between three PDR units. Parameters such as air velocity, monitor orientation, particle size were not the sources of variation, as they remained constant for all the tests.

It is often noted in studies that one of the major sources of variations in measured dust levels by the dust monitors could be the size distribution of the parent dust (Soderholm, 1989, Volkwein, 2002; Ramani, 2004). However, in these tests, the size distribution of the parent dust source, dust generation and airborne mechanism has been consistent for all the tests. All units were exposed to similar temporal and spatial environmental conditions. Therefore any differences in their responses were due to the sampling characteristics of the dust monitors alone. The monitor differences can be attributed to the differences in sensor detection range of units, which is 'in-built' to the calibration factors of the PDR units as these units does not have any manual calibration feature. Usually if the optics of the sensor is contaminated the calibration of the monitor gives 'high background' reading. Interestingly, during the tests, none of the three PDR units gave any 'high background' or 'calibration problem' conditions despite exposure to high dust levels. Also, time had no significant influence on the sensors or lenses, or on the correction factor of the real-time monitors as all the three units had the equal exposure period.

A study by Thorpe and Walsh (2002) showed that effects of three separate PDR orientation to the airflow (upright, on its back on its side) and its influence on measured concentration showing the variation in correction factor of 0.69 to 0.92. In this study, for the same orientation of three PDR units had the correction factors of 0.95, 1.10, and 1.13. Therefore, the PDR orientation to the airflow may not be the source of variation. Lastly, light scattering real-time dust monitors depend solely on air movement to move the dust particles into the sensing zone of the monitor. It is not known, if the particle charges of airborne respirable dust have any specific effect in terms of their movement towards the sensing chamber that could have contributed to the recorded differences in dust levels by the three PDR units.

7 CONCLUSIONS

The following conclusions can be drawn from the laboratory evaluation of real-time monitors evaluated using average and peak recorded dust levels. From the statistical analysis of side-by-side comparison of three PDR units and HD cyclone indicate that there is no significant difference in measured dust levels. The use of peak-recorded levels indicates that the differences in recorded dust levels between monitors exist. The evidence from the study suggested that, while the 'average' dust level is a commonly used parameter for evaluating monitor performance, the use of the 'peak display' parameter may lead to different conclusion on the variations between measured dust levels. The implication of this is that, in practice, the random selection and use of a real-time monitor for engineering related dust control application may be in favor or against the seriousness of the dust problem and could impact the decision making process on the appropriate allocation of financial and administrative resources. It is proposed that for any new real-time monitors the use of peak display level as a parameter of evaluation may ascertain the probable sources of variations in recorded levels. These additional data analyses steps may facilitate the adjudging of the sources of variations in measured dust levels in real-time monitors. The intention of this paper is not to suggest that the peak levels be should used in place of average levels for exposure monitoring, rather an evaluation parameter in the understanding of variations experienced by researchers worldwide. It is recognized that Tapered Element Oscillating Microbalance instrument developed by Rupprecht and Patashnick (USA) may be a step closer in minimizing errors in dust measurement than the light scattering instruments.

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A case for revision of time-honoured mine ventilation design parameters-main airways

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ABSTRACT: Mines around the world are inherently and operationally diverse. Current methodology for the design of mine ventilation and cooling infrastructure involves controlling major safety and health hazards using standard design parameters. Globally, it can be estimated that mine ventilation systems from various commodities, circulate over 150,000 m^3/s of exhaust air through shafts, in addition to over 2,000,000 workers are transported through shafts and travel ways in labour intensive mining countries every day. A key objective of this paper is to share experiences and provide a case to consider a revision of 'time-honoured' standard design velocity parameters. These design values are often used in determining number and size airways in mains, size and number of shafts. This paper uses exhaust shaft velocity data from different mining commodities and limited coal mine data from main airways for discussions on the need for revision of design values.

Based on the collated global exhaust shaft velocity data, it is noted that a significant portion of shaft velocities are outside the current design velocity of 20 m/s (some operating up to 30 m/s) and other significant portion of the exhaust shafts operate in the critical velocity range. Large variations in field velocity data direct us to question the appropriateness of these velocities in the current ventilation designs. Most common reasons for not considering such a standard review were found to be, viz., they are 'time-honoured' values and cannot be changed, mine fan pressure limits, possible spontaneous combustion risks, leakages or simply did not consider such a possibility. It is hoped that the paper provides an agreement on the need for revision of standard design velocity values and provide area of opportunities to future mine ventilation and cooling designs.

1 Introduction

The ventilation of mines involves the management of the atmospheric environment. The current practice of mine ventilation is believed to have originated from the recorded works of Atkinson (1854) in the north of England (McPherson, 1984). The mine ventilation system involves supply, control of air and its movement to meet the health and safety standards and provide comfort to workers. Mine ventilation is a strategic component of an operation, whereby regardless of the production, mine has to be ventilated. Figure 1 shows the mine ventilation and refrigeration system key value drivers that present optimization opportunities.

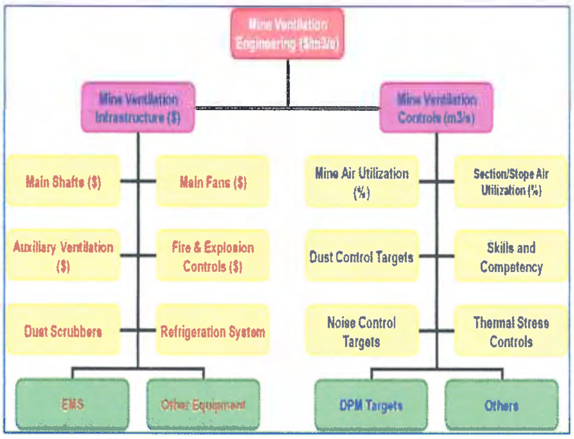

Figure 1. Mine ventilation system key value drivers.

These value drivers can be classified into mine capital ventilation infrastructure cost components and mine operating cost components and typically fall under the ownership of the mine ventilation department.

With the rising awareness of new hazards and their stringent safe limit values, ventilation infrastructure and designs must have capability to handle any unanticipated capacity surprises. For example; a semi-automated world-first automated truck loop in a diamond operation without an appropriate occupational environment had resulted in failure of critical components of that equipment. Therefore, provision of adequate ventilation applies to both labour intensive and automated mining operations.

To date, reaching current mining depth of 4100 m below surface (South African gold mines) and safe operation of some of the gassiest and hot coal mines (Australia) in the world can be attributed to developments in mine ventilation and refrigeration, rock mechanics and step changes in mine designs. These facts have demonstrated that the global mining industry is capable of exceeding barriers considered impossible in the past, are routinely overcome today through technical innovation.

The typical health and safety hazards found in mines are gases, dust, heat, and diesel particular mater (DPM). Mining depth and its associated health and safety hazards vary from continent to continent and between commodity types. For the South African platinum mining industry, a depth of 1000 m is equivalent in heat load factor [virgin rock temperature (VRT) confirmed by site measurements] to the environmental conditions of gold mining at a depth of 3000 m (Figure 2). Similarly, Australian coal mines (Queensland) at a depth of 400 m are on the verge of confronting the same challenges that the gold mines did in South Africa, in addition to the sponcom and/or methane challenges.

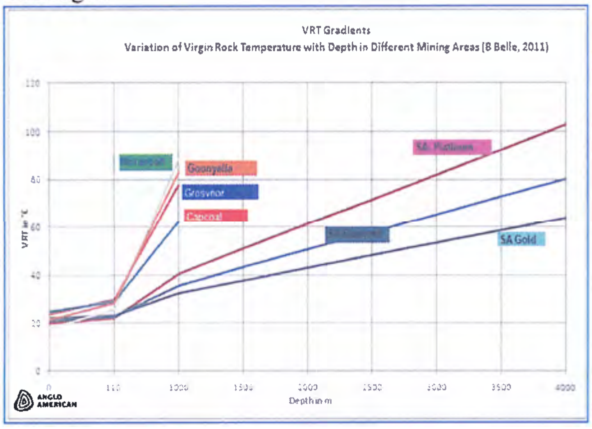

Figure 2. Relationship between VRT, depth, geography and commodity type (Belle, 2011).

The impact of advances in production technology on mining requires maintaining a suitable and cost effective occupational environment. However, it would be too risky to be satisfied with what has been provided over few decades ago. Based on observations and interactions with the mining and ventilation professionals globally, it is noted that despite all the achievements, there are still opportunities for optimizing the mine ventilation systems. This paper identifies and attempts to show the current practices and potential opportunities by reviewing the ventilation design parameters, in particular main airway velocities.

It was noted that in the 1940s, the recommended air velocity in intake men and material shafts, exhaust shafts, main airways, secondary airways were 10.16 m/s, 15.24 m/s, 5.08 m/s and 2.54 m/s respectively (Jeppe, 1946). In later years, a significant amount of work was done in the 1960s with regard to the optimum air carrying capacity of mine shafts and mine airways (Barenbrug, 1963). In addition, other ventilation engineering resources and guidelines (MVS Data Book, 1999; McPherson, 1984, Mousset-Jone, 1986) have provided the basis and background to current design values for mining and ventilation engineers globally. There have been historic surveys of ventilation practices which provide useful information on ventilation air factors and costs associated with ventilation (Mousset-Jones, 1986). However, in recent years, it is not common to find such detailed surveys or collated information on typical air velocities in different mining commodities.

The mining industry has evolved in the last two decades and efforts have been made at mining operations to quantify ventilation costs (Belle, 2005) and internal revision of exhaust shaft design velocity values (Belle, 2008) with certain cost information and field experiences of operating mines over few decades.

There are widely published and accepted ventilation design standards on airway velocities, viz., men and material shaft, dedicated intake shaft, exhaust shaft, travel road, conveyor road, working faces, main intake roadways, main return roadways (Jeppe, 1946; Lambrechts, 1974, Lambrechts and Howes, 1989; MVS Databook, 1999; McPherson, 2009). These ventilation design values have significant influence during mine planning in terms of main shaft and main airway sizes, number of roadways in mains or panels to carry certain design ventilation loads, e.g., 6 heading mains or 8 heading mains, 2 heading roads or 3 heading roads in coal mines.

Figure 3 show an example of a simulated ventilation model of an operating longwall coal mine with seven heading mains and an exhaust fan system and air velocities that are discussed in this paper (Figure 4).

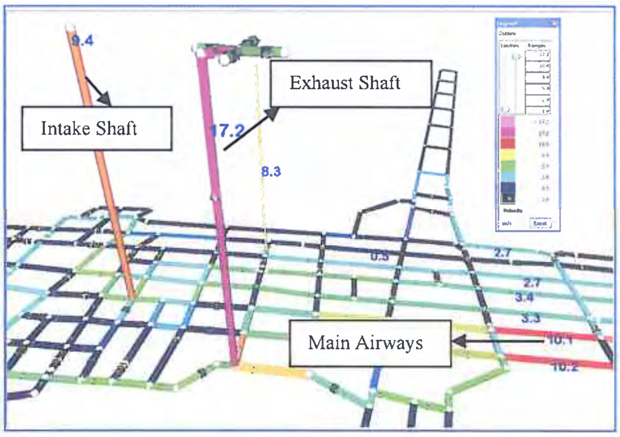

Figure 3. Airway velocities of an operating gassy longwall coal mine.

Figure 4. Coal mine main fan and duct system on shaft.

At present, most of the mine ventilation planning and designs make use of recommended airway velocities (Table 1) based on historic studies and experiences. These values are often reflected in internal mine design standard documents or project design reports and internal project review guidelines. Other important recommended design standard is the velocity range of 7 m/s to 12 m/s, which are known as critical velocity zone, are to be avoided in wet exhaust shafts to prevent water blanketing. In practice, regardless of the wet condition, this air velocity range is often applied stringently in design calculations.

Table 1. Recommended maximum velocities (m/s).

Area	V1*	V2 (coal)**	V3** (metal)
Working faces	4	-	-
Conveyor drifts	5	5	5
Main haulage routes	6	-	-
Smooth lined mine airways	8	-	-
Ventilation Shafts	20	18-22	18-22
Decline Intakes	-	6-8	6-8
Dedicated Intake Shaft	-	18-22	18-22
Downcast Shafts	-	10-12	10-12
Intake Airways	-	2-5	6-8
Return Airways	-	3-5	6-8

* McPherson (1984); Mousset-Jones (1986)
** MVS Data Book (1999)

Key air velocities of main airways that carry the ventilation load are mine intake shafts or declines, mine return airways, main conveyor belts, main intake airways and main return airways. Based on these recommended maximum airway velocities, mining designs will often determine the number of mains, and shaft sizes for an expanding operation or new projects.

As seen in the Table 1, a rule of thumb used for most vertical intake men and material shafts should not exceed 10 to 12 m/s due to cage vibration and its use as a travel way. Only coal mine operating with a downcast system in Queensland operates with a downcast air velocity of 9 m/s. Recently, there have been efforts to increase intake men and material air velocities up to 12 m/s with mine cage speeds up to 18 m/s as in the case of 3000 m deepest single drop South Deep shaft in South Africa. Similarly, in South Africa, deepest diamond mine cage speeds up to 14 m/s with downcast velocity 10-12 m/s and most thermal coal mines, platinum and gold mine downcast shaft air velocities are typically between 10-12 m/s with winter air temperatures of negative 4 ° C with use of bulk air cooling on surface.

As this paper mostly focuses on exhaust shaft air velocity, field observations have indicated that, regardless of the commodity or continent where mine operates, almost all life of mine (LOM) design values provided are in the same range of 20 m/s to 22 m/s. From a due diligence perspective, there have been no documented views expressed on these optimum exhaust air velocity design values. The only references that the author can trace exhaust shaft velocity of 20 m/s and other maximum air velocities is found in technical paper titled "the mine ventilation planning in the 80's" by Prof. McPherson (1984) and MVS Data Book (1999).

In many operations globally, these maximum velocities by now have become embedded and any suggestion of velocities higher than the recommended maximum velocities are often seen with extreme caution or dismissal. These situations many a times simply results in loss of improvement opportunities or areas for further technical debate. However, velocities higher than recommended operating velocities are not new to mine ventilation fraternity and can be found in the mine ventilation networks. For example; velocity ranges of 40 m/s to 50 m/s can be found in an exhaust fan duct system as shown in an example computational fluid dynamics (CFD) model in Figure 5.

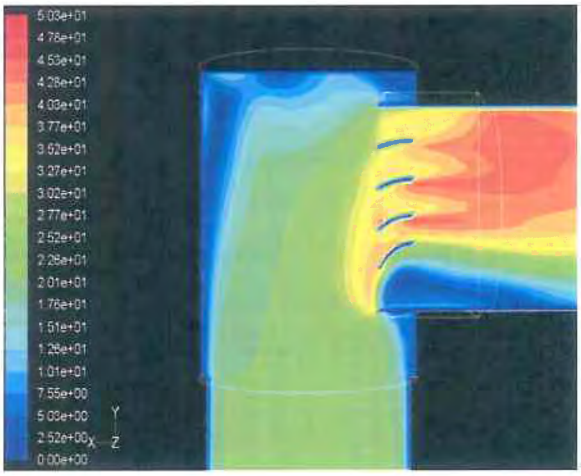

Figure 5. An example of simulated velocity contours (legend left hand side) in an exhaust shaft and fan duct system (Source: Basu, A, 2007).

This paper attempts to use exhaust shaft velocity data from different mining commodities and limited coal mine data from main airways for discussions on the need for revision of ventilation design values.

2 Airways Background

In most deep underground mines, there are two types of vertical shafts, i.e. intake and exhaust shafts (Jeppe, 1946). In some shallow coal mines of South Africa and Australia, vertical intake shafts are very rare and substituted by intake portals for travel and conveyor belt. Shafts and or portals are the main arteries of a mine, because it is through shafts that all underground workers travel to and from their workplaces, along with the hoisting of materials, conveying of utilities such as electric power, chilled water and ice transport, service water, pump reticulation, communication, emergency access, and ore and waste rock hoisting; while also circulating fresh air through the mine workings and back to surface. It can be demonstrated that current shaft sizes are primarily dependent on ventilation and cooling requirements as shown in example deep gold mine in South Africa (Figure 6).

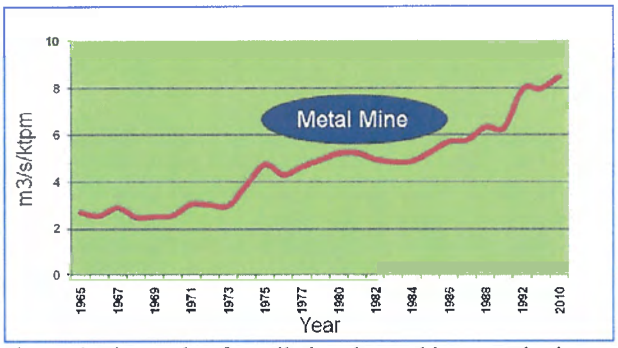

Figure 6. Timescale of ventilation demand in a metal mine.

It can be noted that the metal mine air factor (m^3/s/ktpm) has tripled from 1960s to 2000s. This increase in air factor can be attributed to increased depth, management of hazards due to change in exposure limits, increased use of diesel equipment. Similarly, for longwall coal mines in developed countries for the period between 1980's and 2010's, it can be estimated that the average coal mine air factor ranges from 0.4 m^3/s/ktpm to 1.5 m^3/s/ktpm. This air factor value may vary significantly depending on in-situ gas conditions and monthly production. For example; current Queensland coal mine air factor varies from 0.8 m^3/s/ktpm to 1.5 m^3/s/ktpm. Analyses of air factors of the first longwall coal mine in Queensland (Australia) operated for two decades had an average air factor of 1.5 m^3/s/ktpm as shown in Figure 7. Due to the difficulty in obtaining ventilation and production statistics, ascertaining a general coal or metal mine air factor was not possible. Regardless of production, each mine must be continuously ventilated to maintain the safe underground environment.

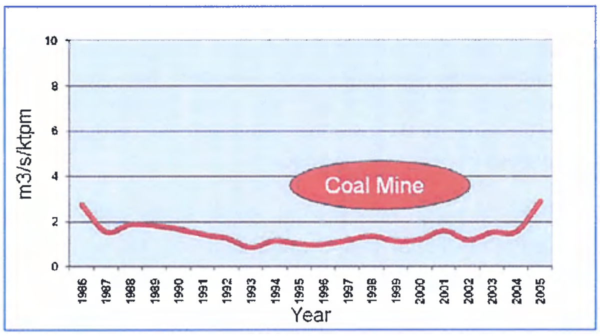

Figure 7. Timescale of ventilation demand in a coal mine.

2.1 Economic Airway Size Determination Parameters

In practice, the parameters that determine airway size are dependent on intended use of the airway, viz. exhaust or intake combined with any logistic requirements such as hoisting of ore or to accommodate equipment used underground. These airway sizes are designed in jointly with mining and most of the other engineering disciplines. The challenge of designing large shafts to accommodate equipment, rock and material hoisting and to pass large quantities of air safely and with minimal interference must be weighed against the sinking cost and equipping shafts. These factors continue to test the mining industry in striving to meet capital efficiency demands while minimising air power costs over the life of mine.

The total cost of owning and ventilating the airway is a combination of capital and operating costs (Barenbrug, 1963). Obviously, the most economical or optimum size vs. air velocity combination in an airway is achieved when that total cost of ownership is a minimum. The most economical combination is that which will yield the lowest total cost over the life of the shaft (Figure 8). Same principle can be employed to determine other main airways, but is not used during main airway designs. Other dominant factors that determine the main airway sizes are geo-technical design aspects, mining layouts, equipment used amongst others.

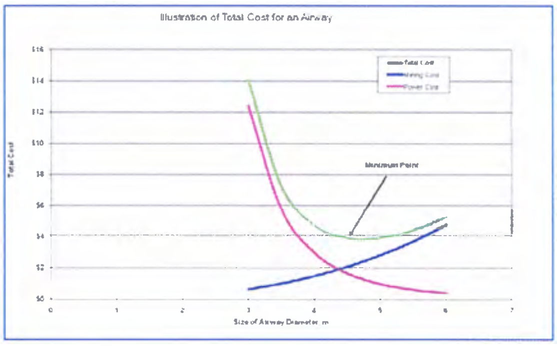
Figure 8. Relationship between airway size and total cost

In mine ventilation, for a fixed air quantity, large diameter shafts with large capital infrastructure usually results in lower fan power requirements, whilst small diameter shafts require large fan power because of the high aerodynamic resistance to air flow in the shaft (Figure 8). In the case of intake shafts used for men, materials and air conveyance, the development of airway size and cost relationship is challenging due to the complexity of allocating the cost of ore or man hoisting, power reticulation, water and/or ice distribution in the share of the costs.

In order to obtain the minimum cost of ownership it is necessary to possess a reliable and confident estimate of the annual cost of operating a shaft. It is common knowledge amongst the mining community that the costs of shafts vary considerably with diameter, length, method of sinking (raise bore, conventional blind sink), nature of ground, geographic locations, cost of labour, availability of skills, and demand for shaft sinking services from time to time. Therefore, this paper has attempted to develop a current relationship using the sinking cost data obtained from various projects and internal mining cost reports (Figure 8). These shafts were mostly from Southern African operations and few data from mines in Ireland and Australia. It must be noted that the reconciliation of initial project costs and final shaft costs cannot be found readily and it can be said that they are much higher than the original sinking costs.

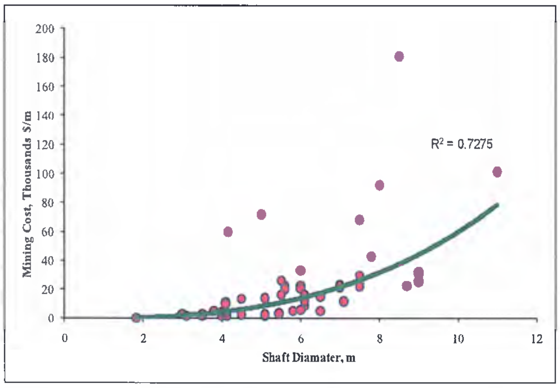

Figure 8: Relationship between sinking cost and diameter

The strong power trend association between the shaft sinking cost and the shaft diameter is explained with a positive correlation coefficient (r) of 0.85. The plot demonstrates that 72% of the total variation in sinking cost can be explained by the above power relationship. The key cost factors of other 28% of the total variation in mining cost are difficult pin down due to confidentiality and trade related issues. Attempt has been made to explain on 'unknown' factors that influence cost variations from the shaft sinking experts. Some possible explanations on 'unknown' factors that skew the cost-diameter relationship are type of contract on standing time, contingencies, possible intersection with large water bodies, number of shaft stations, availability of rigs and shaft sinking expertise and possibly differences in labour costs between countries and mining regions.

In order to ascertain possible attributes to the relationship between cost and the size of airway, parameters such as the commodity, shaft diameter size, shaft depth and geographic location was used. Association between the known variables on the shaft cost were evaluated using analysis of variance (ANOVA) using a statistical package on exhaust shaft sinking cost data. There is no conclusive relationship between individual stand-alone parameters, viz., shaft diameter, shaft depth, commodity type on mining cost. Statistical analyses have indicated that the interaction of two critical parameters, i.e., shaft diameter and depth has a significant influence ($p = 0.012$) on sinking costs that decide the optimum size of a shaft. Therefore shaft diameter and depth factors were used in the mining cost of the exhaust shaft size model.

The annual ventilation operating cost for an exhaust shaft can be expressed as a function of the following parameters (Barenbrug, 1963):

$$Voc = f(Ma,\rho,L,P,A,K,\$,\eta,\$m) \qquad (1)$$

Where,

Voc	= Ventilation operating cost
Ma	= Volume or mass of air conveyed
ρ	= Mean density of air in the shaft
L	= Length of shaft
P	= Perimeter of shaft
A	= Area of shaft
K	= Friction factor of airway
$	= Power cost per unit
η	= Fan-efficiency
$m	= Maintenance cost of fans

The techno-economic model developed indicates that the operating cost of moving fresh air through a deep mine from service shaft to exhaust shaft is currently about $2,100/m^3/s (Belle, 2005). An internal study investigated using a shaft size selection model with key elements that would influence the exhaust shaft sizes, viz. friction factor, life of mine, air quantity, shaft depth, electricity and mining cost. Some of the key inferences from modeling study (Belle, 2008) that are deemed relevant to the discussion are summarized below:

1. Influence of electricity cost: For a specific air quantity, the influence of an increase in electricity cost does not have any significant influence on optimum exhaust air velocity and can therefore, for all practical purposes, be taken as constant. The influence of electricity cost will be felt only when the increase in cost is more than five times the current price. It is noted that in the past few years, percentage increases in electricity costs are difficult to anticipate and these key assumption may skew the optimum size values derived from the model.

2. Influence of depth: The evaluation showed that there are no changes in the optimum up cast shaft air velocity and increase in depth of shaft. Shaft depths vary according to commodity type. For example, shaft depths of coal mines in Australia and South Africa can vary from 100 to a maximum depth of 400 m; gold mines go down to 4.0 km below collar, while diamond and platinum mines reach a depth of 800 m and 1500 m respectively. Due to the rock temperature gradients experienced in platinum mines, additional air above the critical depth (where the wet bulb temperature reaches the workplace design wet bulb temperature) would be required to manage the excessive heat at work place.

3. Friction factor (K): Friction factor is probably a most important factor that is often neglected during airway resistance calculations or simulations during ventilation planning. These values are obtained from past measurements based on specific type of established mine airway conditions. With changes in current mining practices, and size of equipments used in airways, it may be appropriate to review the application of these historic values. There have been examples whereby the larger exhaust shaft sizes were proposed by incorrectly using K factors in shaft size designs, i.e., K factor of 0.01 Ns^2/m^4 instead of 0.005 Ns^2/m^4. The financial impact and poor design of incorrect K values is large. Therefore, determining the realistic K factors from underground pressure-quantity surveys and using current data in determining the airway sizes is important for the future.

Use of correct value of K factor is not only essential to vertical shafts, but also are important to horizontal airways. For example, in coal or metal mines, some of the long back-return roadways are restricted for accessing on a regular basis. The roadway conditions may vary due to geological disturbances or unplanned accumulation of water. Those conditions pose a difficult planning challenge for the mine ventilation engineer in the correct use of K factor for determining the fan pressure differential. In such instances, a useful reminder for an operating mine is an age old practice of cleaning and maintaining the roadways (like perimeter or back return airways of longwall faces in coal mines) to reduce the airway resistance. Figure 9 shows an example in gold mine whereby the (800 m) roadway resistance was reduced from 0.448 Ns^2/m^8 to 0.0625 Ns^2/m^8 by cleaning and in effect reducing the pressure differentials by 720 Pa.

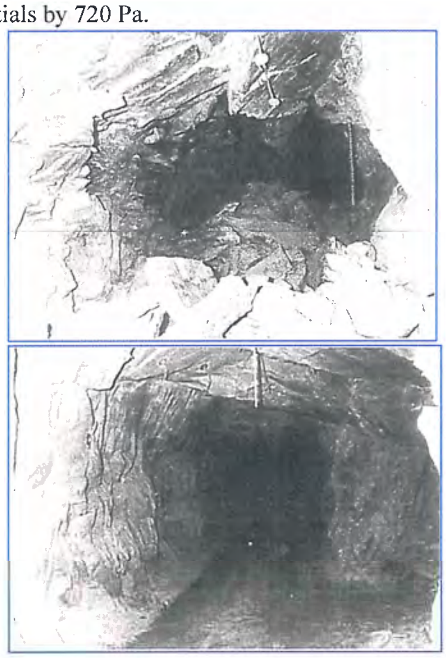

Figure 9: Effect of smooth walls (Top: before cleaning; Bottom: after cleaning; Greyling, J., 1978)

4. Life of mine: In practice, mine planning departments consider the LOM to be in the region of 20 years. However, there are ample mining examples whereby mine shafts have exceeded their LOM plans and have expanded their operations in excess of two decades. By keeping all the other parameters the same, for the typical LOM of 15 to 25 years, the optimum exhaust velocity remained the same. However, one needs to be careful on making typical assumption of LOM values like 25 years during design assumptions. Recently, a coal mine asset optimization exercise revised the proposed exhaust shaft size of 6.0 m to 4.5 m by revising the LOM parameter from 25 years to less than 8 years.
5. Shaft sinking cost: It has been noted that in the last few years, large demand for shaft sinking services and shortage of sinking expertise have resulted in a significant increase in sinking costs. This is one of the crucial parameter in shaft size selection model and cannot be found readily and experiences suggest that the final cost is much higher than the original sinking costs. The analyses have re-iterated that airways should be maximized for their air carrying capabilities.
6. Air quantity: The simulations using various air quantities with a fixed shaft size showed that the optimal air velocities follow a bi-modal trend for a specific range of air quantities, noting that power required increases with a cube relationship.

Currently, dynamic simulation packages like Ventsim Visual have the features to assist in optimum shaft size selection to assist in decision making using above parameters.

3 Main Airway Velocities: Design vs. Practice Discussions

This section of the paper discusses the operational experiences of airway velocities in particular shaft exhaust velocities and limited main airway velocities. Historically, velocities in intake and exhaust shafts are either prescribed by legislation to mitigate the effect of hazards on workers and equipment or past limitations based on experiences. In the case of vertical intake service shafts, high air velocities will affect dangerously the movement of conveyances past each other in the shaft and will invariably increase the pollutant concentration where rock hoisting takes place. The designs of intake service shafts have changed over time with the evolution from rectangular to more aerodynamically efficient and more structurally competent circular shafts. It was recognised in the 1950s that the friction factor and economic air velocities of a concrete lined ventilation shafts is advantageous when compared to rectangular shafts (Barenbrug, 1963).

In the 1950s, the planned exhaust shaft velocities of underground mines were in the region of 15.2 m/s with a shaft depth of 1500 m (Lambrechts and Deacon, 1962). However, some of the reported measurements have indicated a velocity range of 12 m/s to 17 m/s with the exception of one such shaft measuring a velocity of 7.5 m/s (Kroon, 1963), which was within the critical velocity range for water blanketing. The water blanket or droplet dancing effect as a result of air velocity in the critical zone of 7 m/s to 12 m/s was observed in the 1950's. At correct conditions, water blanketing could place the exhaust fan into stall zone, depending on the range of critical velocity lies on the fan curve. Under normal circumstances, the water runs down the shaft walls to the brow of the shaft; where it will be swept back up into the exhaust shaft at the right critical velocity range.

Another interesting out of the ordinary observation that was made in a coal mine (Y2010), where the water blanket or droplet dancing effect was observed even at exhaust velocity of 17 m/s (Figure 10) and resulting in fan operating closer to the stall zone. The problem was resolved by combination of pumping water out of the shaft bottom, water collection steel drain ring at the bottom of the exhaust shaft, and putting up fish net to reduce water re-entrainment by droplet impaction. This example suggests that the water blanketing effect can occur even outside the time honored critical velocity ranges between 7 m/s and 12 m/s.

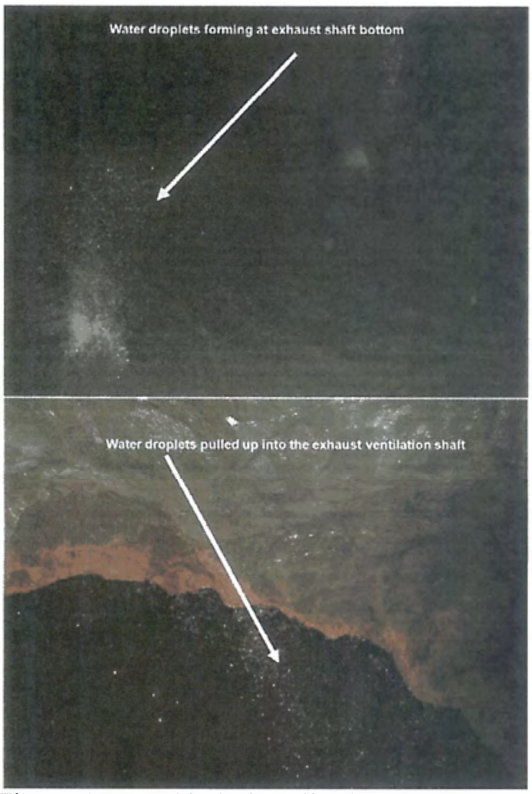

Figure 10. Water blanketing effect in a coal mine shaft velocity of 17 m/s

Figure 11 shows the histogram data relating to exhaust shaft air velocity acquired from different commodity mines from around the world.

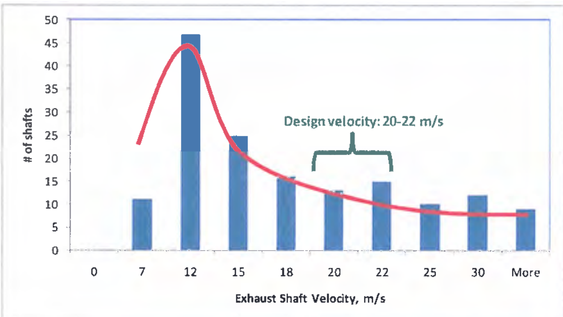

Figure 11. Global mine exhaust velocity data distribution

These exhaust shaft velocity data was collated from individual mines and by reviewing technical reports, ventilation proceedings, and specific project reviews. From the plot it is noted that a large number of these operations are in fact operating inside the critical velocity zone and outside of current design velocity of 20 to 22 m/s. This statistic provides us an opportunity to reflect on reality on ground and values currently used for such ventilation designs.

Figures 12to 15 shows the exhaust air velocity distribution profiles recorded in different commodity types, viz., platinum, thermal coal, gold and metallurgical coal.

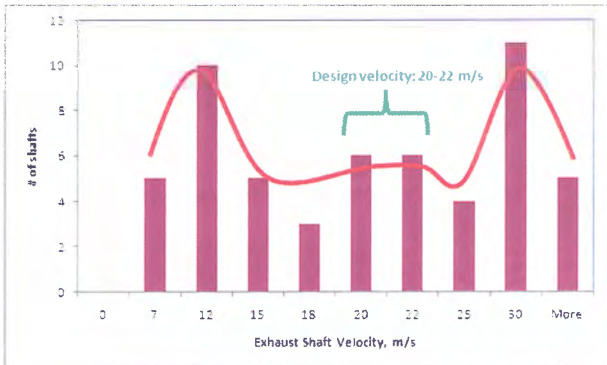

Figure 12. Platinum shaft exhaust velocity distribution

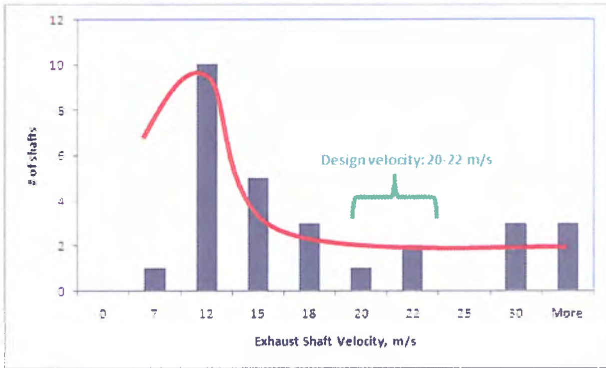

Figure 13. Thermal coal shaft exhaust velocity distribution

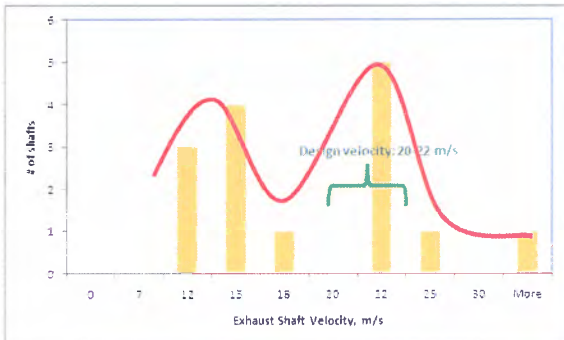

Figure 14. Gold mine shaft exhaust velocity distribution

For example, in platinum mines, nearly 40% of the shafts are operating at velocities greater than the design limit of 22 m/s which was unknown to the operations. The inferences from these values are that fans are possibly operating at lower efficiencies or lower system resistances than intended in the original ventilation designs. Another possible reason could be ascribed to the fact that the mining operations are not adhering to the original planned mining layouts.

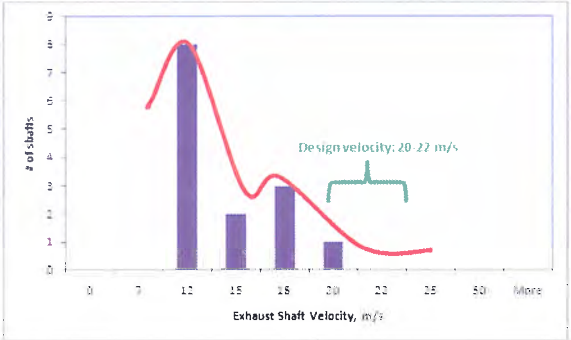

Figure 15: Metallurgical coal exhaust velocity distribution

As indicated in earlier sections of the paper, another commonly quoted ventilation design value is 4 m/s in conveyor road, face areas and intake airways to manage the physical discomfort of large dust particles striking the skin (although not a health hazard). The question often less debated is if these values still hold true based upon recent empirical data or studies or if workers ever likely to be using that particular travel road on a regular basis. As part of this paper, an attempt has been made to collate main airway velocities (excluding shafts) from few operating coal mines (Figure 16).

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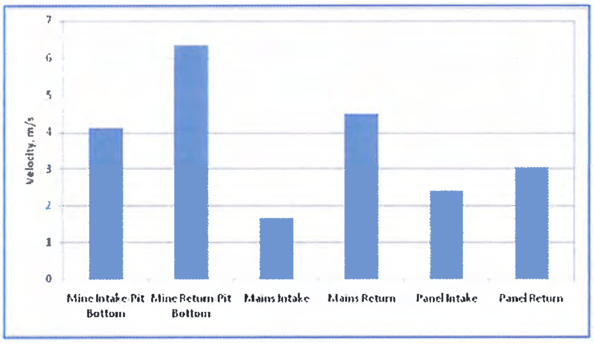

Figure 16: Typical main airway velocities in coal mines

At an operation level, these main airway velocities are important as they determine number of roadways in mains or panels for a design ventilation load capacity. No attempt has been made in this paper to determine optimum airway velocities in horizontal airways as done in the case of vertical exhaust shafts.

4 Conclusions

Globally, it can be estimated that mine ventilation systems from various commodities, circulate over 150 000 m^3/s of air through their ventilation shafts. The cost of these shafts is significant for natural resource companies. By revisiting the current design velocity for an optimum velocity should meet additional demands of increased infrastructure demands of operating mines. Shafts with additional air handling capabilities, due to revised ventilation design parameters (considering auto compression heat load), would provide additional development or production faces.

Although the practice of increased air velocity was not often 'known' before, based on the global exhaust shaft velocity data, a significant portion of shafts are operating outside the current design velocity value of 20 m/s (some up to 30 m/s) and in the 'forbidden' design critical velocity zone of 7 to 12 m/s. In most mines, data suggests that none of the mines have ever reached the LOM design exhaust shaft velocity value of 20 m/s. Regardless of what has been proposed as a standard design velocity, most of the times, the decision on shaft size is based on timely availability of rigs, changes to mining plans than originally proposed and modifications to it. In the analyses of this paper, cost of sinking is assumed to be remaining the same and in most cases this is unlikely and therefore the determined design velocity is of conservative value. As a way forward, the reasons attributed to such high velocities or low operating velocities and its potential impact in terms of energy, fan inefficiency and need for adherence to the planned mining layouts and re-visit of friction factor assumptions in ventilation network simulations is suggested.

With the fluctuations in the commodity boom and significant increase in capital costs, the design of mines need to be regularly reviewed, and if required, updated, to achieve the significant return on investment. With different ventilation 'ruling parameters' existing between mines, varied geographic locations, structurally different ore bodies and depth of mining contributing to the complexity, it is difficult to advocate standard ventilation design parameters related to infrastructures such as shafts. These observations are applicable not only to exhaust shafts but they are appropriate to underground airways such as intake, conveyor and return airway ventilation design air velocities. Similarly, as observed in Australian coal mines, the main fan pressure limitations of 4 kPa for minimizing spontaneous combustion risks needs to be re-visited. These views yet times prevent improvement opportunities of mine ventilation systems for improving underground occupational environment.

With changes in mode of worker transport and other changes in mining practices over the last three decades in most mining countries, a review of these historic main airway (horizontal and vertical) ventilation design standards is warranted. In author's views, such reviews will provide assurance to the appropriateness of using such values for future designs or development of new set of design velocities.

This paper does not recommend an increase in exhaust or other airway shaft velocities rather provides a case to consider review of airway velocity design values while providing safe and optimum ventilation infrastructures. Most common reasons for not considering such a review of design velocity are that they are time-honoured' design velocity values and cannot be changed, ignorance, mine fan pressure limits, possible spontaneous combustion, leakages or simply did not consider such a possibility. Other issues such as high pressure differential between intake and return airways on ventilation control devices (VCDs) and leakage aspects, worker discomfort need to be addressed during risk assessment if design velocity values are altered.

The paper was written in the hope that the future mines and expanding operations would challenge these ventilation design values and optimize mine designs in the most capital efficient manner possible. In conclusion, this paper has re-iterated the historic lessons that a safe and highly expensive ventilation shafts should not be under utilised by circulating too little air through it. It is noted that practical benefits of increasing air velocities may not be readily discernable for some years to come unless openness to discuss the views through field demonstrations.

5 Acknowledgements

The author is indebted to many local and international professionals, consultants, technical reviewers, and academic for their technical contributions and constructive criticisms to this paper in many forms.

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Real-time air velocity monitoring in mines - a quintessential design parameter for managing major mine health and safety hazards

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REAL-TIME AIR VELOCITY MONITORING IN MINES - A QUINTESSENTIAL DESIGN PARAMETER FOR MANAGING MAJOR MINE HEALTH AND SAFETY HAZARDS

Bharath Belle

ABSTRACT: Mines should be safe places in which to work. These safe places are achieved by means of natural and mechanical means of ventilation. Air velocity is a quintessential ventilation design parameter in diagnosing and ascertaining the adequacy of ventilation for managing mine health and safety hazards. Although Australian coal mines are recognized as being the safest mines in the world using both real-time and tube bundle monitoring systems, monitoring of airflow at critical locations in real-time is glaringly deficient and poor ventilation monitoring practice. This paper discusses the needs for real-time velocity monitoring and the implementation benefits of it in mines. What is an acceptable velocity measurement error in the carbon era? Current carbon emission guidelines do not clarify the measurement challenges associated with air velocities, let alone air velocity accuracy. Historically, there are references to acceptable measurement errors ranging from ± 5 % to ± 20 %. Measured differences in monthly ventilation surveys against the real-time airflow monitoring were found to be 13.3 % resulting in annual carbon costs of A$580 000 for a CH_4 level of 0.2%. It is considered that, it is never too late to implement real-time velocity monitors in Australian mines, a safety enabler and a leading practice in the mature mining world.

INTRODUCTION

Mines should be safe places for all those who work in them. These safe places are achieved by means of natural and mechanical means of ventilation. Air velocity is a quintessential ventilation design parameter in diagnosing adequacy of ventilation for managing major mine health and safety hazards. Use of minimum air velocity as a design parameter is an integral part of various ventilation engineering planning spheres to provide assurance on regulatory requirements as well as quality of hazard controls in some form in most mining countries. Therefore, monitoring of air velocity and in turn air flow in real-time is an essential practice in assuring continuous provision of safe occupational environment.

Issues of velocity measurement in mines have been studied by various research agencies including, Thimmons and Kohler (1985), AAC (1990), Hardcastle *et al.* (1991, 1993), Casten (1995), Martikainen *et al.* (2011). A notable study is the work of Thimmons and Kohler (1985). This work noted after a review of measurement practices that flow determination is more of an art than a science. This demonstrates that during velocity compliance determination, it is possible to introduce the operator bias, i.e., novice or veteran, instrument bias, sampling location, and frequency of measurements as required by the respective safety regulations. In such instances, real-time velocity monitors provide ventilation engineers with non-emotional data for evaluating the underground conditions and effectiveness of the mine ventilation systems. This paper attempts to explore the needs, challenges and operational aspects of implementation of real-time velocity systems. Benefits derived from installing real-time air velocity monitoring installation on main fan systems at a Bowen basin coal mine are discussed.

According to McPherson (2006), prior to the invention of vane anemometers (Figure 1) in the nineteenth century, the only practicable means of measuring rates of airflow in mines was to observe the velocity of visible dust or smoke particles suspended in the air. It is still a practiced method by the 'shift boss' or 'deputies' to estimate the air movement or direction of flow in the absence of real-time velocity monitoring instruments at hand by simply throwing some float dust found in the roadways to gauge the airflow and direction at very low air velocities, i.e., non detectable instrument measurement ranges.

Recent global catastrophic events in some form can be attributed to the outcome of inadequate ventilation, lack of ventilation (air velocity) monitoring thus creation of a flammable gas mixture and absence of dedicated long term mine ventilation engineers (unlike contractors) who are responsible for airflow and gas management underground. These hazards when unmanaged would be dangerously unforgiving and are mostly managed by better mine ventilation conditions. Figures 2 and 3 show the

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alarming fatal statistical consequence due to gas and dust explosions and frictional ignition potential in both gassy and non gassy mines.

Figure 1 - Friedrichs Anemometer, Model 1400 (0.3038 m/s to 20.3 m/s), (Source: MVS Journal, 1957)

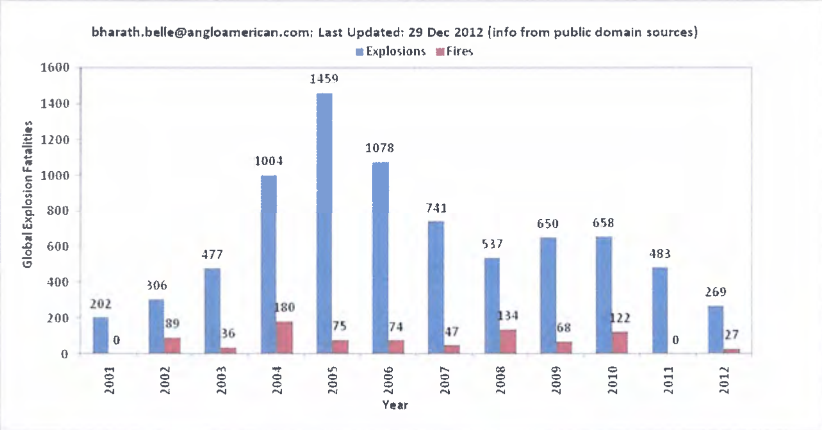

Figure 2 - Statistics on global mine explosions

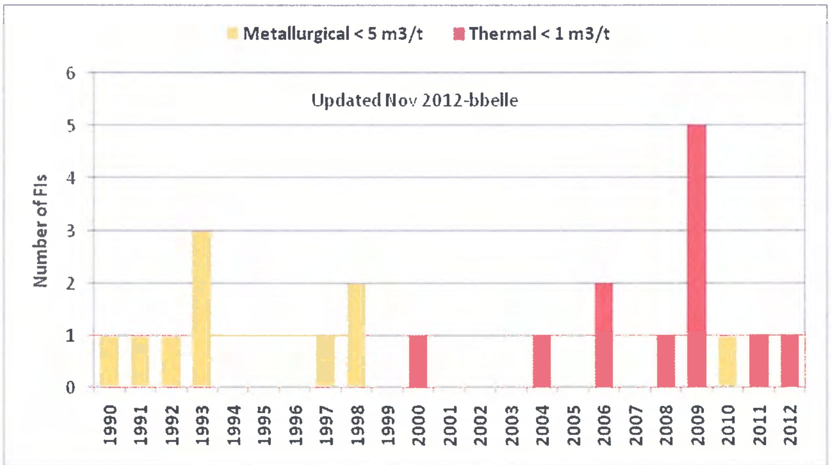

Figure 2 - FI incidents in gassy and non gassy mines (PS- reference to the use of FI limit of 5.75 m^3/t in PHMPs or COPs to be discouraged)

As seen above, explosion risks in coal mines are ever present because of inherent presence of methane gas. In order to minimize the risk profiles of these catastrophic events, it is timely that all interested parties in mines accept improvement opportunities in the following hierarchical control namely, air velocity (ventilation) monitoring:

- Accepting the need for continuous monitoring of hazards in the environment that is continuously changing (read gases and dust);

- Accepting the need for continuous monitoring of air velocity and ventilation that is continuously changing (read airflow) regardless of magnitude;
- Accepting that in a complex mine ventilation network, frequent manual ventilation monitoring in main returns or intakes is a cumbersome process and has practical limitations;
- Accepting the availability of Intrinsically Safe (IS) real-time monitoring tools for underground use in the current technologically advanced workplaces;
- Accepting that continuous air velocity monitoring devices can provide leading indicators of unanticipated conditions in the event of a failure or provide early warning of ventilation effectiveness or deficiencies;
- Accepting that traditional measurements aided by continuous monitoring would enhance the response time in the event of emergencies or re-entry;
- Accepting that approved IS real-time velocity monitors are available in Australia;
- Accepting that just as in other real-time gas monitoring tools, velocity monitors also need maintenance;
- Accepting that continuous velocity monitoring is a leading practice in other parts of the coal and metal mining world (UK, Canada, South Africa, and Poland);
- Accepting that improvements in velocity monitoring would assist the mines in controlling and providing improved quality of air;
- Accepting that real-time velocity monitor is a safety and production enabler.

ROLE OF AIR VELOCITY IN MINE DESIGNS AND REGULATIONS

Typical elements of occupational environment design are shown in Figure 4. These mining hazards resulting from natural and mining factors are managed by adequate mine ventilation using air velocity as a fundamental and quintessential design parameter. Air velocity expressed in metre per second (m/s) is the change of position and direction of moving air with time. Critical aspects that are considered in the design and planning of mine ventilation networks are air velocities and their direction in the working face, intake, return, tailgate, conveyor road, intake shafts, return shafts, main drifts, travel roads, haulage roads, longwall face, Last Through Road (LTR), over casts, bleeder road and regulators. These in turn with cross sectional area (m^2) would assist the mine ventilation engineer on the air flow rate (m^3/s) and in calculating the pressure differentials or calculating the efficiency of mine ventilation systems. Also, air velocity measurement along the maingate, mid-face and tailgate of a longwall enables the ventilation engineer to quantify the leakage of air into the goaf areas as well as estimate the heat loads and carry out in thermodynamic calculations.

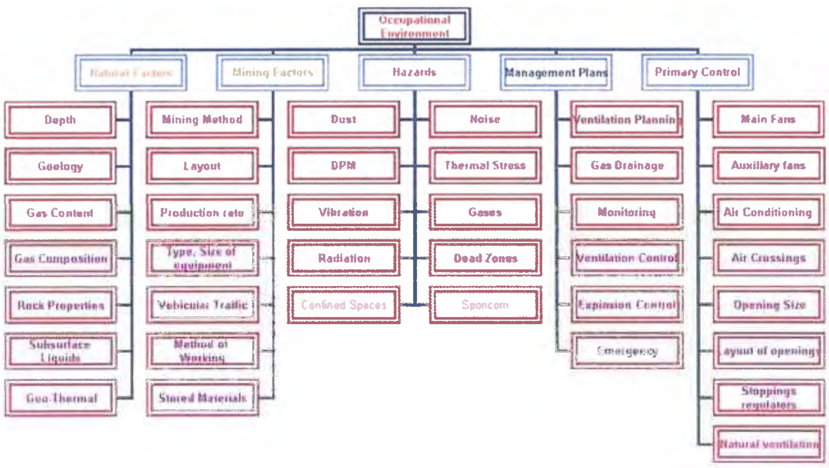

Figure 4 - Elements of occupational environment design

Typical ruling ventilation design parameters for various mineral types are shown in Figure 5. It must be noted that based on the place of operation (read continent), the ventilation ruling parameter may change due to the provision of minimum hazard limits at different commodities and mining countries.

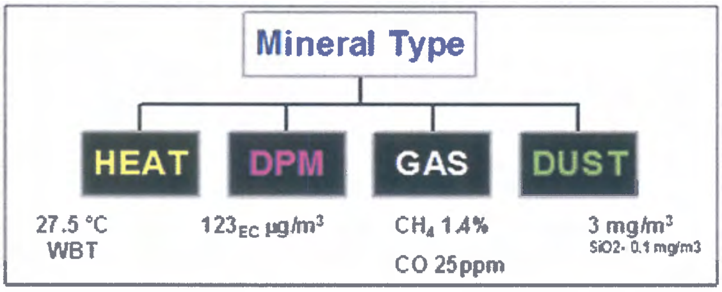

Figure 5 - Critical hazards in defining minimum air velocities

The methods considered for the minimum ventilation using air velocities for the working areas, viz.:

1) Ventilation for a minimum velocity (to dilute dust or gas or other identified hazards);
2) Ventilation for diesel engines to manage Diesel Particulate Matter (DPM) and gases;
3) Ventilation for heat management;
4) Ventilation for blasting re-entry time.

The normal design process is to calculate the air requirements using each of the above methods and to provide sufficient ventilation to meet the highest hazard management together with limit values prescribed in local regulations. This highest requirement is termed as the 'ruling parameter'. In some instances, this requirement is excessive or impractical and changes must be made to the mining or equipment parameters to reduce the ventilation requirements or cause another of the three parameters to become dominant.

Table 1 summarises an example of the air quantity requirements for the face ventilation system using various design criteria for identified hazards. The correct ventilation design factor used in the estimation can be debatable and the choice of factors is based on the individual operational experiences and the understanding of ventilation inflexibility needed during the life of the mine (LOM). In this example, measuring air velocity is the only means to ascertain the sufficient air quantities are supplied to manage the hazards, thus demonstrating the importance of measuring air velocities.

Table 1 - Example of determination of ventilation air quantity requirements

Condition	Design Criteria	Air Quantity	Leakage, 10%	Pressurisation, 15%	Required Air Quantity
		m^3/s	m^3/s	m^3/s	m^3/s
Legal-Min Std	$0.25\ m^3/s/m^2$	2.72			
Re-entry multi-blast development	30 min wait; 8 air changes; 60 m tunnel	2.90			
Re-entry-Secondary blasting	10 min wait; 8 air changes; 30 m to face	4.36			
Dust clearance*	1.0 m/s	10.89			
Diesel Engine-DPM	$0.0482\ m^3/s/kW$	6.884			
Diesel Engine-Avg. Heat**	$0.065\ m^3/s/kW$	9.23			

*based on type of dust and make [this example is for kimberlite dust for a block cave, incline cave and sub level cave mining methods, Belle (2005)] ** It is assumed that the average intake air WBT will not exceed 18° C

The following paragraph summarizes the example expressions of air velocity in the ventilation code of practice (COP) and legislations of mining intensive countries. These requirements illustrate that manual and or electronic means of real-time velocity monitoring devices would enable to provide assurance needed on meeting those compliance requirements.

- The QLD mine safety legislation requires that the Principal Hazard Management Pan (PHMP) must ensure that the ventilating air provided for the mine is of sufficient volume, velocity and quality to remove atmospheric contaminants from mining operations and maintain a healthy atmosphere at the mine during working hours. Also, it must ensure that the effective working temperature requirements are met. Effective temperatures are determined using wet bulb and dry bulb temperatures and air velocity. (Coal Mining Safety and Health Regulation 2001, Regulation 343-345)
- Controlled ventilation for a working place in each standing working place that is on the intake side of a working place and in each working place in an ERZ1 must provide for a ventilation current of an average velocity of at least 0.3 m/s measured across the cross-sectional area of the roadway in the working place. (Coal Mining Safety and Health Regulation 2001, Regulation 343-345)
- Mine safety legislation requires that in areas of the mine where persons work and travel, the ventilation system provides an average air velocity of at least 0·3 m/s measured across the work or travel area (Model Work Health and Safety (Mines) Regulations 2011 *Section 649)*
- The prescribed Chinese ventilation regulations, viz., minimum ventilation volume per person (4 m^3/min/person); decline travel airway velocity limit of 8.0 m/s; depending on location or activity a minimum ventilation velocity of 0.25-0.50 m/s and minimum diesel emission dilution factor of 0.06 m^3/s/kW.
- US regulation 30 CFR 75.350(b) limits belt air velocity to 5.08 m/s; 30 CFR 75.327(b) limits air velocity in trolley haulage entries to 1.27 m/s provided the methane content can be maintained below 1%.

Typically, ventilation systems are designed, implemented and monitored to manage the gaseous and particulate hazards. The following paragraphs reinforce the importance of 'air velocity' in mine ventilation designs and thus the need for accurate measurement requirements. Velocity values are widely published with accepted ventilation design standards on airways, viz., men and material shaft, dedicated intake shaft, exhaust shaft, travel road, conveyor road, working faces, main intake roadways, main return roadways (Jeppe, 1946; Lambrechts, 1974; Lambrechts and Howes, 1989; MVS Databook, 1999; McPherson, 2009). These proven or unproven 'design velocity' values (Table 2) have significant influence during mine planning in terms of main shaft and main airway sizes, number of roadways in mains or panels to carry certain design ventilation loads, e.g., six heading mains or eight heading mains, two heading roads or three heading roads in coal mines.

Table 2 - Typical ventilation design velocities (m/s)

Area	V1*	V2 (coal)**	V3** (metal)	Australian Guidelines***
Working faces	4	-	-	0.3
Conveyor drifts	5	5	5	
Main haulage routes	6	-	-	
Smooth lined mine airways	8	-	-	
Ventilation Shafts	20	18-22	18-22	
Decline Intakes	-	6-8	6-8	4-7
Dedicated Intake Shaft	-	18-22	18-22	
Downcast Shafts with hoisting	-	10-12	10-12	<10
Intake Airways	-	2-5	6-8	
Return Airways	-	3-5	6-8	
Overcasts	-	-	-	2-5
Auxiliary ventilated headings	-	-	-	0.5-0.75
Limit for safe pedestrian access+	-	-	-	<12

* McPherson (1984); Mousset-Jones (1986);** MVS Data Book (1999);***Draft
+If the second egress path is along the overcasts

Figure 6 show an example of a simulated ventilation model of an operating longwall coal mine with seven heading mains and an exhaust fan system and air velocities.

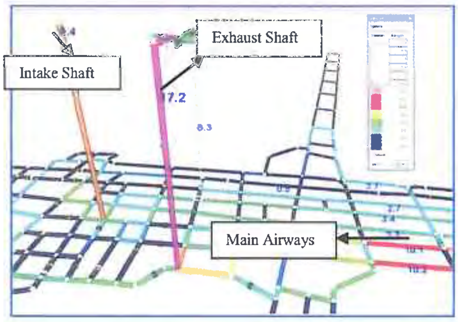

Figure 6 - Airway velocities of an operating gassy longwall coal mine

A summary of main airway velocities (excluding shafts) from few typical operating Australian coal mines is shown in Figure 7.

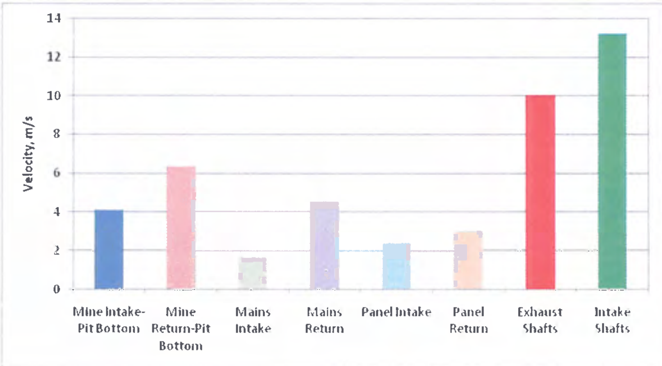

Figure 7 - Typical airway velocities in coal mines (Belle, 2012)

Amongst various air velocity design factors, another commonly quoted design air velocity is 4 m/s in conveyor road, face areas and intake airways. The basis for this value is to manage the physical discomfort of large dust particles (Figure 8) striking the skin (although not a health hazard) after McPherson (1984). The question is often less debated or questioned is if this conveyor road air velocity value still holds true based upon recent empirical data or studies based on increased daily production rates or speed of conveyor belts or if workers ever likely to be using that particular travel road (by walking) on a regular basis.

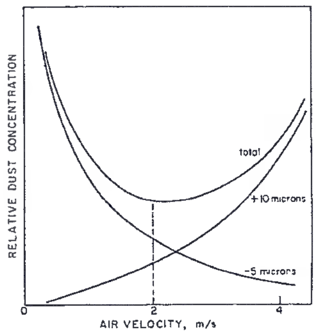

Figure 8 - Relationship between air velocity and relative dust concentrations (McPherson, 1984)

Real-time velocity monitoring leading practices

Air velocity is an indicator to monitor and control the hazards and is typically guided by design values. Traditionally, air velocity is monitored by a competent person (deputy or ventilation officer) by means of manual measurement tools such as vane anemometer by means of complete airway traverse or centerline measurements as required by the legislation or mine ventilation COP. However, in recent years, the need for electronic means of monitoring in real-time for management of hazards has become a reality. Use of real-time velocity monitoring device and by measuring the airway size, airflow rates or gas make are readily determined.

Various studies have been done in the recent years on the use of real-time velocity monitoring in mines by research institutions or instrument suppliers. Real-time ventilation monitoring in coal and metal mines is a leading practice worldwide probably started in UK mines (vortex based velocity monitors). For example, almost all collieries in South Africa have been using the real-time air velocity monitoring devices underground over at least three decades. One of the limitations of these real-time velocity monitors for use in Australian underground coal mines is the complex process of approval certificates by the respective legislative or testing authorities for use in underground mines. However, some of these IS real-time velocity monitors are approved in other mining (coal and metal) countries such as UK, Poland, South Africa, Canada.

Summary of practical benefits from real-time air velocity monitoring are:

- Continuous monitoring of the efficiency of the mine environment system and mine safety in the prevention of mine fires, spontaneous combustion and explosion events.
- Estimation of real-time carbon monoxide, methane and other noxious gas flow rates as an indicator for Trigger Action Response Plans (TARPs) in PHMPs.
- Estimation of gas emissions during panel development and longwall retreat.
- Accurate determination of heat loads and air cooling capacity for thermal hazard management.
- Improved confidence in Ventilation Air Methane (VAM) emission data.
- Estimation and reconciliation of specific methane emissions (SME) for longwall panels and mine emissions.
- Utilisation of real-time air velocity parameter/tag in the widely used Longwall Visual Analyses (LVA) tool.

In the case of Australian mines, monitoring of airflow underground at critical locations in real-time is not an accepted practice and reasons for its non-use are not documented. Anecdotal evidence indicates that perceived maintenance systems prohibit the pursuit of air velocity monitoring systems. Another commonly expressed reason is that the ventilation systems do not change frequently. Other debate typically diverts into the choice between real-time velocity or real-time pressure differential measurement which ultimately results in neither of the systems being considered. What has become noticeable is that most explosions or fire events have occurred in a smallest 'window' of change that occurred to the ventilation systems.

Figure 9 show the typical locations of real-time velocity monitoring in bord and pillar sections (left), underground velocity check using Kestrel (a digital anemometer) in low seam ~ 2 m (middle) and high seam (4.5 m) seam (right) coal mines in South Africa.

Figure 9 - Location of real-time velocity monitoring and correlation of underground installation in various mining heights in South African collieries

Real-time air velocity monitors in conjunction with CO, CH_4 and smoke sensors are typically placed at intake and return airways. At the beginning of the shift the Kestrels are calibrated against the known air velocities on the surface and later used underground to check if there are any significant deviations from the real-time vortex real-time velocity monitors. Typically the shift boss would calls the control room operator on surface and check the real-time air velocity readings for any significant deviations. Figure 10 shows the hand held Kestrel calibration on the surface, real-time vortex type air velocity sensor and mobile real-time monitoring sensor cluster of velocity, CH_4, CO and smoke sensors.

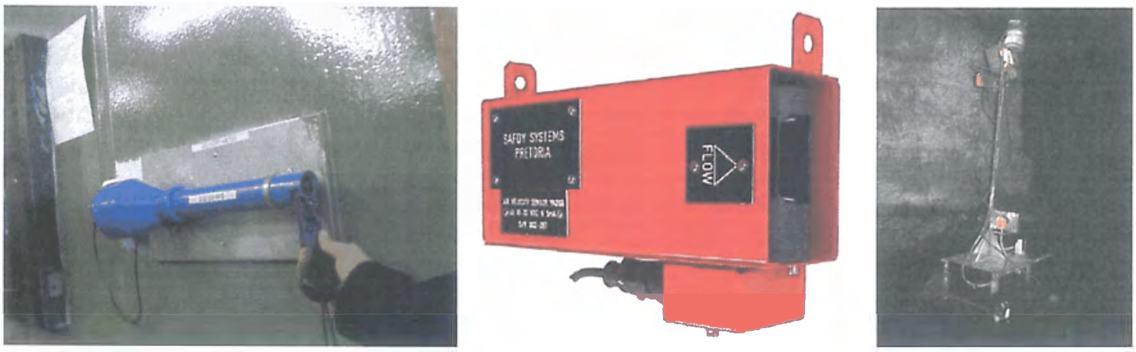

Figure 10 - Calibration of Kestrels on surface (left), vortex real-time velocity monitor (centre); CH_4, CO, smoke and velocity monitoring instrument cluster in section return (right) in South African collieries

ROLE OF VELOCITY MEASUREMENT - LW FACE AND MAIN FAN DUCT VELOCITY PROFILES - AN OBSERVATION

Velocity measurement is an quintessential activity in an underground mine to monitor the hazards on a daily basis by the deputies and ventilation officers. Application of air velocity is typically expanded to understanding the ventilation system effectiveness through velocity profiles like roadways, mine fan ducts and shafts. Velocity profiles are typically carried out to establish the velocity at different points along the longwall face or main fan ducts. Traditionally longwall face velocities of between 1.8 to 2.5 m/s have been considered optimum for longwall operations at conventional height. However, these values have evolved over time. Similarly, main fan duct velocity profiles would provide the main fan operational characteristics and air turbulence profiles in the main fan ducts.

Longwall face velocities

Figure 11 shows the LW face air velocities measured on two consecutive days by two different operators on three different longwall faces. Similarly Figure 12 shows the longwall main gate, panel intakes and return air velocities measured at three different longwalls. It is noted that they are also influenced by the location of shearer along the longwall face or if the shearer is operating. For example (Figure 11), for LW 3 (right), shearer was operational on day one (LW3-A); shearer was standing still at main gate chock ten and tail gate chock 160 on day two and three respectively.

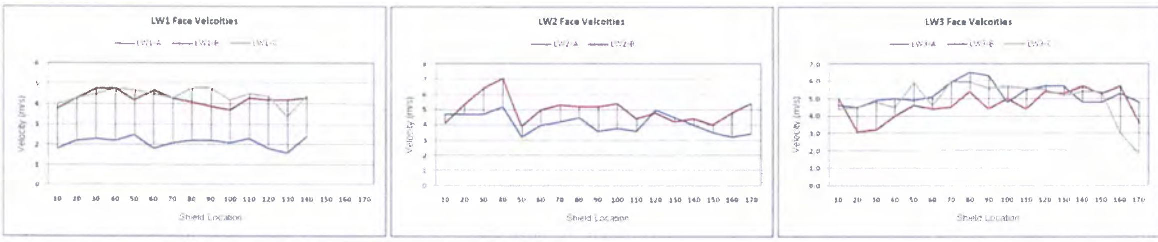

Figure 11 - Influence of manual velocity measurement by different operators in different longwall faces

Measured air velocities when the shearer was operational were higher than when it is standing still at TG or main gate position. However, it is not true for LW2, where the results were opposite to that of LW3. It is important to note that these measurements are typically measured using instruments such as Kestrels (not vane anemometers that are used by the ventilation engineer).

Figure 13 provides the longwall face velocity contours of measured longwall face air velocity data, viz., Chock 15 (top Left), Chock 75 (top middle), Chock 115 with shearer present (bottom Left) and Chock 135 (bottom right). As seen from these profiles based on air velocity measurements, these velocity contours can provide both a visual depiction of the air flow pattern and also a means of quantifying airflow. These profiles are useful to understand the possible location or presence of gas as well as possible scenarios for ventilation to leak into the goaf. Furthermore, it demonstrates the need to locate the daily velocity measurements taken by deputies throughout the industry. These air velocity readings along with the wet bulb temperature (WBT) and dry bulb temperature (DBT), which would provide the longwall effective temperatures and their status in relation to the TARPs.

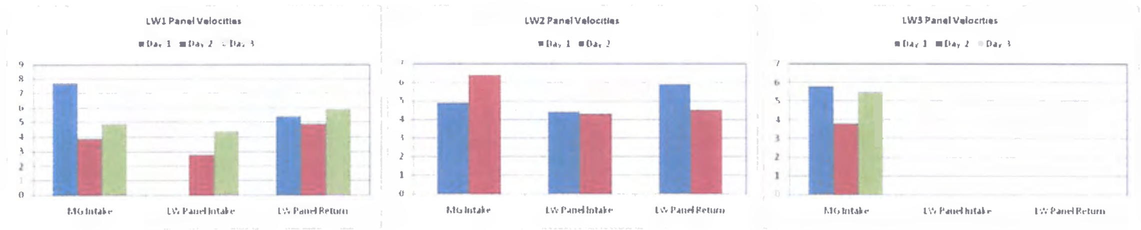

Figure 12 - Operator influences on measured air velocities along the longwall face

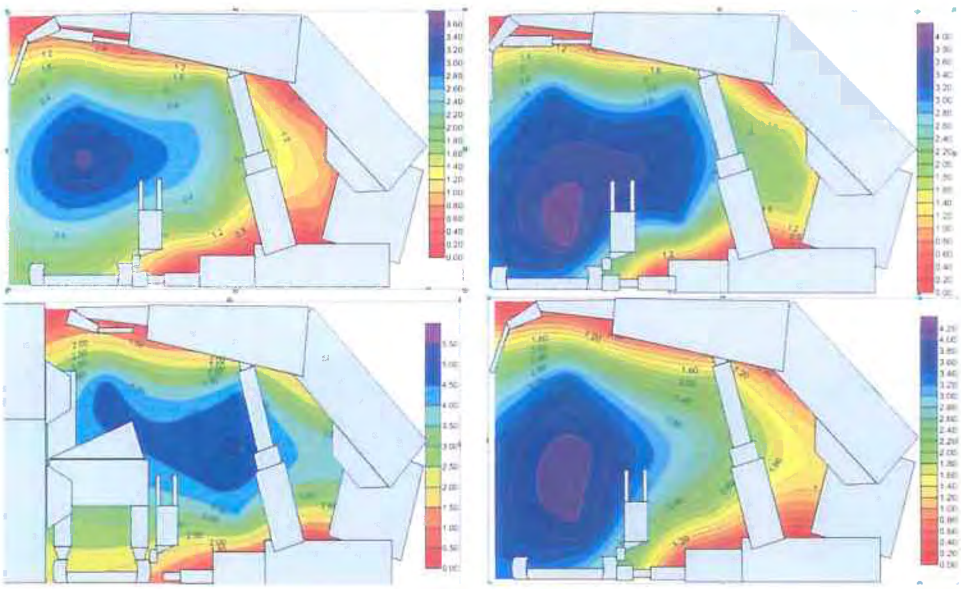

Figure 13 - Isovels along the longwall face maingate (top left), mid gate (top right), shearer (bottom left) and tailgate (bottom right) locations

These velocity profiles would enable the operator on the Frictional Ignition (FI) potential to ventilation leakage potential into the goaf area. In the prevention of FI, critical monitoring parameters of interest are methane, section or face air velocity, alarm settings of these monitors (Belle, *et al.*, 2012). In all or most of FI incident investigation reports it is noted that there was a failure to analyse the pre-ignition gas trends or velocity trends due to limited manual gas records or unconnected real-time gas recording and data collection system or velocity measurements. Improvement in collection of this crucial information is worth the effort for improved understanding and management of FI risks in the LW or development face areas. Therefore, real-time velocity monitoring installation along the longwall face would be a step in the right direction to the mining industry.

From measurement examples (longwall), it is noticed that without major changes to the ventilation flow, the differences in air velocity readings are significant despite each observer using the similar equipment and measurement techniques. These differences translate themselves onto some other parameters such as determining the effective temperatures for thermal stress or longwall panel gas make or longwall panel CO make. The resulting outputs further translate themselves onto the TARPs or longwall Specific Methane Emission (SME) models, monthly ventilation survey reports or review of simulations models such as Ventsim, or even during accident investigations on Frictional Ignitions (FI). Therefore the need to measure the air velocity beyond the statutory measurement location and their frequencies is increasingly becoming a practical reality.

Main fan duct velocity profiles

Figures 14 and 15 shows the isovels of main fan ducts measured from two different exhaust shafts (A and B) with a total seven different fans. These velocity profiles provide a graphical presentation of any issues that can be identified in main fan performance or turbulence associated with the shaft bend designs. What is valuable is that the velocity measurements derived from velocity pressure measurements provide the status of the fans or its future long term use. The isolvel plots suggest that they are definitely different to ideal velocity contours obtained in Computational Fluid Dynamic (CFD) simulations provide by main fan suppliers.

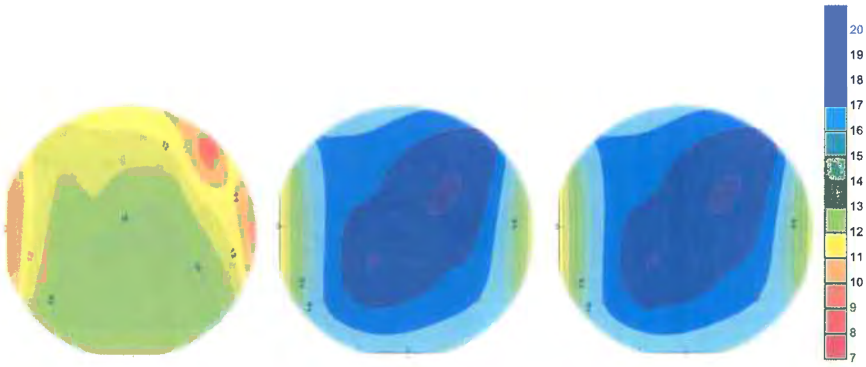

Figure 14 - Isovels measured at three different fan ducts from an exhaust shaft-A

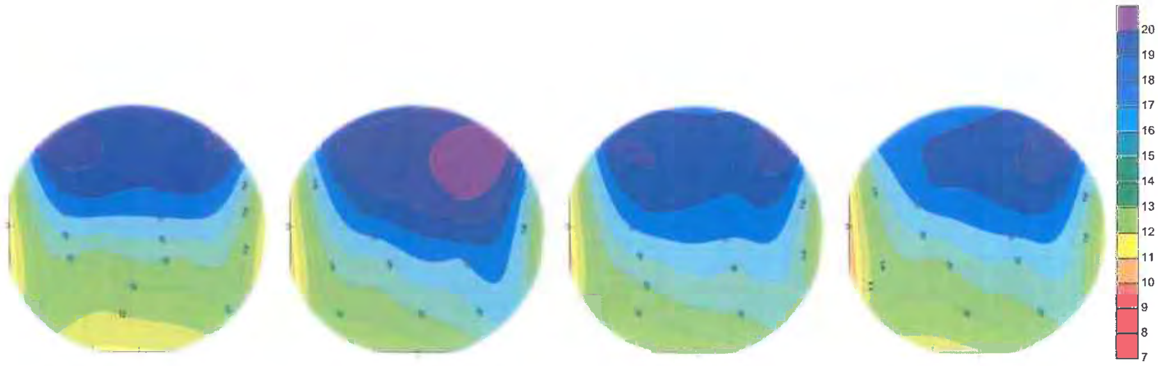

Figure 15 - Isovels measured at four different fan ducts from an exhaust shaft-B

There are several studies on the use of correction factors (including factory correction factors and their given range of velocities) in the literatures; its application in practice is remote. For example Thimmons and Kohler (1975) have suggested that the measurement should always be made at a minimum distance of three roadway diameters upstream of an obstruction and ten roadway diameters downstream of an obstruction. In reality, presence of these ideal locations is scarce or simply they do not exist. Measurement experiences suggest that each operation or a location underground or even the velocity contour profiles of a roadway which is dynamic is different and thus development or application of these correction factors are remote.

Critical measurement aspects that are commonly faced by the ventilation surveyors underground and during surface fan performance evaluations are:

1. Art of velocity measurement (years of experience u/g and correlating monthly ventilation reports to independent surveys);
2. Practical locations of velocity readings to be taken underground (high velocity turbulent regions or sharp bends);
3. Instruments used and their calibration on surface (kestrel or vane anemometers);
4. Underground environment conditions (humid and dusty vs. comfortable conditions);

5. Time constraints and understanding of significance of 'velocity values' to be used after the ventilation surveys.

Above pragmatic measurement challenges offer the users the benefits of fixed real-time velocity monitoring systems to minimise various operator (human) errors identified above.

Australian experiences of real-time velocity monitoring on main fans

With no means of measuring emissions from the mine in real-time and without compromising current mine monitoring systems dedicated for mine safety, specifically sponcom, explosion prevention and management of thermal stress, the need for dedicated real-time airflow monitoring at strategic underground locations is quintessential. Typically, during most shifts various measurements are taken and ideally these are analysed for suitable trends. These trends at times may identify the deficiencies in controls or measurement errors. For example, analyses of recorded underground data on temperature measurements and associated air velocities suggested that LW mid face temperatures were higher than the tailgate temperatures with constant ventilation flow. Later the measurement bias was rectified through toolbox talk, whereby temperatures and air velocity were taken at the same time on each shift and at the same location on a consistent basis. These accurate data are typically used to evaluate the performance and effectiveness of the mine cooling systems.

Figure 16 shows possible location for real-time velocity monitoring system in an underground drift which is a common practice in overseas mines.

Figure 16 - Suitable location for real-time air velocity measurements to carry out performance evaluation of Bulk Air Cooler (BAC)

In recent past, the introduction of a carbon price on Green House Gas (GHG) emission has further necessitated the need for accurate airflow data from mine exhaust systems. The biggest variable in the carbon emission is the airflow. Most mines have established the emission inventory using the existing manual ventilation measurement practices in accordance with the obligations of the National Greenhouse and Energy Reporting Scheme (NGERS) Act (2007). The NGERS Act underpins the Carbon Pricing Mechanism which was introduced on 1st July 2012. The mine Ventilation Air Methane (VAM) is a significant constituent (over 70%) of past, current and future underground carbon emissions.

A significant opportunity exists in Australian coal mines to build a robust, compliant, accurate and transparent VAM reporting system through improved real-time airflow monitoring systems instead of the current user of manual monthly ventilation surveys. Both internal and later external VAM compliance audits have identified the need for a paradigm shift in VAM monitoring systems in terms of resolution and frequency of measurement of key data components. The common findings from most carbon audits is that the current single monthly ventilation survey data for VAM estimation is deemed as a 'Potential Risk of Non-Compliance' due to the materiality of 'Run-of-Mine Coal Extracted from Gassy Underground Mine' emissions. Typically any changes in ventilation system (such as slowing down of fans or maintenance of a single fan or brief power failures) or errors associated with the ventilation measurement are not captured in the estimated carbon emissions. This is because the monthly ventilation surveys do not capture them. For example, 400 m^3/s of airflow with 0.3% methane, 10% changes in airflow alone would relate to additional carbon tax of AUD$1.4 million per annum. Similarly, a 5% error in manual measurement flow at 0.36% methane over a 5 year period would have an emission cost of AUD$10.9 million at carbon price of AUD$23. Acknowledging these significant costs, the VAM monitoring system considered by mines is typically independent of current systems which are dedicated to mine safety.

As a proactive approach, most mines are implementing the underground use of approved IS ultrasonic flow monitoring devices at the exhaust shaft fan ducts. It is also noted that a handful of coal mines are in the process of implementing these real-time monitors underground. Current installation of monitoring systems at exhaust shaft fan ducts or underground shaft bottoms incorporate independent measurement of real-time exhaust airflow, CH_4, CO_2, temperature (WBT and DBT), moisture and pressure to improve VAM measurement accuracy which is a largest variable in the VAM greenhouse gas estimates. The introduction of leading practice of real-time monitoring of airflow and low range gas measurements at fan ducts will enable mines to produce transparent emission reports and also enable immunisation from carbon tax estimation errors. Recently, these systems have been implemented at mines in NSW, Moranbah North Mine and are also proposed to the new Grosvenor mine.

Figure 17 shows the implementation of real-time ultrasonic air velocity monitoring system installed on main fan ducts. Figure 18 demonstrates the daily real-time air flows measured at individual fans (for a period of 34 d) and the average airflow measured over a period of 5 months. The average airflow measured from underground surveys over a period of five months was 253 m^3/s. Similarly, the average flow recorded using real-time air velocity for a period of 32 d was 219.78 m^3/s with a difference of 13.25 %. Other benefits of obtaining the air velocity trends from each fans is to evaluate individual fan performance against the planned airflow in mine ventilation designs and ventilation simulation models. In this example, it is easily noticeable that Fan A is significantly different than the other two main fans. In addition, the impact of airflow measurements is significant on the estimation of greenhouse gases as well as associated annual costs in the region of $580 000 for a methane concentration level of 0.2%. Table 3 provides the carbon costs associated with variations in measured velocities and methane concentrations.

Figure 17 - Installation of real-time air velocity monitoring on main fan ducts

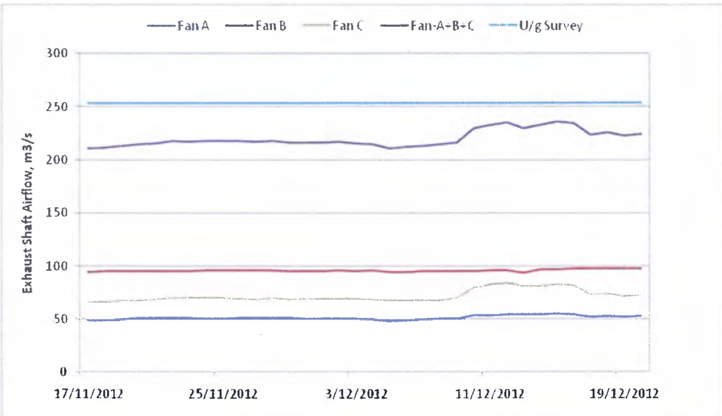

Figure 18 - Differences between real-time air velocity monitoring on exhaust shaft and u/g ventilation surveys

Another significant parameter that is used in determining the airflow is the area of a roadway. Typically, 5% is considered to be an acceptable error during underground airway measurement survey. Even with this low level of acceptable error the carbon cost is significant, i.e., at 0.2% methane level for a roadway area of 20.30 m^2, 5% accepted error in airway area (m^2) would be costing around ±$200 000 per annum (Table 3).

Table 3 - Cost of carbon with variation in methane levels and various accuracy levels on measured air velocity

Air Velocity m/s	8	8.5	9	9.5	10	10.5	11	11.5	12
CH4 %	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
0.025	$ 390,560	$ 415,000	$ 439,411	$ 463,823	$ 488,235	$ 512,647	$ 537,058	$ 561,470	$ 585,882
0.05	$ 781,176	$ 829,999	$ 878,823	$ 927,646	$ 976,470	$1,025,293	$1,074,117	$1,122,940	$1,171,763
0.075	$ 1,171,763	$1,244,999	$1,318,234	$1,391,469	$1,464,704	$1,537,940	$1,611,175	$1,684,410	$1,757,645
0.1	$ 1,562,351	$1,659,998	$1,757,645	$1,855,292	$1,952,939	$2,050,586	$2,148,233	$2,245,880	$2,343,527
0.125	$ 1,952,939	$2,074,998	$2,197,057	$2,319,115	$2,441,174	$2,563,233	$2,685,291	$2,807,350	$2,929,409
0.15	$ 2,343,527	$2,489,997	$2,636,468	$2,782,938	$2,929,409	$3,075,879	$3,222,350	$3,368,820	$3,515,290
0.175	$ 2,734,115	$2,904,997	$3,075,879	$3,246,761	$3,417,644	$3,588,526	$3,759,408	$3,930,290	$4,101,172
0.2	$ 3,124,703	$3,319,997	$3,515,290	$3,710,584	$3,905,878	$4,101,172	$4,296,466	$4,491,760	$4,687,054
0.225	$ 3,515,290	$3,734,996	$3,954,702	$4,174,407	$4,394,113	$4,613,819	$4,833,524	$5,053,230	$5,272,936
0.25	$ 3,905,878	$4,149,996	$4,394,113	$4,638,230	$4,882,348	$5,126,465	$5,370,583	$5,614,700	$5,858,817

What is an acceptable velocity measurement error in the carbon era? Current guidelines do not necessarily clarify the measurement challenges associated with air velocities, let alone measurement air velocity accuracy. Historically, there are few references to acceptable measurement errors. Thimmons and Kohler (1985) have expressed the definitions on accuracy requirements for mine ventilation applications. They had expressed the accuracy of ± 20 % is satisfactory based on the ventilation measurement practices of 1970s. Also, recently, there are suggestion of ± 5 % error value that is viewed as an acceptable air velocity measurement error in an underground mine (Martikainen, *et al.*, 2011). Considering the recent financial impacts, lack of a standard on an acceptable measureable error persists and in addition, which velocity measurement instrument to be seen as a 'reference true velocity measurement device' to estimate the accuracy of a velocity measurement device needs to be established by the mining industry.

ENVIRONMENTAL MONITORING SYSTEM, MAINTENANCE DILEMMA AND IMPLEMENTATION BENEFITS

Continuous on-line velocity monitoring systems will facilitate the establishment and maintenance of a safe environment underground if well installed, maintained and monitored. Such a system will give early warning of a fire, spontaneous combustion heating, abnormal methane or carbon monoxide gas concentrations and a failure or weakening of the air flow. Prompt response can then be taken to deal safely with the abnormal situation provided controls are in force and manageable. Interested parties in determining the real-time velocity monitoring strategy are the ventilation officer, mine manager assisted by the mechanical and electrical engineering manager. Most of the Australian coal mines incorporate CH_4, CO, CO_2, and O_2, barometric pressure monitoring systems stationed at strategic locations along the intake, return and longwall face and on the surface. The monitoring of air velocity at strategic positions will indicate the status of the air distribution in the mine on a continuous basis. For example, the ventilation and heat simulation software tools like Ventsim Visual have the facility to incorporate real-time velocity tags for live simulations. The real-time velocity monitors will give early warning of a weakening in airflow or a ventilation failure. It will also indicate a weakening trend in airflow and action can therefore be taken before a gas accumulation develops. Benefits of real-time velocity monitors will provide ventilation engineers additional information on whether an increase in gas levels is due to increase in gas release rate or reduced ventilation.

Just as in real-time and tube bundle environmental monitoring systems, the maintenance of real-time velocity monitoring system is of vital importance. Confidence in the system will be lost if the system is not maintained and kept in a fully operational condition as in other real-time measurement parameters such as CH_4, CO, O_2 sensors. All existing real-time and tube bundle systems require adequate maintenance as per the Australian Standard (AS) 2290.3. Failure to address this will lead to misinterpretation of conditions underground and should be addressed without delay by relevant person responsible for the installation and maintenance of the monitoring systems. As in the case of existing environmental monitoring systems, the inspection should include provision for frequency of cleaning of monitors, testing of response of monitors, replacing malfunctioning monitors, a documentation system to include installation, cleaning, testing and replacement dates. As in the case of existing environmental monitoring systems, air velocity monitors must be provided with battery back-up power which must switch on automatically in the event of a power failure.

CONCLUSIONS

The monitoring of air velocity at strategic positions will indicate the status of the air distribution in the mine on a continuous basis. The velocity monitors will give early warning of a weakening in airflow or a ventilation failure. It will also indicate a weakening trend in airflow and action can therefore be taken before a gas accumulation develops. Benefits of real-time velocity monitors will provide the ventilation engineers additional information on whether the increase in gas levels is due to increase in gas release rate or reduced ventilation.

Use of real-time air velocity monitoring technology underground can aid the ventilation engineers, longwall operators, technical services managers, safety officers and emergency response personnel for any unanticipated surprises on gas or ventilation situation and develop speedy interventions and thereby reduce production downtime. Therefore, monitoring of air flow in real-time is essential and is a leading practice in evaluating the performance of underground environment conditions and must be pursued by the Australian mining industry into the next decade.

Air velocity and area of a roadway, and wet bulb temperatures (WBT) and dry bulb temperatures (DBT), CH_4, CO_2, barometric pressure are the key parameters that will assist in understanding the key hazards (gas, dust, sponcom, thermal), associated risks and the effectiveness of controls provided at the workplace. Therefore, it is important that these parameters are accurately measured by those who are responsible for them.

It is hoped that the implementation of real-time velocity monitors that are glaringly absent in the Australian coal mines that have one of the best gas monitoring systems would consider this improvement opportunity to clear out any distractive comments or criticisms on Australian Safety and Health Systems. In author's opinion, it is never too late to implement the real-time velocity monitors in mines, a life saving safety enabler and a leading practice that exists in the rest of the coal mining countries.

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Mine ventilation design velocity standards for underground mines – mine operators' perspectives

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ABSTRACT: It can be argued that the phrase 'air velocity' is the cornerstone of mine ventilation system designs based on standards almost half a century old. Despite it determining major ventilation airway sizes and their number at the underground mine planning stage, little attention is given to the 'velocity standard values' and their applicability to current mining practices. In the absence of universally rational 'air velocity standards', the authors of this paper have provided empirical evidence that supports and challenges the use of existing air velocity standards for optimal future designs. Analyses of their background suggest that these 'air velocity' standards owe their origins to historical adoption (Mine Ventilation Society of South Africa and other ventilation text books) while some can be related to controlling airborne respirable dust, or to studies of mine fires involving conveyor belts in the laboratory. What was missing in most of the design expert review documents (leading or misleading) was the rationale for using these air velocities and also lack of reference to 'no go' values. For example, is it 8m/s or 10m/s or 12m/s in main returns or is it 6m/s for main intakes and panel intakes or main declines?

This paper explores the operational approach to 'maximum air velocity values' combined with a 'safety factor' that is to be used in the mine ventilation designs and provides guidance on its use. These reviewed 'design velocity standards' are based on operational experience and would benefit ventilation officers and others responsible for ventilation control devices (VCDs) and for performing ventilation surveys. The fundamental standard velocity design values are critical to determine the size of airways, number of airways, shaft sizes, the operational specifications of main fans, mine cooling systems, and finally the operating and capital costs of ventilation systems. The authors have jointly shared data on actual operational air velocities in main intakes, return airways, working panels of coal (bord-and-pillar and longwall), gold, platinum, diamond and base metal mine development and production sections. It is intended that these design values will provide adequate 'safety factors' to those miners, ventilation officers or deputies who are responsible for the use, operation and upkeep of critical airways and ventilation planning engineers who might use these as guidance instead of 'random values' as contained in various ventilation studies, for the benefit of future mine ventilation designs.

1 INTRODUCTION

Mine ventilation practices have their very early roots in the fires started at the bottom of up-cast airways to induce airflow through the underground workings, as used by the Greeks and Romans to the layouts described in the works of Georgius Agricola who in his book "De Re Metallica," published in 1556, demonstrated pictorially these and other ventilating methods. The mine ventilation system involves supply, control of air and its movement to meet health and safety standards and provide adequate working conditions for underground personnel. Mine ventilation is a strategic component of any underground mining operation.

With the rising awareness of new hazards and their stringent safe limit values, ventilation infrastructure and designs must have the capability and flexibility to handle any unanticipated capacity shortfall often linked to unforeseen increases in production rates. In addition to those associated with the modernisation of mining methods, the typical health and safety hazards found in mines are gases, dust, heat, ionising radiation and diesel particulate matter (DPM). Mining depth and its associated health and safety hazards vary between commodity types.

The impact of advances in production technology on mining requires maintaining a healthy and safe occupational environment that is also cost effective. However, the increasing imperative for increased system efficiency and effectiveness makes the use of standards devised decades ago inadequate and at times plain risky. Based on observations and interactions with mining and ventilation professionals globally, it is noted that despite all the achievements, there are still opportunities for optimising mine ventilation systems.

Mining hazards resulting from natural and mining conditions are generally managed by adequate mine ventilation that utilises air velocity as a fundamental and quintessential design parameter. Air velocity (or, more correctly, air speed- a scalar value) expressed in meters per second (m/s) indicates how rapidly the general body of an air current flows through a mine excavation (airway). Critical aspects that are considered in the design and planning of mine ventilation networks are air velocities and their localised direction in the working face, intake, return, tailgate, conveyor road, intake shafts, return shafts, main drifts, travel roads, haulage roads, longwall faces and stoping panels, last through roads (LTRs), overcasts, bleeder roads and air regulators. In turn, consideration of the excavation's cross sectional area (m2) yields the air flow rate (m3/s) through it and is also used in calculating the pressure differentials and hence the efficiency of mine ventilation systems. The calculation of an air utilisation index (AUI) is also a popular tool used by some ventilation engineers to assess the fraction of the total air quantity supplied to a particular workplace e.g. a stope in a narrow reef mines. Also, air velocity measurement along the maingate, mid-face and tailgate of a coal longwall enables the ventilation engineer to quantify the leakage of air into the goaf areas as well as estimate the heat loads and carry out in thermodynamic calculations.

The methods considered to determine the minimum ventilation requirements are based on air velocities to meet the needs of various operational requirements, viz.:

(1) to dilute dust or gas or other identified hazards encountered under normal mining operations
(2) to dilute (and manage) dust, gases and particulates emitted by diesel engines to manage DPM and gases.
(3) Provide adequate heat sink capacity for heat management
(4) Provide a safe and viable blast re-entry time

Where a number of these requirements coexist simultaneously, the normal design process determines the air requirements for each of the above requirements, and providing sufficient air volumes to manage requirements for the highest hazard (often in line with limit values prescribed in local regulations). This highest requirement is termed as the 'ruling parameter'. In some instances, the ruling parameter is excessive or impractical and changes must be made to the mining or equipment specifications to normalise the ventilation requirements or cause another of the four operational requirements to become dominant.

Table 1 summarises an example of the air quantity requirements for the face ventilation system using various design criteria for identified hazards for a typical 3.3m x 3.3m development heading. The final ventilation design factor selection in the estimation is debatable as it would be based on the choice of factors linked to the specific operational experiences and the attainment of ventilation system flexibility needed during the life of the mine (LOM). In this example, measuring air velocity is the only means to ascertain that sufficient air quantities are supplied to manage the hazards, and demonstrates the importance of doing this.

Table 1: Example of determination of ventilation requirements

Condition	Design Criteria	Air Quantity, m3/s
Legal-Min Std.	$0.25\ m^3/s/m^2$	2.72
Re-entry multi-blast development	30 minute wait; 8 air changes; 60 m tunnel	2.90
Re-entry-Secondary blasting	10 minute wait; 8 air changes; 30 m to face	4.36
Dust clearance*	1.0 m/s	10.89
Diesel Engine-DPM	0.0482 $m^3/s/kW$	6.884
Diesel Engine-Avg. Heat**	0.065 $m^3/s/kW$	9.23

*based on type of dust and make [this example is for kimberlitic dust for a block cave, incline cave and sub level cave mining methods, Belle (2005)] ** It is assumed that the average intake air wet bulb temperature (WBT) will not exceed local conditions of 18°C.

At present, most of the global mine ventilation planning and designs make use of recommended airway velocities (Table 2) based on historic studies, empirical data and experiences. These values are often reflected in internal mine design standard documents or project design reports and internal project review guidelines. Another important recommended design standard is the

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velocity range of 7 m/s to 12 m/s, which is known as critical velocity zone to be avoided in wet exhaust shafts to prevent water blanketing. In practice, regardless of the wetness condition, this air velocity range is often applied stringently in design calculations.

The need for the revision of time honoured ventilation design values for mining and ventilation engineers globally has been raised elsewhere (Belle, 2011), and is also part of general discourse on cost efficiency. These ventilation design criteria have significant influence during mine planning for example in terms of main shaft and main airway sizes, number of roadways in mains or panels to carry certain design ventilation loads, e.g., 5 or 6 or 7 heading mains in coal mines; 2 or 3 heading longwall panels in coal mines. In recent years, it was not common to find collated information on typical air velocities for different mining configurations (commodities). In the midst of these guidance values, mine designs are made using historic values. There are widely published and accepted ventilation design standards on airway velocities, viz., men and material shaft, dedicated intake shaft, exhaust shaft, travel road, conveyor road, working faces, main intake roadways, main return roadways (Jeppe, 1946; Lambrechts, 1974, Lambrechts and Howes, 1989; MVS Databook, 1999; McPherson, 1984, 2009).

Key air velocities in main airways that carry the ventilation load are mine intake shafts, travel and conveyor drifts, mine return airways, main intake airways and main return airways. Based on these historic recommended maximum airway velocities, mining designs will often determine the dimensions and number of mains and shaft sizes for either expanding operations or new projects.

Table 2. Recommended maximum velocities (m/s).

Area	V1*	V2 (coal)**	V3** (metal)
Working faces	4	-	-
Conveyor drifts	5	5	5
Main haulage routes	6	-	-
Smooth lined mine airways	8	-	-
Ventilation Shafts	20	18-22	18-22
Decline Intakes	-	6-8	6-8
Dedicated Intake Shaft	-	18-22	18-22
Downcast Shafts	-	10-12	10-12
Intake Airways	-	2-5	6-8
Return Airways	-	3-5	6-8

* McPherson (1984); Mousset-Jones (1986)
** MVS Data Book (1999)

From a due diligence perspective, there have been few documented views expressed on these optimum exhaust air velocity design values. The only references that the author can trace exhaust shaft velocity of 20 m/s and other maximum air velocities is found in technical paper titled "The mine ventilation planning in the 80's" by Prof. McPherson (1984) and the MVSSA Data Book (1999).

In many operations globally, maximum velocities in airways have now become embedded in design codes and any suggestion of velocities higher than the guidance maximum velocities are often seen with extreme caution or dismissal. Often, this intransigence simply results in missed improvement opportunities. However, velocities higher than recommended operating velocities are not new to the mine ventilation fraternity and can be found in mine ventilation networks. This paper indicates that in many current practices in different mining commodities despite there being defined air velocity criteria, use is made of 'safety factor' values – irrespective of an inherent need for revision of ventilation design values.

2 VARIATION OF VENTILATION DEMANDS IN VARIOUS MINING COMMODITIES

It can be demonstrated that current airway sizes are primarily dependent on ventilation and cooling requirements as shown in Figure 1.

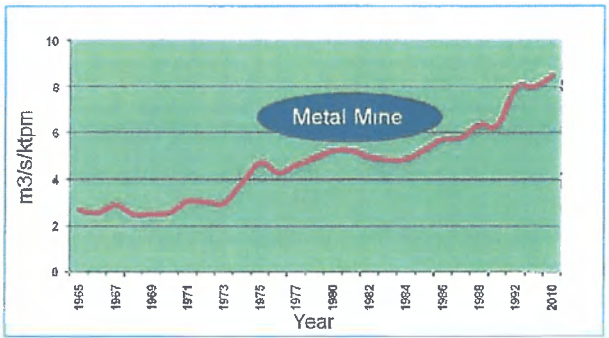

Figure 1: Timescale of ventilation demand in a South African gold mine (Belle, 2012).

It can be noted from Figure 1 that the metal mine air factor (unit volumetric flow per unit tonne mined, i.e.:m3/s/ktpm) has tripled from 1960s to the end of the century. This increase in air factor can be attributed to increased depth, management of hazards due to change in exposure limits, and increased use of diesel equipment. Similarly, for longwall coal mines in developed countries for the period between the 1980's and 2010's, it can be estimated that the average coal mine air factor ranges from 0.4m3/s/ktpm to 1.5m^3/s/ktpm. This air factor may vary significantly depending on in-situ

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gas conditions, effectiveness of gas drainage and monthly production rates. Analyses of air factors of the first longwall coal mine in Queensland, Australia, operated for two decades, indicated a fairly constant average air factor of $1.5m^3/s/ktpm$ as shown in Figure 2. Due to the difficulty in obtaining ventilation and production statistics, ascertaining a general coal or metal mine air factor was not possible.

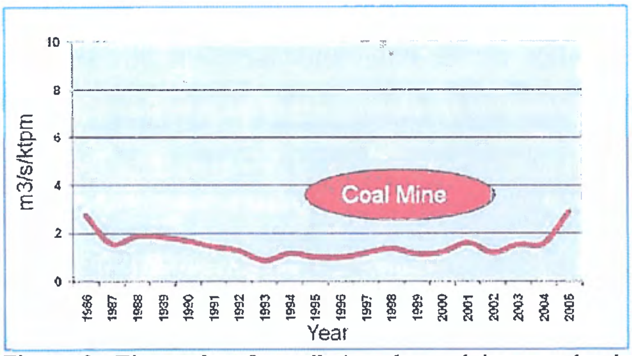

Figure 2: Timescale of ventilation demand in a coal mine (Belle, 2012).

2.1 *Legislated Air Velocities*

The following paragraphs summarise air velocity requirements in the ventilation codes of practice (COP) and legislation of a number of countries. These requirements illustrate that manual and or electronic means of real-time velocity monitoring devices would enable to provide assurance needed on meeting those compliance requirements.

- The Queensland mine safety legislation requires that the Principal Hazard Management Pan (PHMP) must ensure that the ventilating air provided for the mine is of sufficient volume, velocity and quality to remove atmospheric contaminants from mining operations and maintain a healthy atmosphere at the mine during working hours. Also, it must ensure that the effective working temperature requirements are met. Effective temperatures are determined using measured wet bulb and dry bulb temperatures and air velocity. (Coal Mining Safety and Health Regulation 2001, Regulation 343-345)
- Also, in terms of the same legislation, controlled ventilation for a working place in each standing working place that is on the intake side of a working place and in each working place in an ERZ1 (Explosion Risk Zone 1) must provide for a ventilation current of an average velocity of at least 0.3 m/s measured across the cross-sectional area of the roadway in the working place. (Coal Mining Safety and Health Regulation 2001, Regulation 343-345)
- In addition, Safe Work Australia mine safety legislation requires that in areas of the mine where persons work and travel, the ventilation system provides an average air velocity of at least 0.3 m/s measured across the work or travel area (Model Work Health and Safety (Mines) Regulations 2011 Section 649).
- The prescribed Chinese ventilation regulations stipulate minimum ventilation volume per person (4 $m^3/min/person$); decline travel airway velocity limit of 8.0 m/s; and, depending on location or activity, a minimum ventilation velocity of 0.25-0.50 m/sec aimed to attain a minimum diesel emission dilution factor of 0.06 $m^3/s/kW$.
- US regulation 30 CFR 75.350(b) limits belt air velocity to 5.08 m/s; 30 CFR 75.327(b) limits air velocity in trolley haulage entries to 1.27 m/s provided the methane content can be maintained below 1%.
- In South Africa, with the change of legislation from the Minerals Act to the Mine Health and Safety Act in 1996, the prescribed minimum working face air velocity of 0.25 m/s and air quantity of $0.15m^3/s/m^2$ of development heading face was removed and replaced with a risk-based process that ensures the mine operator would perform a risk assessment to determine the minimum air velocities and quantities that would be required to ensure that hazards and pollutants are controlled.
- In a very similar way, Ontario legislation does not stipulate any air velocity requirements (minima nor maxima) but hinges air requirements on the attainment of adequate and stipulated time-weighted exposures for carbon monoxide, radon daughters and diesel particulate matter.
- Polish regulations (§ 190. 1) suggests that air velocity in areas with methane presence, except chambers, cannot be lower than 0.3 m/s, but if there is electricity cable it cannot be lower than 1 m/s. With the application of stoppings, air velocity can be lower if gas concentrations are correct. In addition, air velocity cannot exceed 5 m/s for mine workings (e.g. longwall), 8 m/s for transport drifts (but maximum of 10 m/s) and 12 m/s in intake shafts with cages (Wrona, 2013).

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3 VELOCITIES: DESIGN VS. PRACTICE DISCUSSIONS—NARROW REEF ORE-BODIES: PLATINUM

South African underground platinum mines are typically narrow tabular reef operations that use system design criteria similar to those used by gold mines. The differences between the two commodities are to be found in a steeper rock temperature gradient and, in some cases, flat-dipping reefs that characterise platinum operations. The steeper rock temperature gradient implies that the depth at which refrigeration is required is virtually halved. Therefore the effect of auto-compression is reduced by the same proportion and the effect of air movement is utilised more readily for effective air cooling to delay the introduction of refrigeration. This is exemplified in Table 3. The flatter reef, particularly where wider chromite UG2 reefs are present, allow the use of mechanised or hybrid mining methods that involve the use of diesel-powered equipment.

Table 3: Recommended Velocities in Platinum Mine

Area	Air speed (m/s)
Stope panels (min)	0 m to 650 m depth (21.5°C to 36.0°C VRT): 0.40 650 m to 1000 m (36.0°C to 43.5°C VRT): 0.60 1000 m to 1500 m (43.5°C to 54.5°C VRT): 0.80
Bord and Pillar: Last through road (min) Service road (min)	 1.5 1.0
Conveyor drifts	3.0 (antitropal) 6.0 (homotropal)
Vertical, equipped intake shafts	10.0 to 12.0
Ventilation Shafts (exhausting or intake)	18.0 to 22.0
Main haulage routes	6.0 to 8.0
Decline Intakes	6.0 to 8.0
Chairlift declines	6.0 (max)
Horizontal Exhaust Airways	10.0 (max)

The mining methodology and configuration impact further the choice of face air speeds and the horizon below which refrigeration will be required. Typically the latter is between 600m and 650m below surface (between 35.5°C and 36.5°C VRT). Table 3 shows design air criteria typical of some platinum operations. This design code therefore acknowledges the need to provide additional air, as indicated by the higher stope panel design air speeds, as the virgin rock temperature (VRT) increases with depth in an attempt to delay the point at which air cooling and refrigeration will be introduced. However, even below the 650m "horizon", higher air speeds are sustained to reduce the rate at which temperatures increase as air flows through the workings.

Figure 3 indicates the variation of average stope panel air speed in mines within a group of platinum mines over a number of years. The results indicate an increase in the average air speed over the years.

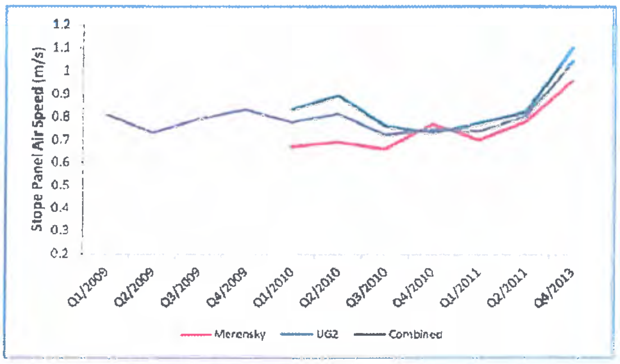

Figure 3: Variation of stope panel air speed in Platinum mines

The data shown indicates that average velocities tend to exceed the minima indicated in Table 3–therefore indicating that there is a tendency for air velocities and quantities to exceed standards. This should be seen as an effort aimed at delaying the onset of refrigeration where this is not used yet or at extending the capacity of existing refrigeration plants. In both instances the indication is that the efficiency of existing systems is on the rise. Despite this, power requirements are increasing and it is arguable that system design parameters could have been over-stated. One aspect that is not provided by this data however is the existence of "outliers" that represent work places where the air speed is below the set minima and where conditions are likely to be sub-optimal.

Figure 4 indicates the variation of air speed in the exhaust shaft of a number of platinum operations. The plot also shows the "critical speed" zone. The data in Figure 4 indicates that air speeds in exhaust shafts vary considerably and that a number are below the design "minimum speed".

In addition, a few of these operate within the critical zone – although in only one of the six, the shaft is likely to handle very humid air. In addition, Figure 4 indicates that a number of these shafts operate well above the upper air speed limit. The lower speeds are indicative of shafts where production has been reduced and that therefore require considerably less air than originally intended. Where the air speeds are considerably higher than the design values, it is possible that the main fan design volumetric capacity has been increased to meet tonnage requirements higher than the original design parameters – possibly due to the simultaneous mining of multiple reefs. In other cases (shafts 1 and 2), the shafts are very short, less than 40m, and therefore making the higher power demand tolerable.

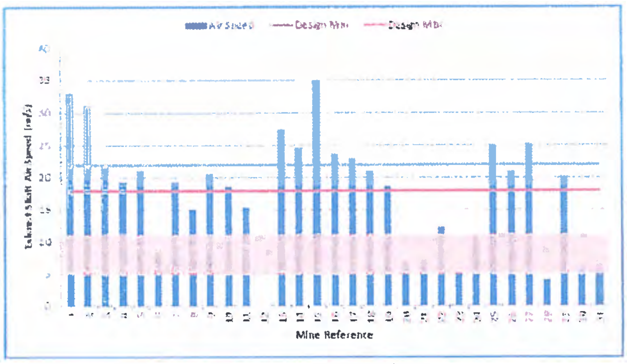

Figure 4: Exhaust shaft air speed distribution

It may also be argued that the higher volumetric flows are due to excessive underground airway leakage. In all cases, where higher volumetric flow occurs, the main fans' selection criteria contained contingency factors that are currently exploited.

The "air factor" is used as a high-level comparative indicator of the mine's air requirements. Figure 5 indicates the distribution of air factors for a selection of shafts taken from the grouping used in Figure 4. Figure 5 shows the relative air factors while Figure 6 shows these as a function of the VRT at the bottom of the respective mine.

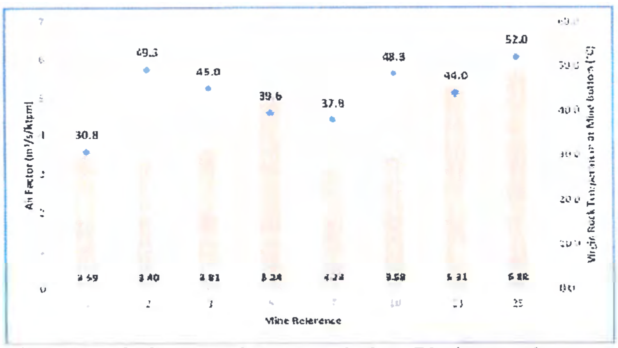

Figure 5: Air factor and VRT variation, Platinum mines

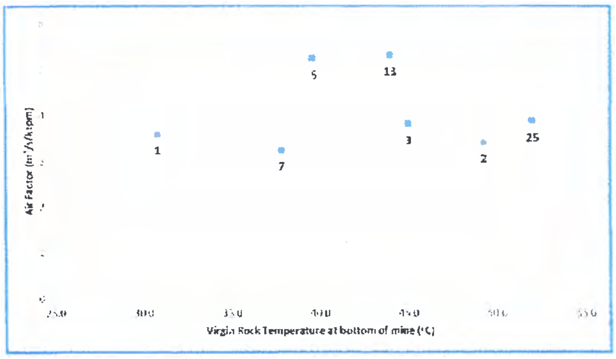

Figure 6: Air Factors vs. VRT at shaft bottom

Figure 7 illustrates the fact that the air quantity allocation is seemingly not related to the depth of operations. Typically this is due to the "degree of scatter" of mining operations throughout the mine and also to the fact that different shafts are at different level of maturity in their life.

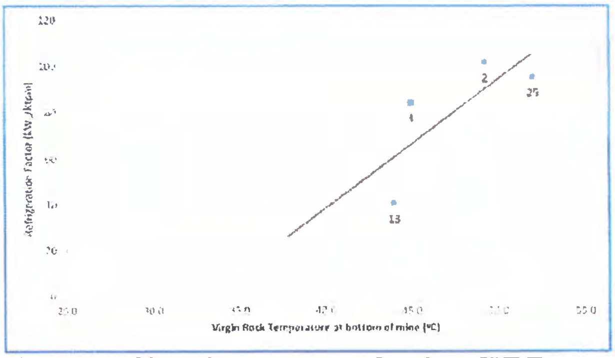

Figure 7: Refrigeration Factors as a function of VRT

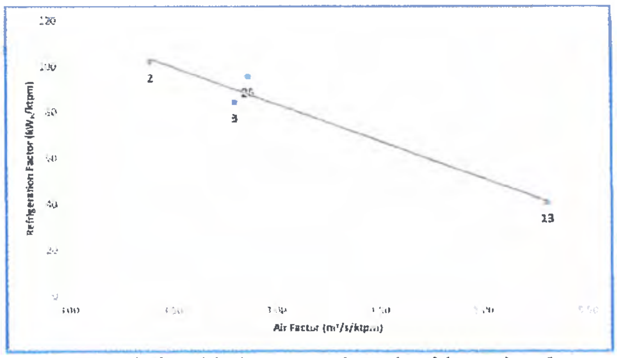

Figure 8: Relationship between air and refrigeration factors

Similarly with the "refrigeration" factor defined as the refrigeration capacity per kiloton mined per month, Figure 7 depicts the refrigeration factors for the mines that use refrigeration. In this case a more definite relationship seems discernible. Again, the reason for this is related to the intensity and distribution of mining operations at different depths. Irrespective of the seemingly erratic relationships shown above, there seems to be a more balanced relationship emerging when the refrigeration factors are compared to the VRT as shown in Figure 8. This indicates that a logical

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balance has been struck between air power and air cooling capacity.

4 VELOCITIES: DESIGN VS. PRACTICE DISCUSSIONS—NARROW REEF ORE-BODIES: DEEP LEVEL GOLD MINES

Historically, legislation has been the key driver of improved health and safety in most mining jurisdictions globally. Today many mining companies are trying to give effect to their duty of care and this is due to having multi-national company footprints and shareholder demands for ethically produced good/minerals. Legislation such as the Minerals Act in South Africa prescribed certain minimum requirements such as a minimum air velocity on a working face of not less than 0.25 m/s and a minimum volume to be supplied to a development end of $0.15m^3/s$ per m^2 of face area. These parameters in turn were used for planning purposes and companies began to add factors of safety to ensure that are compliant with legislation, had minimum re-entry periods after blasting and increased worker productivity with improved environmental conditions. The Mine Health and Safety Act (1996) ushered in the era of risk based controls which had a further influence on minimum velocities and air requirements. Table 4 below provides some of the minimum planning criteria used, current practice and factors of safety.

It is noticeable that at the working face, air velocities are considerably higher than the design parameters to provide better environmental conditions and boost safety and productivity. Ventilation remains the most critical component in providing an acceptable working environment for the workforce. Ventilating air is the primary means of diluting and removing Heat, Dust, Gases, and Radiation (Radon Gas). One of the largest deep level gold mining companies has set the following as design parameters for underground work places in its South African Mines (Thermal Stress COP, 2010); Maximum wet bulb temperature of 28.5oC; Minimum stope face velocity of 1 m/s; Minimum air volume in a development end of 0.3 $m^3/s/m^2$. The prime considerations for these targets were:

1. The cognitive ability of workers is not significantly impaired; this has a direct bearing on the ability of personnel to work safely.
2. At 28.5°C un-acclimatised workers will not be exposed to any significant risk of Heat Stress.
3. A wet bulb temperature of 28.5 °C together with an average velocity of 1.0 m/s results in the air having a Specific Cooling Power of 300 W/m^2. A person performing hard physical work, for example lashing (shovelling), will generate approximately $240W/m^2$. These conditions will therefore enable acclimatised workers to perform at 100% of their productive potential.

The requirements for diluting diesel exhaust emissions also influenced the increase in ventilation requirements over time, with requirements ranging from 0.03 and 0.06 m^3/s per kW of rated engine power and a minimum velocity of 0.5 m/s at the point of application (Howes, 2011).

As mining depths increased and thermal gradients got steeper (in some areas where virgin rock temperatures were higher at shallower depths), air and refrigeration requirements increased, thereby also increasing airway velocities and excavation (airway) sizes. It is estimated that since the first mine refrigeration system installed in 1919 the growth in worldwide refrigeration capacity was linear at about 3 megawatts of refrigeration (MWR) per year until 1965, when the total capacity reached about 100 MWR. Since 1965 the growth in capacity has been exponential, with a doubling every six or seven years (Howes, 2011).

To illustrate the thermal influence on air velocities, data was sourced from two large deep gold mines in South Africa. Mine 1 has a mean rock breaking depth (MRBD) of 1141 m with a mean virgin rock temperature (VRT) of 36.7 0C and mine 2 has a MRBD of 3176 m with a mean VRT of 56.5 0C. Mine 1 and Mine 2 have mean heat production rates of 178 kW/kt and 450kW/kt respectively. The average stope velocities are indicated in the graph below, with various limits indicated, limit 1 refers to the previously legislated 0.25 m/s, limit 2 refers to a 0.3 m/s accepted standard in some mines, limit 3 indicates the 0.5 m/s generally applied stope face velocity and limit 4 is the 1 m/s that most mines plan to achieve, in combination with a wet bulb temperature of 28.5 0C, a specific cooling power of 300 W/m^2.

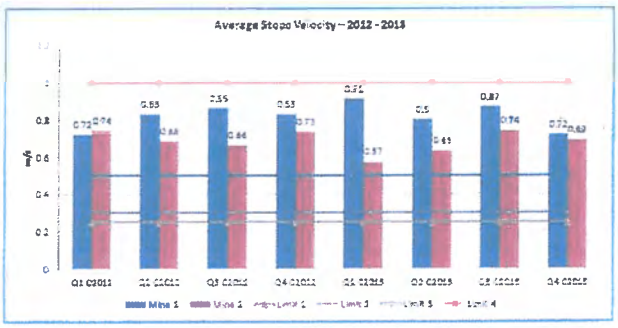

Figure 9: Average Gold Mine stope velocity distribution

Many "rule of thumb" factors (Table 4) and historically used planning values have been used which need to be researched further under controlled conditions for present day economics and health and safety considerations.

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The demand for improved environmental conditions does influence air velocities on working faces and in turn require increased air velocities in main intake and return airways. This is where, as indicated previously, further research may be required to add to the existing body of knowledge on optimal and cost effective ventilation. Table 5 shows the current intake and return airway velocities and upcast and downcast air velocities in two gold mines when compared with design velocities in Table 4.

Table 4. Recommended velocities (m/s) in Gold Mine.

Area	**Current Practice, m/s**
Working faces*	0.3-0.5, up to 1
Development Ends	0.3
Ventilation Shafts	18-22
Decline Intakes	6-8
Dedicated Intake Shaft	18-22
Downcast Shafts	10-12
Intake Airways	6-8
Return Airways	6-8

* Regulated minimum at work place and development end are 0.25 m/s; 0.15 $m^3/s/m^2$.

Table 5. Operating velocity profiles in gold mines.

	Mine 1	Mine 2
Airway	Velocity (m/s)	Velocity (m/s)
Downcast Shaft	11.7	10.4
Upcast Shaft	26.0	18.9
Main Intake Airway 1	7.7	5.4
Main Intake Airway 2	9.4	4.6
Main Intake Airway 3	6.5	5.4
Main Return 1	12.8	6.4
Main Return 2	8.2	5.4

As a short case study, at mine 1, serious dust problem were being experienced in the intake airways, with dust measurement indicating up to 1.03 mg/m^3 (during tipping in the shaft) and 0.04 mg/m^3 (when no tipping was taking place). The dust measurements were taken using a real–time dust monitor (PDR 1500). The air velocity measured flowing over the tips was 9.4 m/s (exceeding current accepted norms for intake airways, see Table 4). A planned station loop/parallel airway will reduce the velocity to about 4.5 m/s and also the dust concentrations. This reduction in the velocity and dust problem is anticipated to reduce the number of "foreign body in the eye" incidents.

5 VELOCITIES: DESIGN VS. PRACTICE DISCUSSIONS—BORD AND PILLAR MINES

A good mine ventilation system always begins with the initial development of the mining plan, which should always have alternatives. A well thought out ventilation system can minimise long term problems, builds in flexibility for expansion without exorbitant cost, reduces capital expenditure, and phases-in capital outlay over the life of the project. Table 6 summarises typical bord and pillar coal mine (South Africa) design velocities aimed at managing the air quantity requirements for the mine's working sections. Figure 10 shows the operational air velocity distribution in this thermal coal mine, typical of many others.

Table 6: Recommended velocities in bord and pillar coal mines.

Area	**Velocity*, m/s**
Last through velocity (no detectable methane)	1.2
Last through velocity (detectable methane)	1.5
Longwall Mid-Face Design Velocity	3.0
Worked out Panel Minimum Velocity	0.5
Minimum Roadway Velocity for Non-flameproof vehicles	0.5
CM On board scrubber unit *	0.4 m/s
Maximum Intake Air Crossing Velocity	3.0
Maximum Return Air Crossing Velocity	5.0
Downcast Service Shafts	10.0
Dedicated Downcast	20.0 m/s
Up cast Shafts	13.0-20.0
Intake conveyor air velocity (maximum)	3.0

* Minimum CM On board dust scrubber unit flow requirements is 0.4 $m^3/s/m^2$; minimum face ventilation is 0.25 $m^3/s/m^2$.

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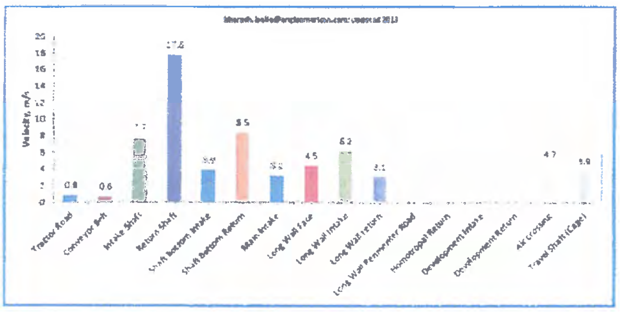

Figure 10: Thermal coal operational velocity distribution

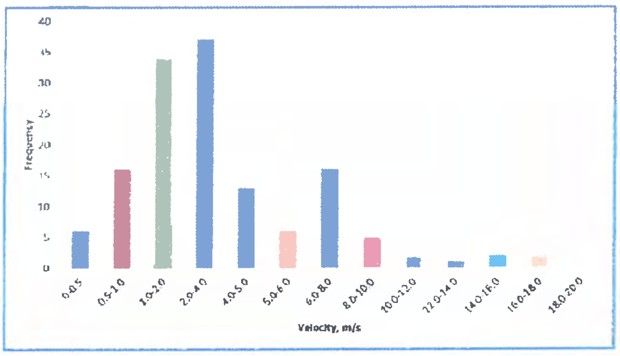

Figure 13: Metallurgical coal velocity distribution

6 VELOCITIES: DESIGN VS. PRACTICE DISCUSSIONS—LONGWALL COAL MINES

Historically, minimum coal mine velocity values are prescribed by legislation to mitigate the effect of hazards on workers and equipment or past limitations based on experiences. Figures 11 and 12 show the velocity profiles of two operating longwall coal mines in Australia. Similarly, Figure 13 shows the histogram of various air velocities in operating coal mines in Australia. At an operation level, these airway velocities are important as they determine number of roadways in mains or panels for a design ventilation load capacity.

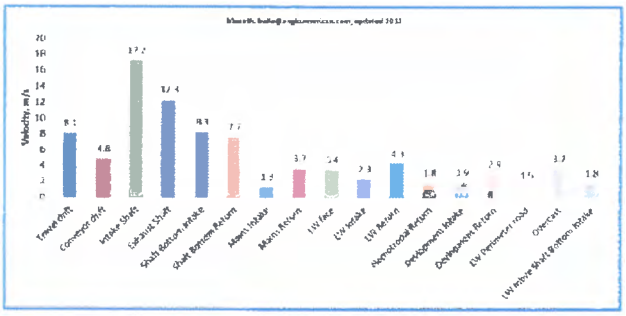

Figure 11: Metallurgical coal (mine A) velocity profile

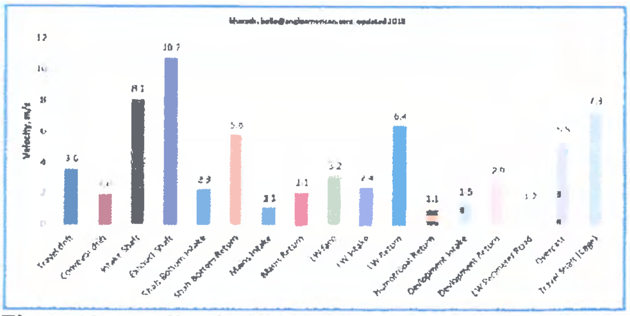

Figure 12: Metallurgical coal (mine B) velocity profile

7 VELOCITIES: DESIGN VS. PRACTICE DISCUSSIONS-DIAMOND MINE

Diamond mining operations in Kimberlitic pipes make use of either block caving or sub-level caving methods. The basis for the design velocities is unknown other than those included in the MVSSA data book. Table 7 provides the design velocities used for diamond mines using block cave, incline and sublevel cave mining methods (Belle, 2007) shown schematically in Figure 14.

Table 7. Operational design velocities (m/s) in diamond mines (Belle, 2007).

Area	Velocity, m/s
Horizontal Intake Airways-no workers	7.0
Horizontal Return Airways- no workers	12.0
Downcast Service Shafts	10.0-12.0
Dedicated Downcast (if necessary)	15.0 m/s
Up cast Shafts	20.0-22.0
Design Tunnel Face Velocity (dust dilution)	1.0 m/s (dust dilution)
Intake conveyor air velocity (maximum)	3.0 - 4.0
Minimum tunnel velocity-other works	0.5
Maximum Rim tunnel, ramp velocity	4.0
Minimum rim tunnel velocity per level (no production)	1.5

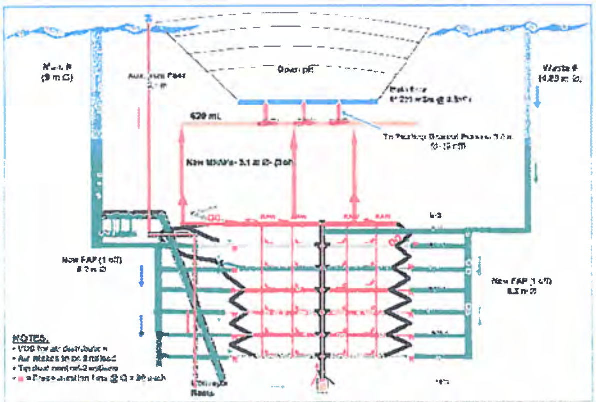
Figure 14: Diamond mine ventilation design (Belle, 2007).

8 INTAKE AIR VELOCITY IN CONVEYOR BELT ROADS.

The design air velocity for intake airways used by conveyor belt installation poses some cogent choices for ventilation engineers.

8.1 *Conveyor Air Velocity*

As indicated in earlier sections of the paper, a commonly quoted ventilation design value is 4 m/s in conveyor roads used in almost all commodities. This is to reduce the physical discomfort of large dust particles striking the skin, although it is not a health hazard. For example, air velocity along the longwall face or main intakes or panel intakes, the air velocity is limited to between 4 to 6 m/s. Similarly, this velocity has an impact in the management of fire hazards. The question often less debated is if these values still hold true based upon recent empirical data or studies or if workers are likely to be using that particular travel road on a regular basis associated with conveyor belt operations.

Conveyor belt or vehicle fires in any mine or tunnels are major safety hazards. Typically, conveyor belt roads should be physically segregated from the rest of the intake airway system, but access to these is necessary for maintenance and repair along their full extent and they are therefore not leak proof. In most coal mines, the airflow along longwall panel conveyor belt roads is homotropal, whereby the belt air is coursed from the main gate to the return airway, travelling in the same direction as the belt. Belt air is therefore not used for panel face ventilation, to reduce the impact of dust pick-up dust or of a conveyor belt fire. US regulation 30 CFR 75.350(b) limits belt air velocity to 5.08 m/s; 30 CFR 75.327(b) limits air velocity in trolley haulage entries to 1.27 m/s provided the methane content can be maintained below 1%.

In the USA the use of air drawn along conveyor belt roads is permitted, with a condition that the air velocity in those airways shall not exceed 1.52 m/s. These requirements are based on small-scale approval tests for fire-resistant conveyor belting conducted at this velocity. Subsequently, at the request of MSHA, fire tests were carried out to understand the impact of higher air velocities of up to 4.1 m/s (Lazzara, 1986). The results indicate that for the test fire conditions, the hazard of propagating conveyor belt fire is reduced at the higher velocity of 4.1 m/s compared with at 1.5 m/s, and even suggest that the 1.5 m/s air velocity limitation is inappropriate. Therefore, it can be reasoned that the widely used maximum conveyor belt velocities are linked to the maximum conveyor belt test study conditions, unless proven otherwise.

Over the past two decades, in subway transport tunnel fire safety management, 'critical' minimum fresh velocity is often sought in order to maintain smoke-free conditions which varies with the size of fire (Tarada, 2000). The 'critical' velocity is defined as the minimum air velocity that will limit the spread of smoke in the event of a fire. Typically, these air velocities are partially based on empirical data and static experimental data which depend on the magnitude of fire, tunnel geometry and slopes. A further study (Tarada, 2010) cautions how the concept of a 'critical' velocity is interpreted and applied in tunnel ventilation designs, since there are a number of drawbacks in the generation of high air velocities in tunnels during a fire emergency such as possible enhanced fire spread where vehicles are involved, and loss of smoke stratification.

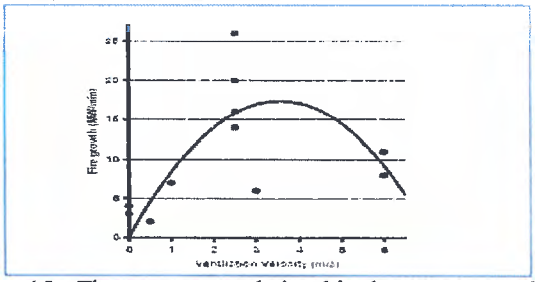
Figure 15: The apparent relationship between tunnel fire growth rate and air velocity (Carvel et al, 2009).

In underground transport tunnels, an international recommendation is that a fixed air velocity of 3 m/s is maintained, to prevent the smoke back-layering in the event of any fire (Nordmark, 1998). Carvel (2010) noted that while no experimental fire tests have been carried out in tunnels with air velocities greater than 6 m/s, available data suggests that there is a decline in fire growth rate with increasing velocity, for velocities above about 4 m/s (Figure 15).

Figure 16 shows a summary of main conveyor road air velocities recorded in coal mines. As noticed, the velocities have exceeded the 'perceived' 4.0 m/s design velocity in some areas of

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the conveyor belt road, while the average velocities have remained below 4.0 m/s.

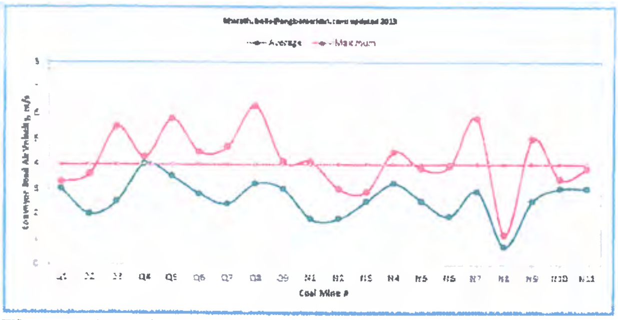

Figure 16: Main conveyor road air velocity distribution in coal mines (Belle, 2014).

8.2 *Exhaust Shaft Critical Air Velocity*

The following discussion attempts to challenge the validity of the 'forbidden' critical shaft velocity range of 7 m/s to 12 m/s used in shaft size determinations. Figure 17 shows the histogram data relating to exhaust shaft air velocity acquired from different commodity mines from around the world. The data suggests that significant number of mines do operate in the 'critical velocity' range. However, air velocity measurements made in a coal mine exhaust shaft have indicated that the water blanket or droplet dancing effect can take place even at air velocity of 17 m/s, unlike the commonly accepted critical zone of 7 m/s to 12 m/s that was observed in the 1950s.

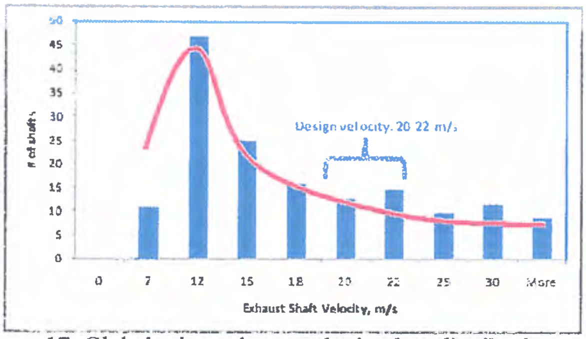

Figure 17. Global mine exhaust velocity data distribution

Within the critical velocity zone, water blanketing could place the exhaust fan into the stall zone, depending on the effective resistance placed by the water blanket above the network's operating point. At this velocity, the water runs down the shaft walls to the brow of the shaft; and is swept back up into the exhaust shaft. At the 'critical' velocity, the water is neither carried out of the shaft nor can it fall to the floor. This creates a plug of water which could result in surging static pressure conditions in the ventilation shaft. The exhaust shaft velocity statistic provided herein reflects on reality on the ground and validity of continued use of such values for such ventilation designs mostly by consultant reports.

8.3 *Maximum Air Velocity – Dust Dispersion*

Amongst various air velocity design factors, another commonly quoted design air velocity is 4.0 m/s in conveyor road, face areas and intake airways.

Reinhardt (1972) showed that over a range of air velocities between 0.3 to 2.6 m/s, approximately 40 % of the coarse dust would settle out of the air within the first 30 m of the return airway and that 70 to 90 % would have settled out within 300 m of the face. For finer dust particles the values for were 0 to 20 %, and 35 to 60 % respectively. Settling and entrainment of coarse dust (greater than 10 microns) is highly dependent on velocity. At lower velocities it settles out of the air readily but on the other hand, it is more readily entrained at high velocities. The NCB report (1978) quotes a Polish study (Gruszka et al) that provided the impact of air velocity on dust levels for various dust fractions (Figure 18). On the one hand increase in air velocity reduces the dust level by increased dilution. On the other hand, increased air velocity may result in greater dust pick up by the air stream.

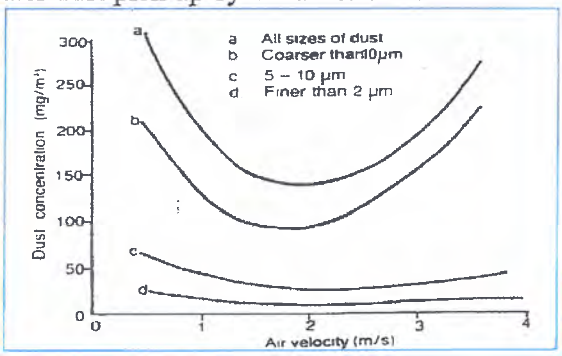

Figure 18: Relationship between dust concentration and air velocity for different particle sizes after Gruszka et al (in MRDE, 1980).

Another reference in relation to air velocity with respect to the dust was found in a UK recirculation study (MRDE, 1980) that recommends future research for ascertaining the relationship between air velocity and dust levels. The MRDE study noted that control of respirable dust levels can be achieved with face air speeds up to 3.5 to 4.0 m/s. Furthermore the study noted that the coarse dust pick up is known to be more susceptible to air velocity than is fine dust. The study concluded that air velocities in excess of about 2.5 m/s would result in increases of coarse dust concentration even with efficient filtration.

In overall, a research study by Ford (1976) suggested a recommended face air velocity below 4.0 m/s because coarse dust becomes intolerable to workmen at this velocity. The basis for this value is to manage the physical discomfort caused by large

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dust particles (Figure 17) striking the skin later suggested by McPherson (1984).

Currently, the question relating to the appropriateness of the 4.0 to 6.0 m/s limit for conveyor roads or intake roads is less debated or questioned. The question is possibly around which criterion, or number of these, determine the final selection: is it based on recent empirical data or studies relating to increased daily production rates or speed of conveyor belts or the likelihood of workers walking through that particular travel road on a regular basis? Furthermore, the authors could not find any evidence of the data source that justified the 4.0 m/s velocity limit based on airborne dust considerations. What was missing in most of the design expert review documents (leading or misleading) was the rationale behind these air velocities and reference to 'no go' values.

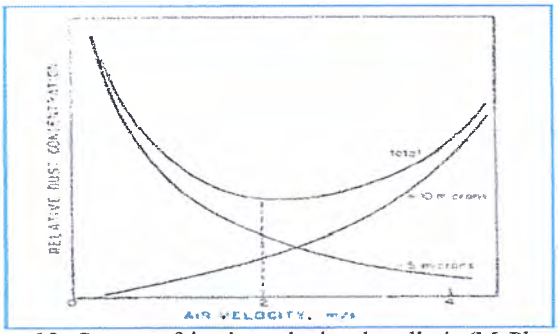

Figure 19: Source of intake velocity dust limit (McPherson, 1984).

The existing 'standard velocity design' values are critical to determine the size of airways, number of airways, shaft sizes, and finally main fans, mine cooling systems, operating cost, capital cost of ventilation systems. For example, is it 8 m/s or 10 m/s or 12 m/s in main returns or is it 6 m/s for main intakes and panel intakes or main declines? The impact of making the "right" selection is significant in terms of number of roadways that are required to be driven for mains or development panels in order to manage a multitude of hazards using adequate ventilation.

In this context a comparison is introduced in Table 8 to indicate the margin by which any design parameter exceeds minimum norms. This can be seen as introducing "factors of safety" or contingencies in the design process e.g. using a design stope panel face velocity of 0.3 m/s rather than the 0.25 m/s based on previously legislated limits in South African mines implies a safety factor of 0.3/0.25 = 1.2, meaning a 20 % "safer" margin.

Use of a "Safety Factor" for each air velocity parameter exemplified in this paper is based on operational experience. It is noted that most of the historic design velocities can be challenged and new velocity criteria may be justified. This can be seen as incorporating additional "levels of safety".

Table 8. Application of "safety" factors to airway velocities.

Current Design Velocity, m/s	Safe and Economic Velocity, m/s	Factor of Safety*
Minimum stope face panel velocity, 0.25	0.3	1.2
Max Intake Roadway, 6.0	8	0.75/1.33
Max Return Roadway, 8.0	10	0.8/1.2
Conveyor Road, 4.0	6	0.66/1.33
LW Face, 4.0	6	0.66/1.33

* These to be reviewed with increased operational data

The proposed approach presented herein is an attempt at exploring and providing guidance for an operational approach based on economically derived design 'air velocities'. This results in the de facto definition of a 'safety factor'. These reviewed 'design velocity standards' are based on operational experience and would benefit ventilation officers and others responsible for ventilation control devices (VCDs) and for performing ventilation surveys. The authors have jointly shared knowledge operational air velocities in main intakes, return airways, coal panels, gold and platinum stope panels, and diamond mine development and production sections. It is hoped that the proposed concept of 'design air velocity' values used along with adequate 'safety factors' will provide further guidance to those who use these concepts as part of their working routines as opposed to merely accepting 'air velocity values' from various ventilation study or expert reports.

9 CONCLUSIONS

With changes in mode of worker transport, increasing levels of mining automation, increased production and other changes in mining practices over the last three decades, a review of historic ventilation design velocity is warranted. In the authors' opinions, such reassessments will provide assurance as to the appropriateness of using air velocity parameters for future designs or for the purpose of developing a new set of safe and economically sound design velocities.

What was obvious from the operational experiences recorded in this paper is that the design velocities are not necessarily reflected in operational data. Historic design velocities are often academically quoted in text books, consultant and authoritative review reports. These do not cater for practical challenges usually encountered at the

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operations. The information summarised in this paper is indicative of current operational velocities experienced in mines and it contrasts with "traditionally" accepted design parameters – often over four decades old.

The concept of testing these by the introduction of the "safety factor" concept may be useful to determine whether an air velocity is indeed 'economic' and/or 'safe' or whether it is just a mere 'accidental' value. The true value of mine air velocities should be tested against well-established safety and health considerations first and then for best economic results – this should automatically exceed any regulatory requirements.

10 ACKNOWLEDGEMENTS

The authors are indebted to all colleagues on various operations who have assisted in collating the data. In addition, the authors are grateful to many local and international professionals and technical reviewers for their technical contributions and constructive criticisms to this paper.

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Coal Dust Monitoring of the Future: Application of Passive Real-Time pDR1000 and Active PDM3700 Continuous Dust Monitors

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Australia

ABSTRACT: In the United States, on August 1, 2014, the US mine regulator's (MSHA) landmark respirable coal dust rule went into effect resulting in reduced personal occupational exposure limit (OEL) for coal dust. From Feb 1, 2016, the US regulation required US coal mine operators to use mass-based continuous (real-time) personal dust monitors (CPDMs) to monitor mine occupations exposed to coal mine dust. This is intended to benefit all parties to learn quickly about the personal dust exposures using MSHA approved PDM3700 continuous dust monitor, which is a trade name of the CPDM that is used for compliance determination in US coal mines commonly termed CPDM. On Aug 1, 2016, the over-all respirable dust standard in coal mines is reduced from the historic 2.0 to 1.5 mg/m^3. Reporting dust levels in real time empowers miners and operators to take immediate action to avoid being exposed to excessive airborne dust. Similarly, passive light scatter based real-time dust monitoring is not new to the global mining industry to evaluate the effectiveness of engineering dust controls.

This paper discusses a comparative study between a passive light-scattering based real-time (pDR1000) monitor and the Higgins-Dewell (HD) type gravimetric sampler, operated in accordance with the international size-selective curve as a 'true sampler' that is used to compare the performance of the CPDMs. The study results have shown that the dust levels measured with the passive pDR1000 units were significantly different to the gravimetric sampler data, unlike observed in the controlled aerosol chamber study (Belle, 2006). The paper highlights the experiences of using PDM3700 in Australian coal mines to assess worker personal dust exposures. The benefits of using the PDM3700 is so significant that a few such units can replace the current strategy of gravimetric based sampling in all in coal mines thus minimizing delays in regulatory exposure data submissions. The paper recommends the introduction of the PDM3700 dust monitor as a legislative tool in Australian coal mining industry and expediting its approval for use in Australian mines.

Key Words: Real-time, coal dust, silica dust, evaluation, mining

1 INTRODUCTION

In the United States, MSHA's landmark respirable coal dust rule was promulgated on August 1 2014 resulting in reduced personal occupational exposure limit (OEL) for coal dust. On August 1, 2016, the overall respirable dust standard in coal mines was effectively reduced from the historic 2.0 to 1.5 mg/m^3 of air (MSHA, 2016). In addition, on February 1, 2016, the US mine regulator (MSHA), required US coal mine operators to use mass based continuous personal dust monitors (CPDMs) to assess worker occupational exposure to coal mine dust in underground coal mines. It is envisaged that the reporting of dust levels in real time will empower miners and operators to take immediate action in avoiding exposure to excessive airborne dust levels. The implementation of the CPDMs will be through the use of the MSHA approved PDM3700 continuous mass based dust monitor using the Tapered Element Oscillating Microbalance (TEOM) principle. PDM3700 is a trade name of the CPDM that is used by the regulator in US coal mines.

In contrast with the TEOM principle, passive light scattered real-time devices have been in use since 1980s to evaluate the effectiveness of ventilation and dust control systems as recorded in various USBM and MSHA dust studies (Williams and Timko, 1984; Page and Jankowski, 1984; Gero and Tomb, 1988). Historically, sources of variations in measured dust levels detected when using real-time monitors have been rationalized for parameters such as dust types, dust levels, monitor orientation, particle size, air velocity, and sensor contamination. It is often noted in these comparative studies that one of the major sources of variations in measured dust levels by the dust monitors could be the size distribution of the parent dust (Soderholm, 1989, Volkwein, 2002).

The conclusions from the above studies are similar in that the use of a real-time monitor as a stand-alone unit is not recommended for personal exposure assessment purposes but rather, more appropriately, for the identification of dust trends during a working shift. The most common sources of variability in the real-time monitoring can be attributed to dust levels, dust type, dust size, air velocity, monitor orientation and contamination of optics.

Conclusions from the research studies indicate that the reliability of stand-alone passive direct-reading light-scattering is inadequate due to their inherent sensitivity to airborne particulate matter other than dust. Despite this, their use is continuing. This is purely because there is no other alternative instrument that incorporates the traditional feature of mass-based continuous dust monitoring for the management of airborne dust in mines.

This paper shares operational experiences in the use of these real-time monitoring instruments for personal dust-monitoring, located side by side in the breathing zone, that were carried out in gold, platinum, coal and diamond mines in South Africa and Australia, under broadly similar conditions.

2 REAL TIME CONTINUOUS DUST MONITORS

Reporting dust exposures in real time empowers miners and operators to take immediate action to avoid over-exposure to airborne dust. Historically, the mining industry has been using various light-scattering real-time dust monitoring instruments operated in active or passive mode to assess exposure to airborne dust. However, none of these were accepted as a compliance tool for personal exposure monitoring. Unlike gravimetric dust sampling methods currently in use that require several days to collect, ship and process, the CPDM's gravimetric based measurement of respirable dust provides more immediate, full-shift exposure data.

Historically, the Hund tyndallometer, a light scattering real-time monitor was used in the 1990s in some South African gold and coal mines for investigations in order to identify excessive dust sources. When the Hund was compared against the gravimetric sampler, the results (Figure 1) showed a very poor correlation in significantly underestimating the dust results. It was therefore recommended to use the Hund only in conjunction with a gravimetric sampler.

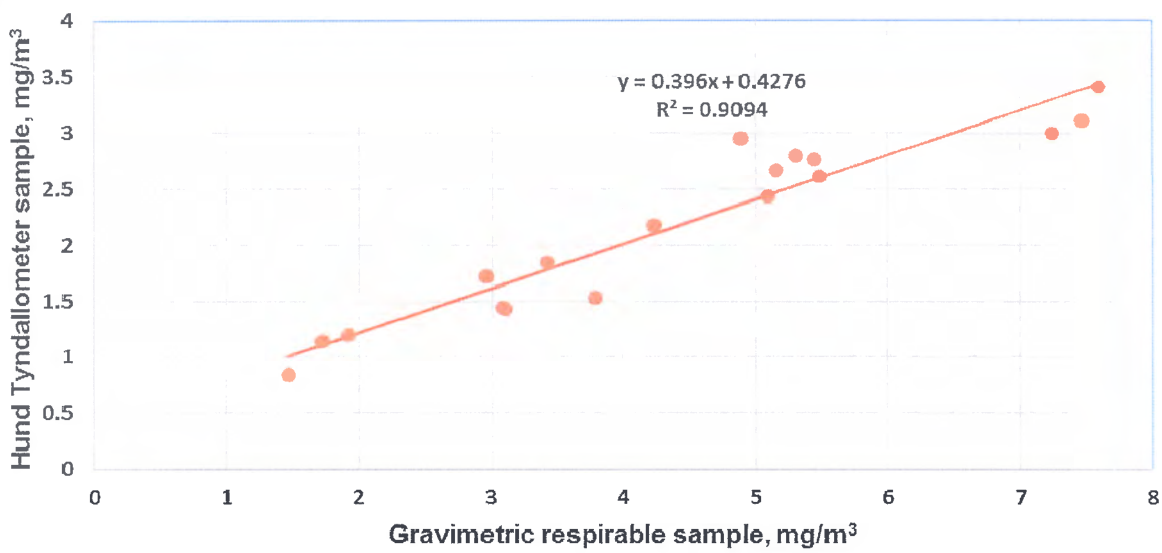

Figure 1: Relationship between Hund-Tyndallometer and gravimetric respirable sample results (Belle, 2002).

Real-time direct-reading instruments for mine dust have been used worldwide for routine engineering controls and risk assessment purposes over four decades due to their added benefits when compared with the gravimetric samplers. Direct-reading instruments or real-time monitors based on light scattering are available to estimate exposure to dust in underground mines. The units operate on a forward light-scattering particle detection principle, which relies on ambient air movement to introduce particles into the sensing chamber. All the available real-time monitors are calibrated using "mono-disperse" particles (Arizona road dust). However, each instrument intended for underground measurements requires a user-determined "correction factor" obtained from a side-by-side gravimetric size-selective sampler, evaluated with "poly-disperse" mine specific dust. There is no "absolute correction factor" available for an individual real-time monitor. The "correction factor" changes with the history of the sampling data obtained in side-by-side comparisons of the real-time monitor and the type of gravimetric size-selective sampler used.

A laboratory study (Belle, 2002) concluded that there was no statistically significant difference in measured dust levels between gravimetric sampler and the pDR1000 (or erstwhile MIE DataRam) in passive mode suggesting that the pDR1000 is potentially closest real-time dust monitor that can be used for use in underground mines.

Any new dust-monitors intended for personal sampling in underground mines should meet the basic requirements outlined below to enhance acceptance by stakeholders:

1. Intrinsically safe for use in coal mines.
2. Sample according to the accepted size-selective criteria (ISO/CEN/ACGIH curve).
3. Meet the ±25% NIOSH accuracy criterion.
4. Preferably use a different quick analysis procedure to the weighting method.
5. Provide real-time concentration values, cumulative shift exposure and sampling time.
6. Be robust enough to withstand the harsh conditions prevailing in mines.
7. Be compact and portable for personal sampling.
8. Be cost-effective in terms of personal sampling.
9. Easy to use and the ergonomically effective for workforce acceptance in use underground.
10. Offer the possibility of collecting dust samples for quartz analysis.

2.1 *PDM 3700 Real-time Dust Monitor (CPDM)*

The PDM3700 real-time monitoring instrument (Figure 2) is commonly known as continuous personal dust monitor (CPDM) in the USA and is approved for underground coal mine use by the US regulator, MSHA. It is a belt-worn, computerized device that measures and displays the real-time, accumulated and full-shift exposure to respirable coal mine dust. This device, as a result of over two decades of development and evaluation which represents a major improvement in respirable dust sampling technology, was approved for use by MSHA and extensively researched by the National Institute for Occupational Safety and Health (NIOSH).

Figure 2: MSHA approved continuous personal dust monitor (CPDM)/PDM3700 (Source: MSHA Fact Sheet, 2016).

The PDM3700 real-time monitor employs Tapered Element Oscillating Microbalance (TEOM) in conjunction with the gravimetric HD type cyclone for its operation as a real-time monitor therefore incorporating gravimetric-based principles. The benefits of the PDM3700 (CPDM) are summarized as follows:

1. Real-time measurement of respirable dust.
2. Computer-controlled pump maintains volumetric flow rate at 2.2 Lpm at mine atmosphere.
3. Heated internal sample line minimizes excess moisture.
4. Ability to sample for longer shifts (up to 12 hours).
5. Monitor weight is approximately 2.0 kg.

3 UNDERGROUND EVALUATIONS

This section of the paper discusses the South African underground evaluation of three sets of real–time monitors (pDR1000) operated in passive mode side by side along with gravimetric samplers. Similarly, Australian evaluation of PDM3700 monitors in underground longwall mines is briefly discussed herein. The pDR1000 real-time dust monitor had a preliminary Intrinsically Safe (IS) certificate for evaluation in mines in South Africa. In Australian mines, in the absence of the IS approval, they were evaluated only in the zone where methane levels were below 0.5%. It was assumed that the HD type gravimetric samplers gave negligible errors and a "true" measurement of personal dust concentration. Therefore, for all monitor comparisons of real-time personal dust levels, the Higgins-Dewell (HD) gravimetric sampler was used as the standard sampler.

3.1 *Test Mines and Instrumentation*

In order to carry out the personal sampling in underground mines, a sampling harness was prepared and the dust monitors were worn in a specific position consistently in all the test mines (Figure 3). The left lapel of the harness contained the HD cyclone, and the pDR1000 positioned side by side in the breathing zone of the wearer. A summary of the sampled mines and sampling locations is given in Table 1.

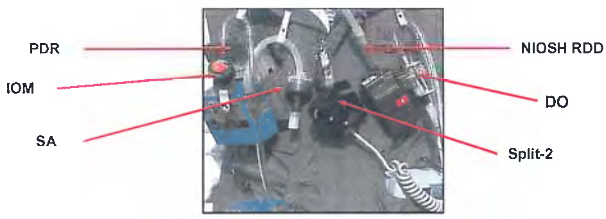

Figure 3: Sampling harness with six personal dust monitors

Table 1: Summary of underground mines and sample areas

Mine Type	Sampled operations
Gold-1/2	Reef and waste tips; shaft levels
	Ore tips along the haulage
	Development heading and stopes
Platinum-1/2	Reef and waste tips; shaft levels
	Ore tips along the haulage
	Development heading and stopes
Coal-1/2	Coal face
	Face, out-bye
	Feeder-breaker and intake
	Transfer points; shaft intake
Diamond	Ore pass
	Haulage way
	Development heading
	Crusher and transfer points
Coal Mine*	Longwall Face

* *NIOSH PDM3700 evaluation*

4 UNDERGROUND RESULTS

During the underground trials, the pDR1000 monitor was evaluated as a passive personal sampler in gold, platinum, diamond and coal mines. The average measured personal dust level recorded by the pDR1000 and the dust level measured by the HD sampler were compared on a one-to-one basis. The scatter plots and regression analysis of the one-to-one relationship between the pDR1000 and HD sampler measurements during the field trials in two gold mines are shown in Figure 4. It must be noted that the high dust levels recorded herein is not a true reflection of underground exposures but rather an effort to understand the behavior of the dust monitors when exposed to high return air dust conditions.

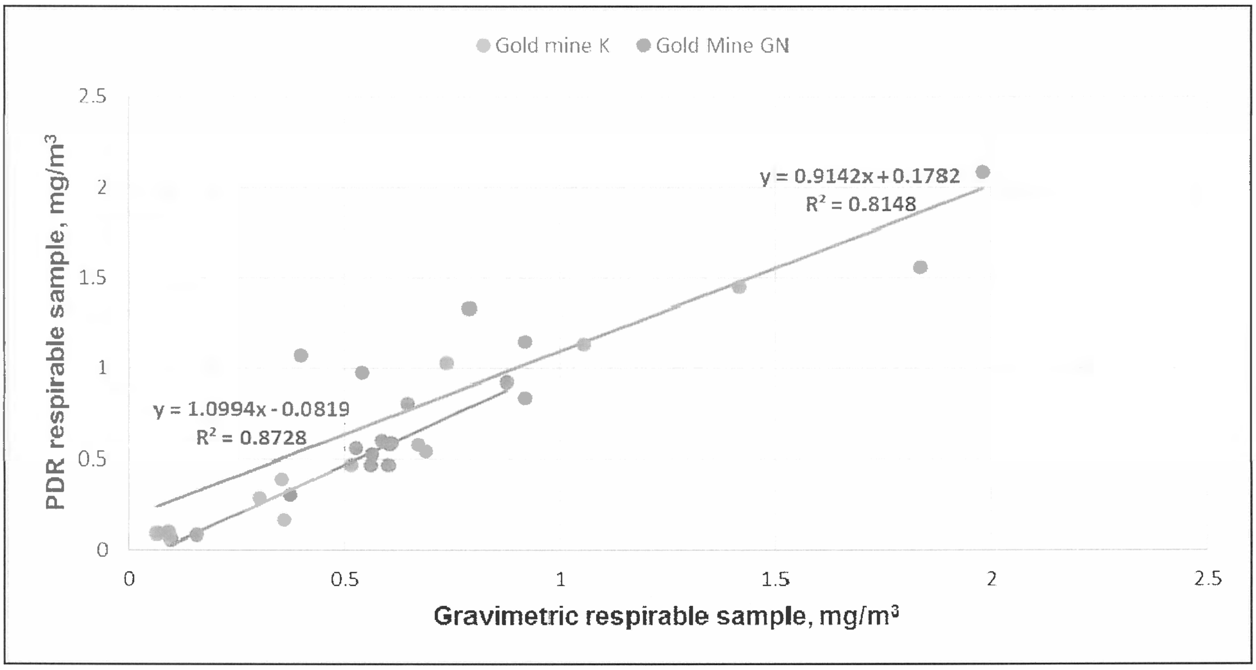

Figure 4: Relationship between pDR1000 and HD samplers in gold mines.

The correlation coefficients (r) between the two samplers in gold mine K and gold mine GN are 0.93 and 0.90 respectively, showing good linearity (r=0.91), despite wide scatter. The regression line from the combined plot indicates that, on average, the pDR1000 monitor overestimates the measured respirable dust concentration by approximately 15% at dust levels below 0.5 mg/m^3. At dust levels between 0.5 mg/m^3 and 2.0 mg/m^3, the data shows that, on average, the pDR1000 monitor overestimates the measured respirable dust concentration by approximately 3.0%.

Similarly, the relationship between the concentration values obtained from the evaluation in platinum mines is shown in Figure 5.

The correlation coefficient (r) between the two monitors in the platinum mine is 0.61. The two monitors show comparatively poor linearity, with wide scatter for all the measured concentration ranges in the platinum mine. Also, it is noted that the measured dust levels in the platinum mine were between 0.2 mg/m^3 and 4.0 mg/m^3. From the regression equation it is estimated that, on average, the pDR1000 monitor overestimates the measured respirable dust concentration by approximately 90% and 40% at the 0.5 mg/m^3 and 2.0 mg/m^3 concentration levels respectively.

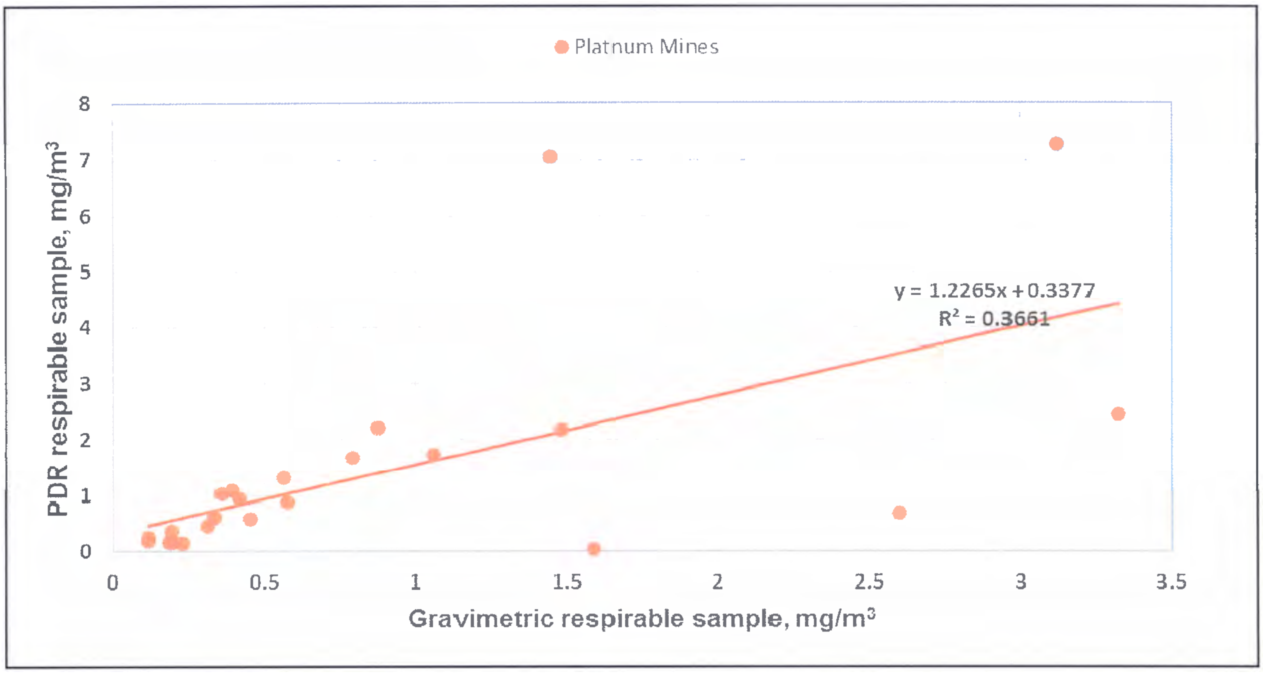

Figure 5: Relationship between pDR1000 and HD sampler in platinum mines.

Similarly, the relationship between the concentration values obtained from the side-by-side pDR1000 and HD samplers during the field evaluation in a diamond mine is shown in Figure 6.

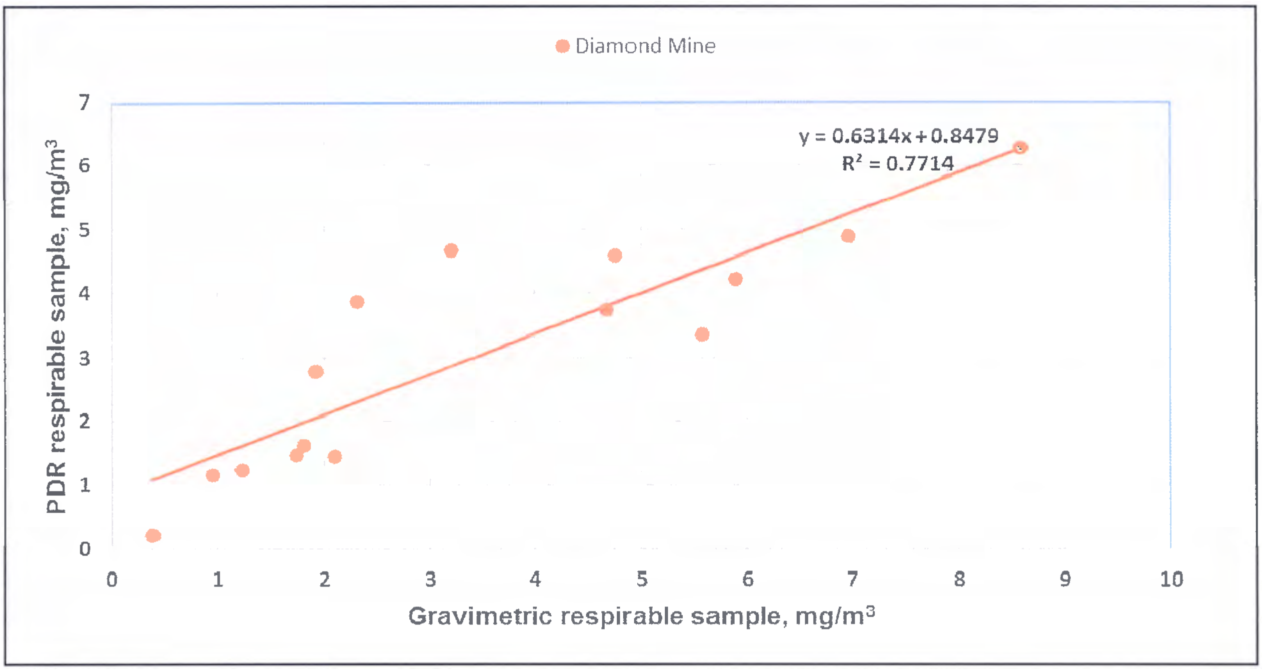

Figure 6: Relationship between pDR1000 and HD sampler in diamond mine.

The correlation coefficient (r) between the two monitors in the diamond mine is 0.88. Kimberlite mines are typically high dust generating commodity when compared with gold, platinum and coal mines. The plot indicates that there was a wide range of measured dust levels. At a compliance level of 2 mg/m^3, the pDR1000 sampler measures approximately the same as the HD sampler. However, at higher concentration levels (> 4.0 mg/m^3), this relationship no longer holds true and the underestimation of measurement by the pDR1000 sampler increased.

In order to determine the relationship between the concentration values obtained from the side-by-side personal pDR1000 and HD gravimetric samplers during the field trials in gold, platinum and diamond mines, the data were plotted as shown in Figure 7. The correlation coefficient (r) between the two monitors in all hard rock mines is 0.78, demonstrating an average linear relationship between the samplers.

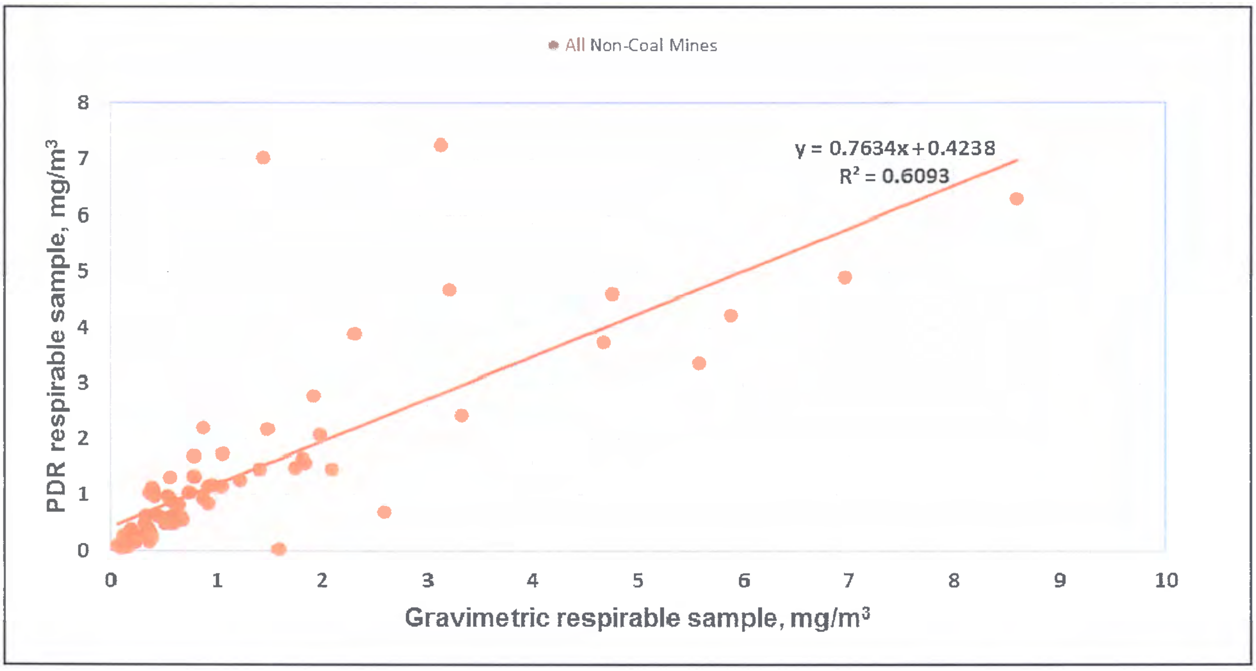

Figure 7: Combined relationship between pDR1000 and HD sampler in gold, platinum and diamond mines.

The relationship between the dust values obtained from the side-by-side pDR1000 and HD samplers during the field trials in two coal mines is shown in Figure 8. The correlation coefficients (r) between the two monitors in coal mine A and coal mine B are 0.71 and 0.89 respectively. The samplers show reasonable linearity (r=0.89) when deployed in coal mines, but the ratio of pDR1000 sampler level to HD sampler level was less than one for coal mines and the measured dust levels were comparatively higher than in the gold and platinum mines. One of the possible reasons for this could be the high air velocities (approximately 2.0 m/s) and the resulting spatial variations in the sampled dust cloud. In order to gain sufficient dust mass on the gravimetric sampler, attempt was made by the personnel to stay on the down-wind of the CM return airway, resulting in higher measured dust levels in coal mine B. The plot indicates that, on average, the pDR1000 sampler underestimates the measured respirable coal dust concentration by approximately 25% at compliance dust limit value of 2.0 mg/m^3.

The combined scatter plot of all mine data (Figure 9) shows an average linear relationship (r = 0.78) between the pDR1000 and HD sampler dust measurements. From the linear regression equation it is estimated that, on average, the pDR1000 sampler underestimates the measured respirable dust concentration by approximately 12% for a compliance level of 2 mg/m^3.

As the measured dust levels increase, the pDR1000's underestimation of these levels also increases, and at 4 mg/m^3 the amount by which it underestimates also doubles. However, at low concentrations (0.1 mg/m^3), the pDR1000 overestimates the reading by approximately five times the measured HD sampler levels.

Based on field evaluations, it is concluded that the use of the pDR1000 sampler in a stand-alone passive mode for compliance purposes is not recommended, and the instrument will seriously overestimate at lower concentrations and underestimate at higher dust levels. From the underground experience, it was found that the pDR1000 monitor is portable and has all the desired features, which would be of benefit to the ventilation officers, dust hazard specialists and the mine inspectors.

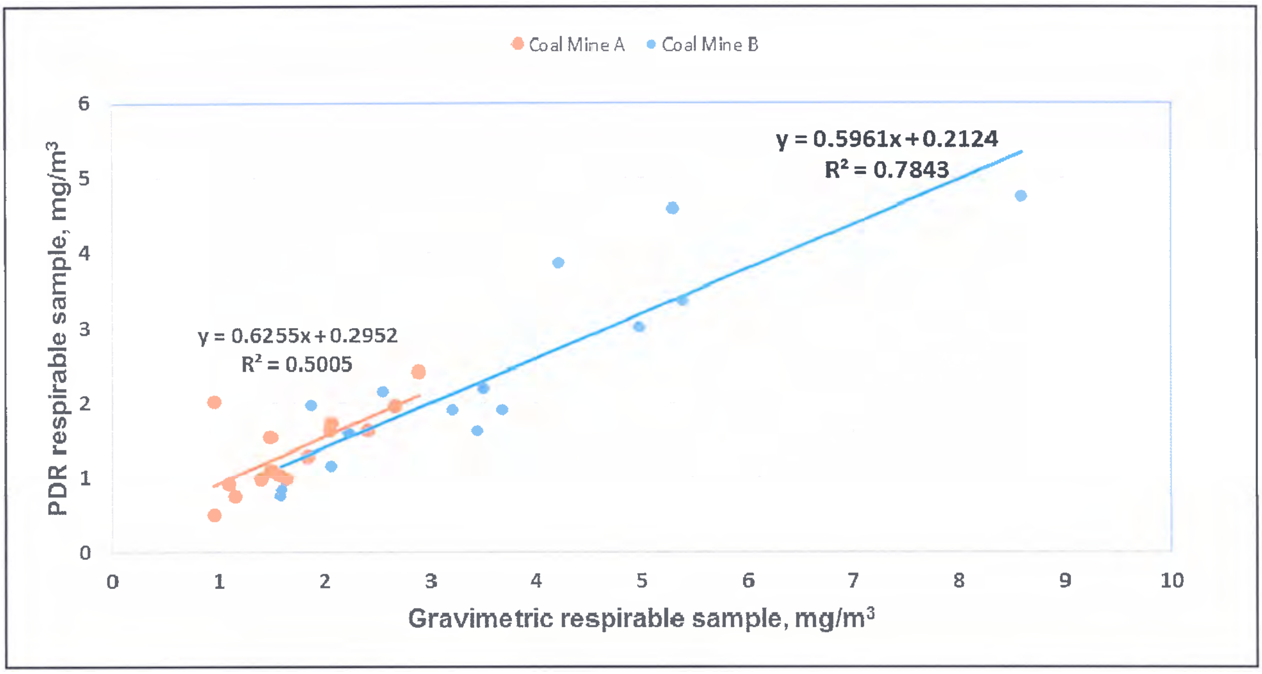

Figure 8: Relationship between pDR1000 and HD sampler in coal mines.

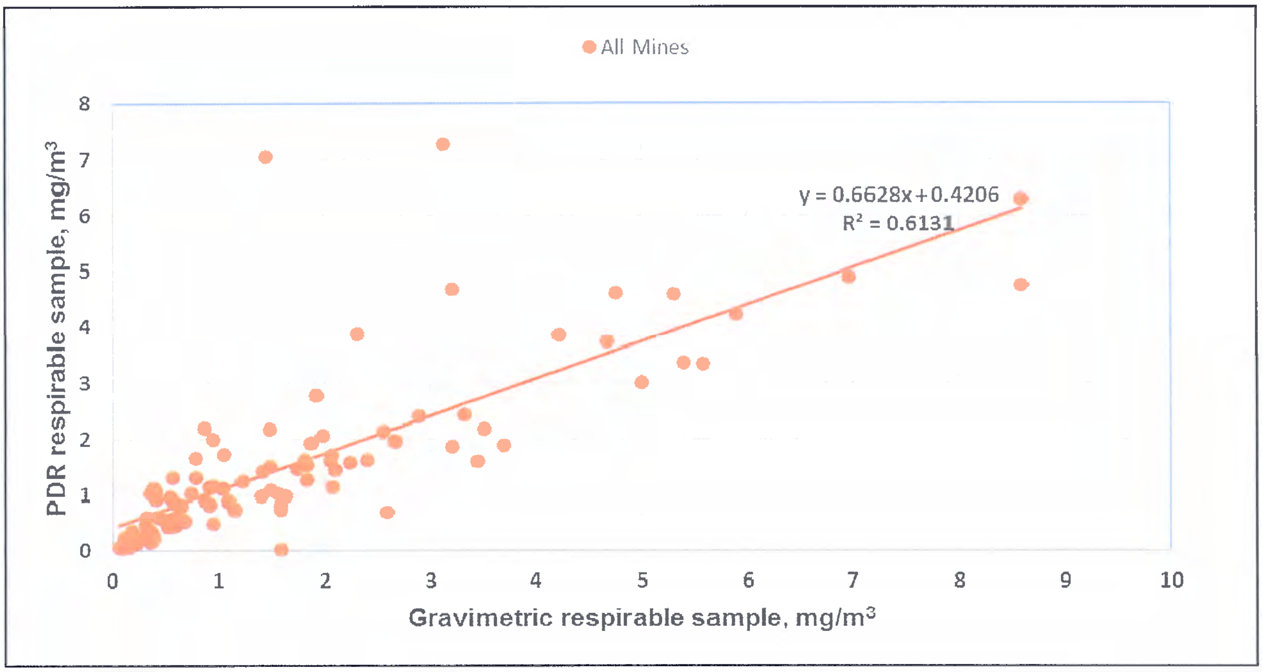

Figure 9: Combined relationship between pDR1000 and HD sampler from all mines.

4.1 *PDM 3700 Evaluation in Coal Mines*

The PDM3700 was trialed in Queensland coal mines for the first time with specific permission and used where the gas levels were below 0.5%. Despite the PDM3700 being an MSHA approved tool used in the USA, it could not be used in Australian mines due to it not meeting the Queensland intrinsically safety (IS) electrical approval. Figure 10 shows an example of the dust real-time monitoring results, together with ventilation and gas monitoring data for an operating longwall mine. It can be seen that at the end of shift dust concentration (2.29 mg/m^3) is below the dust limit of 2.6 mg/m^3 (for an extended shift scenario). What can be inferred from the real-time data is that 35% of the dust concentration came from only two exposure spikes, i.e., 0.76 mg/m^3 of the 2.29 mg/m^3 shift average. This information is very useful considering the current focus on dust management and validating historic results.

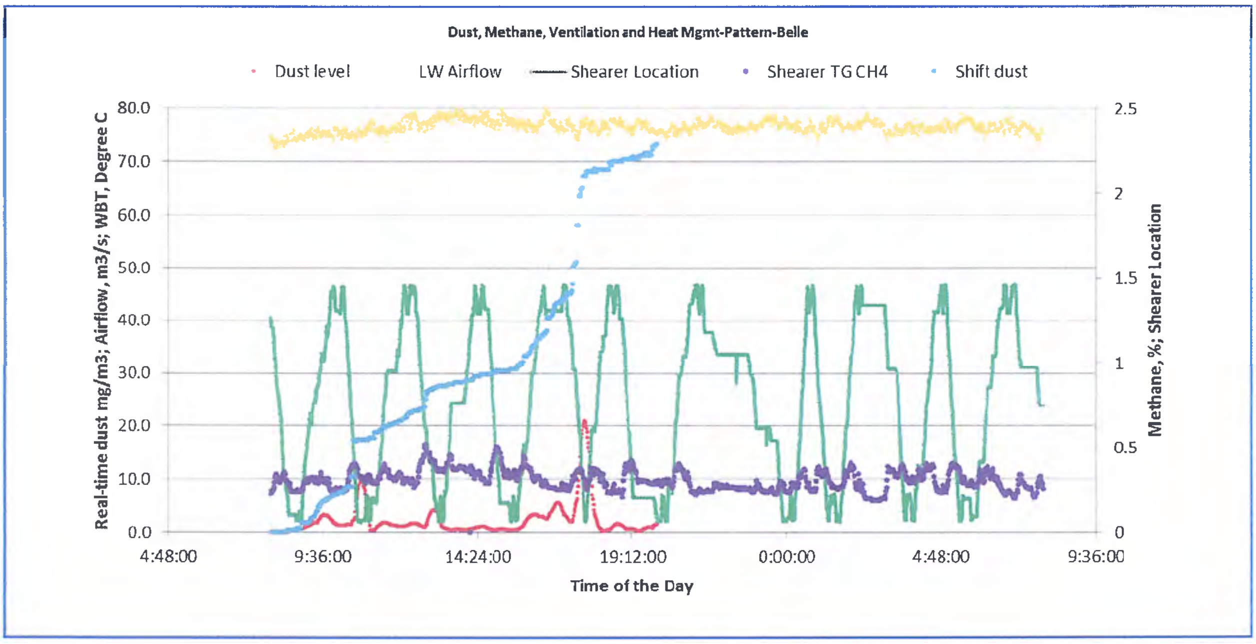

Figure 10: Example of real-time monitoring of dust (PDM3700), methane, and ventilation data in an operating longwall.

The above data demonstrates the usefulness of having additional features such as real-time longwall ventilation flow, ventilation and gas drainage effectiveness during the entire shift to manage dust and methane during a production shift in modern longwall panels. Although, there is a potential for the PDM3700 monitor to provide thermal stress data such as the dry bulb temperature (DBT), relative humidity and barometric pressure data that is already inherent to the PDM3700 monitor. These specific data features need to be further verified through the original manufacturer. In summary, longwall critical hazard and control parameters monitored in real-time—dust, methane, air flow and temperature were evaluated using PDM3700 monitor as personal sampler for continuous miners and longwalls.

The PDM3700 continuous dust monitor represents a significant step change for personal exposure monitoring consistent with the mass-based principle as a surrogate measurement of worker exposure. However, there appears to be concerns for its application in Australian mining community who are of the opinion that the PDM3700 is not complaint and does not meet the AS2985 requirements, i.e., instrument is not able to provide the mass based dust exposure results. In the USA, NIOSH has conducted various peer reviewed scientific studies over a decade with approved methods that are consistent with the AS2985. Prior to MSHA approval, NIOSH demonstrated that the CPDM is an accurate instrument that meets the NIOSH Accuracy Criteria and, therefore, can be used as a compliance instrument (Volkwein et al., 2006). For example, the largest data set with 955 samples in US coal mines by having miners wear a CPDM and a CMDPSU (gravimetric sampler) concurrently was received by the MSHA. In order to determine the bias, NIOSH reviewed the data set and concluded that those results support those published by NIOSH PDM studies. The results showed that the average concentration measured by the traditional US gravimetric sampler (CMDPSU), 0.83 mg/m^3, was virtually identical to the PDM3700 (CPDM) average value of 0.82 mg/m^3. In addition, NIOSH further concluded, reviewing the 955 samples, that there was no statistically significant difference between the data sets, and that the bias between the CPDM and the approved CMDPSU is zero. While the author does not have the data to assess the differences between the US results, it is planned to evaluate the PDM units in Australian and South African conditions and planned to publish the data and the findings.

5 STASTICAL ANALYSES

This section of the paper discusses the analyses of the data using appropriate statistical techniques. All the dust concentration data for each sample set were tested for Anderson-Darling normality and it is evident that the data do not follow a normal distribution. Preliminary data analysis indicated that loge–transformed data gave an improved fit of the normal distribution. Therefore, for the statistical analysis, loge(Ho) and loge(Ha) were compared (paired t-test). The subscripts, Ho (HD sampler) and Ha (real-time sampler), are the dust concentration values measured using the identified personal sampling instruments in the sample pair (random) at various test mines. Hypothesis tests were carried out at each of the mines to test the sampling environment (gold, diamond, platinum and coal). The null and alternative hypotheses for the tested sample pairs were:

$$\text{Ho: } \mu\text{diff} = 0$$
$$\text{Ha: } \mu\text{diff} \neq 0$$

In the paired t-test, hypothesis Ho states that the mean difference in concentration values (transformed values) between side-by-side personal instrument pairs is equal to zero. On the other hand, the alternative hypothesis states that the two personal dust-monitoring instruments positioned side by side in fact measured different mean concentration levels or the difference was not equal to zero. For this analyses, a statistical parameter of 95% confidence level was chosen. The results of the paired t-test statistical analyses are given in Table 2. For the analyses, a cut-off p-value of 0.05 was used.

Table 2: Results of paired t-test (on transformed values)

Statistic	Mine	Sample Pair ($H_{SA\text{-}pDR1000}$)
95% LCL	Gold	-0.118
	Platinum	-0.606
	Diamond	-0.086
	Coal	0.231
95% UCL	Gold	0.137
	Platinum	0.096
	Diamond	0.282
	Coal	0.459
t-statistic	Gold	0.15
	Platinum	-1.48
	Diamond	1.14
	Coal	6.33
P–value	Gold	0.88
	Platinum	0.15
	Diamond	0.27
	Coal	0.00
Hypothesis (Accept or reject)	Gold	Accept
	Platinum	Accept
	Diamond	Accept
	Coal	Reject
Sample size	Gold	31
	Platinum	30
	Diamond	15
	Coal	30
Overall statistics	All mines	
95% LCL		-0.075
95% UCL		0.160
t-statistic		0.72
p-value		0.47
Sample size		106
Hypothesis (accept or reject)		Accept

From Table 2, it is observed that, for all test mines, the measured mean personal dust levels from each pair of HD and pDR1000 real-time dust monitors did not differ significantly and the null hypothesis is accepted, except in coal mine samples. A paired t-test was performed on the combined data of all four dust monitors to determine whether there was a statistical difference in the results obtained from the HD sampler and the pDR1000 dust monitor. The result of the paired t-test was a test statistic with 105 degrees of freedom, $p = 0.47$ indicating no significant difference between the measured mean concentration levels using the HD sampler and the real-time pDR1000 dust monitor side by side. Finally, from the above analysis it is concluded that the pDR1000 is the instrument with the most potential for use in the mines ($p = 0.47$), based on intensive field evaluations in all commodity mining types.

5.1 *Accuracy Criteria*

Table 3 shows summary statistics of the respirable dust values obtained from the side-by-side comparison of the pDR1000 and HD samplers as measured in the coal, gold, platinum and diamond mines by three different units. From the summary statistics, it is observed that there is no clear relationship between accuracy and the measured concentration levels. Overall, the CV of the ratio between the sampler dust concentrations was below the NIOSH accuracy criteria (except in 3 cases out of a total of 18). Overall, the pDR1000 real-time dust monitor failed to meet the NIOSH accuracy criteria and its correction factor for the test mines ranged between 0.53 and 1.74. This could be because of the dust particles sampled, which were poly-disperse and not homogeneously mixed in the mine atmosphere or in the micro environment where the pDR1000 was located, or it could be due to variations in the particle sizes of the sampled dust that is uncontrollable.

Table 3: Summary of the correction factors for the pDR1000 and HD samplers in all mines

Type	pDR1000#	Dust*	PDR/HD	samples	SD	RSD
	#	mg/m3	P3	#		
Coal-A	P1	2.005	0.731	5	0.045	6.15
	P3	1.697	1.131	5	0.561	49.60
	P2	1.431	0.628	5	0.063	10.03
Coal-B	P1	3.516	0.576	5	0.095	16.49
	P3	4.836	0.759	5	0.160	21.08
	P2	2.504	0.640	5	0.230	35.93
Gold-K	P1	0.533	0.862	5	0.252	29.23
	P3	0.483	0.859	5	0.188	21.89
	P2	0.491	0.929	5	0.283	30.46
Gold-GN	P1	0.774	1.210	7	0.681	56.28
	P3	0.889	1.290	7	0.341	26.43
	P2	0.381	0.951	2	0.132	13.88
Platinum	P1	0.731	1.683	14	0.558	33.16
	P3	0.840	1.809	14	1.116	61.69
	P2	2.09	0.144	2	0.173	120.0
Diamond	P1	4.480	0.964	5	0.425	44.09
	P3	3.733	1.095	5	0.342	31.23
	P2	2.212	0.813	5	0.246	30.25
Overall		2.665	0.744	30	0.302	40.59
All non coal		1.293	1.266	76	0.721	56.95
All mines		1.681	1.118	106	0.673	60.19

* Higgins-Dewell gravimetric value

6 DISCUSSIONS

The following paragraphs discuss the operational experiences on the use of light-scatter based and gravimetric based (PDM3700) real-time dust monitors. The results of the study showed that the respirable dust mass measurements obtained with the passive pDR1000, employing the principles of light scattering, were less likely to be related (average relationship) to the measurements obtained with the gravimetric HD sampler. However, when operated in active mode, the response of the real-time instruments can be adjusted so that the mass concentration determined by the light-scattering system is equivalent to that of gravimetric sampler. In addition, the range of correction factors is wide and, therefore, the likelihood of being able to use a real-time monitoring instrument as a "stand-alone" unit is not advisable. However, the instruments can be used for engineering dust control purposes when used in active mode.

In general, the correction factors of the real-time direct-monitors can be explained by the size-dependent light-scattering characteristics of the instruments with respect to any of the respirable size-selective sampling conventions. According to the ISO/CEN/ACGIH convention, an "ideal sampler" cuts off the particles larger than 10 µm. However, the real-time monitoring instruments may measure or detect particles larger than 10 µm due to potential gravimetric sampler inefficiency a shortcoming from which the PDM3700 also suffers. However, this should not be the sole criterion for its use in the mines, which have been using the same gravimetric samplers for over 50 years.

Dust-monitoring instruments also depend on air movement to move the air into the sensing zone of the instrument. The orientation of the gravimetric sampler (and that of the wearer) may also give "biased" results, depending on the particle size. All samplers were exposed to similar temporal and spatial environmental conditions for underground evaluations. Therefore any differences in their responses were due to the sampling characteristics of the dust monitors alone. Time had no significant influence on the sensors or lenses, or on the correction factor of the real-time monitors. In the case of the real-time monitoring instruments, the localised air movement was solely responsible for introducing dust particles, respirable or otherwise, into the sensing chamber (i.e. passive sampling) unlike the active sampling gravimetric-type instruments. For example, in a controlled chamber environment, when three different passive light-scatter pDR1000 units were exposed to the same environmental conditions, each of the units recorded different peak dust levels (Belle, 2006) indicating the complexity involved in the exposure monitoring and its assessment. As a way forward, gravimetric based PDM3700 continuous real time dust monitor is a significant step change in quick personal exposure assessment to improve the health of mine workers.

7 CONCLUSIONS

This paper discusses comparative results between a passive light-scattering based real-time (pDR1000) monitor and the HD type gravimetric sampler that is used in the CPDMs and operated in accordance with the international size-selective curve as a 'true sampler.' The following conclusions can be drawn from the field evaluation of passive pDR1000 in South African mines and gravimetric based CPDM (PDM3700) in Australian longwall mines to assess the worker dust exposures:

- The field results of the study have showed that the dust levels measured with the passive pDR1000 units were significantly different to the gravimetric sampler data, unlike results observed in a laboratory-controlled study where it was previously noted (Belle, 2002) that side-by-side comparison of three pDR1000 units in passive mode and a HD cyclone indicated there to be no significant statistical difference in measured dust levels.
- The real-time Hund tyndallometer results showed a very poor correlation and significantly underestimates the dust measurements. The Hund was therefore recommended to be used always in conjunction with the gravimetric sampler.
- The passive portable real-time dust monitor (pDR1000) proved to have the highest potential as a real-time dust monitor for engineering control evaluations. However, using a real-time monitoring instrument as a stand-alone unit for personal exposure level compliance is not recommended.
- The mass based PDM3700 monitoring experience in Australian coal mines was found to be of significant benefit and to provide quick turnaround results for personal exposure assessment as they provide measurements in real-time to address immediately any dust management deficiency issues. For example, when the exposure results reach 80% of the limit, the PDM3700 instrument will warn and enable the worker to exit the workplace.
- The benefits of the PDM3700 are so substantial that having just a few such units in each mine can replace the current suite of gravimetric-based sampling instrumentation for all types of mining conditions - where currently, results of exposure may take up to a week to yield results. Therefore, its approval for use in Australian mines must be treated as a priority to benefit coal mine workers.
- There is a definite potential for using the PDM3700 as a legislative requirement in Australian mining industry. Prior to implementation, it is recommended that the PDM3700 shall incorporate at least additional '10 sec' measurement records against the current '1 minute' average dust data recordings that is deemed to be a shortcoming in personal exposure assessment but is acceptable as engineering dust management tool.
- Lastly, it is noted that the preliminary investigations related to the quartz analyses have indicated possible interference by the filter material used in the PDM3700. Therefore, additional confidence must be gained using the newly developed dust filter, through further investigations, in order to make use of the full potential of the PDM3700 as a real-time monitoring instrument.

8 ACKNOWEDGEMENTS

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QoN Attachment 9

Coal Workers Pneumoconiosis Inquiry
Public Hearing - Tuesday 31 January 2017 Brisbane
<u>Questions Taken on Notice and Additional Information</u>
Anglo Coal

Additional Information 2 (P.43)

'Position Paper – PDM 3700', 26 September 2016 (document) – Anglo American / Glencore

'Position Paper – PDM 3700', 26 September 2016 (Powerpoint) – Anglo American / Glencore

UNCLASSIFIED

POSITION PAPER

PDM3700

Use of the PDM3700 in atmospheres containing up to 1.25% methane and as a compliance measurement instrument

28 September 2016

AngloAmerican

GLENCORE

POSITION PAPER

PDM3700

The re-emergence of Coal Workers Pneumoconiosis (CWP) in Queensland has reinforced the importance of respirable dust mitigation strategies and measurement techniques. While significant energies are being focused on work procedures, dust suppression systems, and respiratory protective equipment, the only guaranteed means of preventing CWP is ensuring coal mine workers are not exposed to unacceptable levels of respirable dust. Several varieties of dust monitors exist on the market; however, most rely on older technologies which cannot match the accuracy and precision of the proven technology employed in the PDM3700. Industry research has found that the PDM3700 is a personal gravimetric sampling device which should be viewed as a key component of the strategies employed to prevent coal mine workers from developing CWP.

This paper is a collaborative effort between Anglo American Metallurgical Coal and Glencore Coal Assets Australia and seeks to give early information to the Coal Mining Safety and Health Advisory Committee on the introduction of the PDM3700 to Queensland underground coal mines by:

- Demonstrating that through continuous feedback to coal mine workers, the PDM3700 is the most suitable technology currently available to prevent over-exposure to respirable dust
- Establishing an urgent case for the PDM3700 to be used in concentrations of methane up to 1.25%
- Establishing groundwork for the PDM3700 to eventually be used as an approved compliance gravimetric dust sampling instrument

Background

Sampling and analysis technologies for respirable dust have seen few step-change improvements over time. From 1937 until the 1960's, the most common dust sampling instrument was called the midget impinger – a hand-cranked unit which captured particles for the purpose of counting them separately. In the 1960's, new information about how the body contracts CWP was discovered which led to the development of the personal gravimetric sampler (Kissell et al., 2002) as we know and use it today in Queensland and New South Wales. For nearly 50 years this device has served the mining industry as the standard for sampling respirable dust. Several dust-measuring technologies have been evaluated as candidates to replace the personal gravimetric sampler, but most have been found inadequate because of poor accuracy or excessive size and/or weight (Kissell et al., 2002).

The PDM3700 has its origins in the United States coal industry and the unit's predecessor was first used in the early 2000's. The PDM 3700 is a Tapered Element Oscillating Microbalance (TEOM) gravimetric device which provides continuous measurement and display of respirable dust exposures and is the only commercially available unit which meets the legislated requirements for continuous compliance determination of personal dust exposures in the United States mining industry. That is to say, the U.S. Mine Safety and Health Administration (MSHA) regard samples collected by this device as equally compliant as the samples collected by the same personal gravimetric units used in Queensland.

The development of this instrument marks the first step-change improvement in personal dust monitoring in over half a century, and because the PDM3700 provides continuous indication of exposure levels to coal mine workers, it allows the mining industry to shift sampling regimes from being lagging indicators to leading indicators.

In this context, it is proposed that the PDM3700 be used in Queensland coal mines in atmospheres containing up to 1.25% methane until electrical certification is achieved and ultimately be recognised as an approved compliance gravimetric dust sampling instrument.

Technical Considerations

With a variety of personal dust monitors commercially available, it is important to discuss the limitations of the technologies employed as well as highlight the features of the PDM3700 and how it is distinguished from other devices.

Light-Scattering Photometry

In the 1980s light-scattering photometry technology provided operators with a fast, lightweight device to provide instantaneous feedback on dust levels. These devices, such as the Casella Microdust Pro, Grimm Mini-LAS, Hund TM II, Kenelec DustTrak, and SIDEPAK AM510, use a light source and sensor to measure the amount of light reflected/refracted off of particulate matter suspended in the sample atmosphere. In the calibration of these units, that measurement of light is then compared to the mass of the material that has been collected to establish a relationship between the two. It is then assumed that a given measurement of light is indicative of a mass value. While these units are still used today to provide information on relative dust levels (i.e. more dust versus less dust), they are indicative only and have several limitations:

1. They do not have the accuracy required to be considered a compliance tool as they can vary by as much as a factor of two (Page and Jankowski, 1984).
2. Measurements from light scattering devices can vary significantly from gravimetric devices due to differences in dust types, dust size distribution, and environmental conditions (Belle, 2006).
3. All particles of a given size are treated as though they have the same mass. However, in the same way that a ping-pong ball and a golf ball have similar diameters with dissimilar masses, coal particles vary in density.
4. They are highly susceptible to error from humidity and particle agglomeration.
5. Calibration to a measured gravimetric sample is required on a continual basis to ensure results are indicative of true mass measurements.

Although these units can provide beneficial information, their susceptibility to error prevents coal mine workers from confidently assessing their exposure levels. Likewise, they fail as candidates for acceptance as compliance instruments.

Tapered Element Oscillating Microbalance (TEOM)

Originally designed for space related programs, TEOM technology utilizes an inertial mass weighing principle (Patashnick et al., 2002). Basic physics, through Newton's Second Law, establishes that the mass determined dynamically through this technique is identical to the same mass determined statically through a strictly gravitational method. The PDM3700 is based on this principle. In these units, a sample is drawn through a filter sitting atop a hollow tapered tube which is oscillated at a specific frequency. As mass collects on the filter, the frequency of the system decreases; by measuring the frequency change, the accumulated mass is measured. There are many benefits of the PDM3700, specifically:

1. Samples are actively drawn into the unit at 2.2 L/min and separated via cyclone which is in alignment with the ISO 7708 respirable dust size-selective curve as referenced in AS2985.
2. Samples are conditioned by a heating element to reduce the impact of environmental conditions, specifically humidity.
3. Mass is directly measured, not inferred as in light-scattering photometry.

4. Cumulative mass concentration is visible to operators via LED display and also shown as percentage of allowable limit which empowers them to take appropriate action to effect controls that ensure they remain under exposure thresholds.
5. Mass and other data is continuously measured and can be easily downloaded at the end of the shift to show concentrations throughout the period as opposed to the current, time-consuming laboratory process.
6. The OEM has manufactured an alternative filter which would allow the determination of silica content from the same sample.
7. Temperature and barometric pressure are continuously measured and recorded by the unit.

History of the PDM3700

In 1992, the United States Secretary of Labour's Coal Mine Respirable Dust Task Group issued a report which concluded that continuous monitoring of the mine environment and dust control parameters offered the best long-term solution for preventing occupational lung disease among coal miners. It specifically recommended development of monitoring technology capable of providing both short-term as well as full-shift concentration measurements (MSHA Final Rule, 2014).

The PDM3700 is a result of the two-decade long research and development program set forth in 1995 by MSHA and National Institute for Occupational Safety and Health (NIOSH) (Cantrell et. al., 1996). Its predecessor, the PDM3600, underwent rigorous underground field testing in excess of 8,000 hours to prove the accuracy of the technology and was proven to be both an accurate and precise unit which can withstand the harsh environment associated with mining (Volkwein et. al., 2004; Volkwein et. al., 2006). The outcome of this work is a sampling device that permits a single, full-shift measurement of respirable dust which "adequately assures that no miner will suffer a material impairment of health, on the basis of the best available evidence; uses the latest available scientific data in the field; is technologically and economically feasible; and is based on experience gained under the Mine Act and other health and safety laws" (MSHA Final Rule, 2014).

Since February 2016, mine operators in the United States mining industry have been required to use a Continuous Personal Dust Monitor (CPDM) for compliance determination of personal dust exposures. The PDM3700 is the only device commercially available which meets the requirements of Continuous Personal Dust Monitors as outlined in the U.S. Code of Federal Regulations.

The images below show the PDM3700 output screen and two examples of the information which can be displayed on the screen.

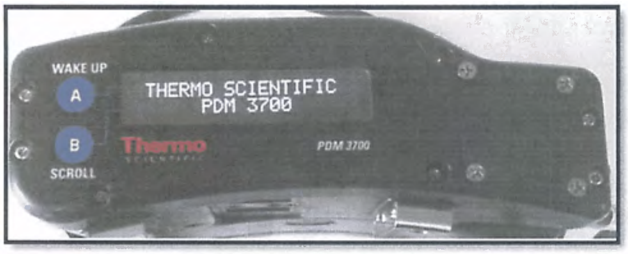

Figure 1: PDM3700 Unit

```
SHIFT LIMIT    2.00
PERCNT OF LIMIT  25%
```

```
30 MIN CONC    0.22
  CUM1 CONC    1.21
```

Figure 2: Display Information

Safety and Health Considerations

Medical

Studies have shown that the risk of Progressive Massive Fibrosis (PMF) among miners without evidence of simple CWP grows with increasing cumulative exposure to respirable coal dust (DHHS, 1995). Likewise, coal miners may accumulate lung dust burdens of more than 10 mg/g of lung over a working lifetime, even at current exposure limits (DHHS, 1995). Therefore, the most critical element in preventing CWP and PMF is empowering the workers at the coal face with information so they can take action to reduce their personal exposure to respirable coal dust with immediacy – not weeks after the event, as with current gravimetric samplers.

Current sampling technologies are incapable of providing coal mine workers with accurate readings of their exposure levels. Ordinary gravimetric sampling units were never designed to serve as continuous monitors, and the information from these units can generally take several days to weeks to get back to the coal mine worker. Conversely, the array of light scattering photometry units which exist are incapable of providing information with the accuracy required to make meaningful determinations. This is critical because of the non-linear relationship between predicted prevalence of disease and mean dust concentrations, as outlined in the 1995 NIOSH report. It showed that reducing exposure by even 0.5 mg/m^3 can significantly reduce the prevalence of simple CWP and PMF over a 35-year working lifetime. More clearly stated, **"every decimal point of exposure matters."** The fundamental advantage of the PDM3700 over all others is its ability to continually measure and display a miner's cumulative respirable dust exposure with unparalleled accuracy. This single piece of data provides coal mine workers with the information they need to play a leading role in ending CWP before it regains a strong foothold in Queensland.

Electrical Limitations

The PDM3700 does not currently meet the requirements for certified portable electrical equipment under Recognised Standard 1. Although the device has intrinsically safe explosion protection suitable for use in U.S. underground coal mines, the conformance of the explosion protection has not been certified by an accredited Australian testing station, as required under legislation. It should be noted that the explosion protection of the device has also been verified and approved for use in underground coal mines in South Africa by Mining and Surface Certification (MASC) under the South African legislation.

Nonetheless, in its current configuration, the PDM3700 can only be used under uncertified portable electrical equipment provisions in Recognised Standard 1. The standard precludes its use in locations where the percentage of methane in the general body of air exceeds 0.5% by volume. This restriction typically prevents the unit being used near the working face on many of our QLD longwall faces – the area where workers are at the greatest risk of exposure to respirable dust.

Proposal for Use in Atmospheres of up to 1.25% Methane

It is the intent of Anglo American and Glencore to pursue certification for the PDM3700 through a nationally accredited testing station. The most promising path to certification appears to be based on the assessment conducted by MASC in South Africa as both Australia and South Africa use common IECex standards for compliance of explosion protected electrical equipment. The findings from their recent certification process were that the PDM3700 is suitable for intrinsically safe Ex ib certification for use in South African underground coal mines. MASC has now been engaged to conduct a conversion to an IECex test report as required by the Nationally Accredited Testing Stations; however, this process may take up to 12 months which guarantees that miners at the working face are without the information they desperately need to avoid overexposure to respirable coal dust.

As a temporary measure, until full certification can be gained, it will be requested that variation be made to Recognised Standard 01 to allow use of the PDM3700 in atmospheres which contain less than 1.25% methane. The device would be subject to the same requirements currently placed on uncertified portable electrical equipment with the only difference being the methane concentration trigger level for withdrawal.

While risk assessments and engineering reviews are currently underway to support the requested change to Recognised Standard 01, a cursory review indicates that an acceptable level of risk can be maintained when consideration is given to the following points:

- The longwall district has the greatest potential for methane concentrations at this level; however, it is typically the most heavily monitored area of a mine.
- The likelihood of exposing these units to concentrations above 1.25% is extremely low as they would be accompanied by a hand-held methane monitor and withdrawn at 1.25%.
- Typical methane concentrations in normal work areas across most longwall faces range from 0.1% at the maingate to 1.0% at the tailgate drive.
- The unit has been certified as Intrinsically Safe in both the United States and South Africa.
- MSHA's tests for Intrinsic Safety requires circuitry to be tested under conditions judged to simulate the most hazardous probable faults or malfunctions and in the most easily ignitable mixture of methane and air (8.3%) (30CFR18.68, 2008).
- Time in service in the United States and South Africa provides a history of use in gassy underground mines.

Once IECex certification is achieved, it will be proposed that the reference to the PDM 3700 in Recognised Standard 01 be removed.

Compliance Sampling Considerations

Ordinary Gravimetric Sampling Technology

Currently, all respirable dust samples in Queensland and NSW coal mines are governed by AS2985-2009: Workplace Atmospheres – Method for Sampling and Gravimetric Determination of Respirable Dust (originally written in 1987). This standard outlines several specific requirements for respirable dust samples including:

- Approved cyclones used to separate the respirable fraction of dust
- Specifications on micro-balance
- Flow rate to be used to collect the sample
- Treatment and handling of sample units and filters
- Calibration of equipment used
- Reporting requirements

Generally, in the process a filter is pre-weighed in a certified laboratory, a sample is collected on the filter, and the filter is post-weighed. Until recently, this system was largely accepted as the benchmark for personal dust monitoring; however, there are some significant drawbacks to the method:

1. It can take days to weeks to receive the exposure results from the testing agency, thereby preventing the operations from understanding exposure events and implementing effective control measures.
2. Results are a single value representing the weighted average exposure for the shift (e.g. 2.1 mg/m^3) which provides no information on contributory events during the working shift.
3. There is no means for operators to see their exposure profile continuously throughout the shift and therefore effect changes to prevent overexposure.

Relying on a lag-indicator, especially one with such a magnitude of delay, is a disservice to our coal mine workers. Regularly, respirable dust exceedance interviews yield little to no benefit as operators cannot recall with enough detail the events that occurred on the shift in question since most interviews are on average more than two weeks after an exceedance event. Similarly, the fact that coal mine workers continue to exceed exposure limits gives rise to a strong case for adopting a technology that can prevent exceedances in real time. In short, an operator cannot be expected to take action to ensure he is not exposed to an unacceptable level of risk unless he has the information to do so.

AS2985 and Proposed Recognised Standard: Monitoring Respirable Dust in Coal Mines

As previously stated, all respirable dust samples are governed by AS2985. This document, by its prescriptiveness, effectively prohibits the use of TEOM technology based devices for compliance sampling. In reality, the PDM3700 is fundamentally compliant with the standard. Its only definitive departure from the standard is the means of calculating the final dust concentration, as shown in Table 1.

Table 1: Comparison of Sampling Technologies

Criteria	Standard Gravimetric Sampling (AS2985)	PDM3700 (TEOM Technology)
Specified Size-Selective Sampling Cyclones	Higgins-Dewell, Simpeds, Aluminium	Uses Higgins-Dewell cyclone
Flow Rate	2.2 L/min (2.5 L/min for aluminium cyclones)	Controlled to 2.2 L/min
Flow Rate Accuracy	± 5%	± 2.5%
Filter Diameter	25 mm preferred, 37 mm permitted	15 mm
Timing Device	Required	Include in unit
Means of Mass Measurement	Five or six-place microbalance (10 µg or 1 µg divisions, respectively)	Tapered Element Oscillating Microbalance
Frequency of Mass Measurement	Filters weighed once before and once after sample taken	Continuous measurement via TEOM
Reporting	Final dust concentration reported as end of shift average	Dust concentration reported continuously throughout shift with end of shift average also reported

Additionally, the Proposed Recognised Standard: Monitoring Respirable Dust in Coal Mines, as recently circulated for comment, specifically states that TEOM technology is not in accordance with AS2985. For the PDM3700 to be used as a compliance tool, such wording would need would need to be amended. To this end, a recommendation has been made to the sub-group responsible for the proposed recognised standard that the wording of the standard should not preclude the use of new technologies.

PDM3700 as a Compliance Tool

Prior to the release of the PDM3700, the U.S. Secretary of Labour's Dust Advisory Committee unanimously recommended in a report that continuous personal dust monitoring (CPDM) technology, once verified as reliable, be broadly used by MSHA for assessing operator compliance efforts in controlling miners' dust exposures and for compliance purposes (MSHA Final Rule, 2014). With respect to compliance sampling, MSHA later found that the NIOSH approval of the PDM3700 further demonstrated that it is an accurate compliance sampling device for determining the concentration of respirable dust in coal mine atmospheres (MSHA Final Rule, 2014; Volkwein et. al., 2004; Volkwein et. al., 2006).

The technology used in the PDM3700 represents a substantial improvement in the way sampling for respirable dust is conducted. Not only can this device provide coal mine workers with immediate feedback on exposure levels, it can also allow operators to provide conclusive respirable dust results to stakeholders in a matter of hours as opposed to days or weeks. The speed of this information is also valuable in interviews with coal mine workers to understand the effects their actions have on their dust exposure throughout the shift.

Another benefit of the unit is that sampling can quickly be arranged allowing large volumes of accurate data to be collected in a short period of time. However, because the results from the PDM3700 currently are not considered valid for reporting purposes, the industry is missing out on a large data set which could be used to gain further insight into respirable dust exposures and control effectiveness.

The industry now has a device which can measure compliance to respirable dust limits while simultaneously providing crucial information to coal mine workers which enables them to take action when exposed to elevated dust levels or reaching their daily exposure limit. It is therefore requested that individual SSEs are supported in their decision to use the PDM3700 as a compliance sampling instrument.

Proposal

In view of the years of research and development that have gone into this unit, the extensive field trials of the predecessor to the PDM3700, and the significant opportunity afforded by the PDM3700 for coal mine workers to understand dust concentrations throughout the shift, the following strategies are proposed:

- Seek changes to Recognised Standard 01 to allow the PDM3700 to be used in atmospheres containing up to 1.25% methane until electrical certification is gained.
- Assist SIMTARS or another accredited body with expertise and manpower to expedite the certification of the PDM3700 and provide a clear timeline for the process.
- Ensure that the proposed "Recognised Standard: Monitoring Respirable Dust in Coal Mines" does not preclude TEOM technology as an accepted means of gravimetric sampling.
- Seek modification to AS2985 to include provisions for the PDM3700 as an approved means of compliance sampling for respirable dust.
- Actively support SSEs should they elect to use the PDM3700 as a compliance sampling instrument.

Summary

Technological advancements in conjunction with a working group commissioned by the United States government have delivered an instrument capable of accurately and reliably measuring respirable dust in underground coal mines which, if embraced, can be used to provide the Queensland mining industry with a greater level of safety. While the current process for the collection and measurement of respirable dust has served the mining industry well for the last half-century, the benefits of the PDM3700 and its proven performance in the United States gives weight to a compelling case for change to the next generation of sampling instruments. The PDM3700:

- **Provides coal mine workers** with immediate and accurate feedback of their cumulative shift exposure allowing them to take corrective action and ensure they are not exposed to unacceptable levels of dust.
- **Provides operators** with continuous monitoring data for enhanced analysis of shift exposures.
- **Provides regulators** with significantly more data, without compromising accuracy, to better understand control effectiveness at operations.
- **Provides all stakeholders** with confidence that sampling is done in a consistent and accurate manner with faster turn-around of results.

For these reasons, the working group from within the Queensland coal mining industry seeks to give you an understanding of our activities; petitions your in-principle support for the PDM3700 as a personal dust monitoring device and ultimately as an approved means of gravimetric sampling; and seeks modification of draft recognised standards to facilitate the objectives of this proposal.

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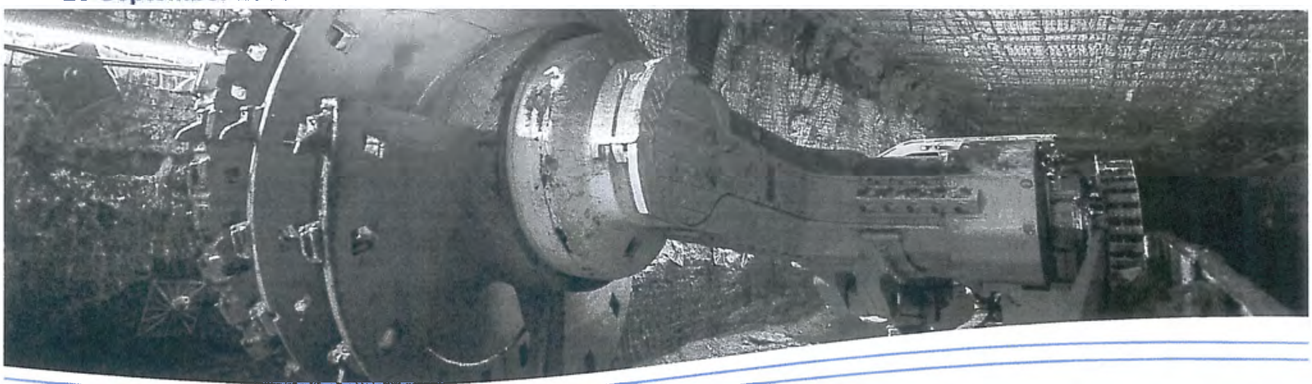
POSITION PAPER

PDM3700

Use of the PDM3700 in atmospheres containing up to 1.25% methane and as a compliance measurement instrument

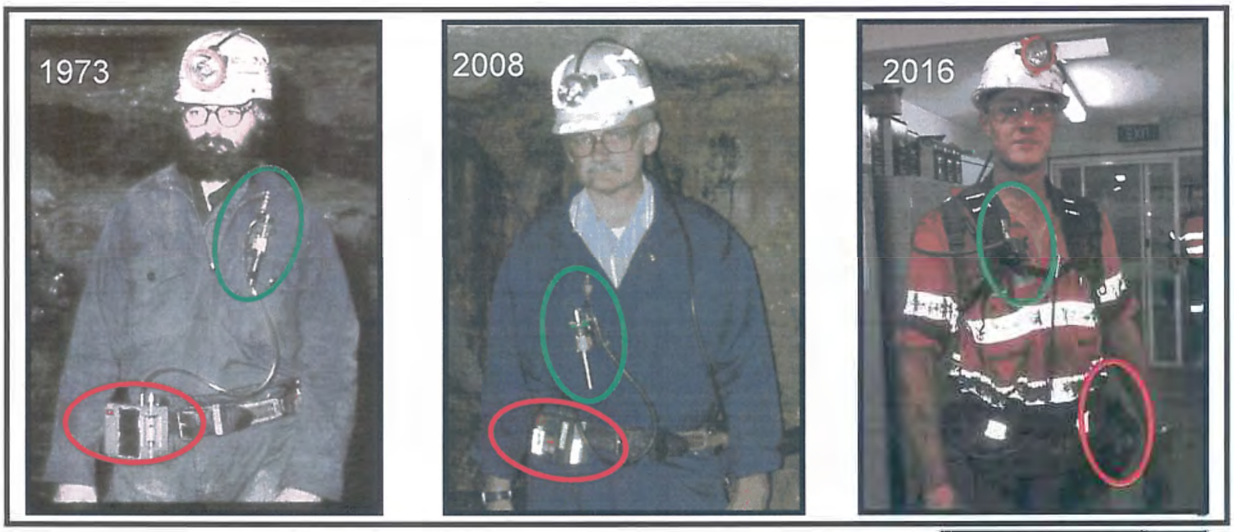
28 September 2016



BACKGROUND

Respirable Dust Sampling

- Sampling methodology largely unchanged since the 1960s
- Still the standard for sampling today
- Technology has advanced all around us, but has lagged in this field



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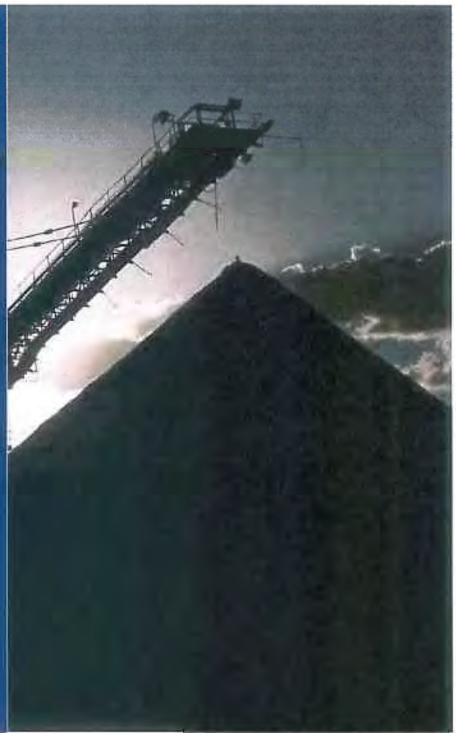
OVERVIEW

- Limitations of current sampling technology
- Overview of the PDM3700
- Safety, Health, and Compliance Considerations
- Speed Humps for Change



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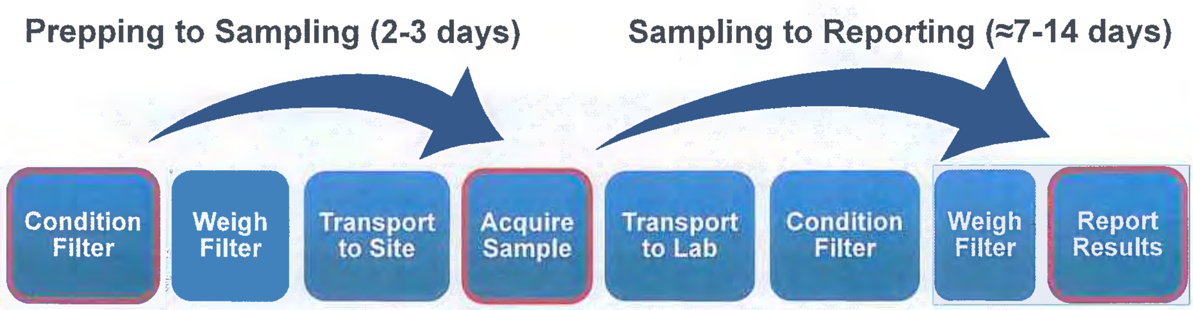
CURRENT PROCESS

Standard Gravimetric Sampling



- Results are reported as single value for shift
- Operators cannot see exposure values during shift
- Opportunity for meaningful review of
- failures is compromised

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NEW TECHNOLOGY

Tapered Element Oscillating Microbalance (TEOM)

History

- First developed for space program
- Technology incorporated into wearable units thanks to development from NIOSH and MSHA
- Field tested and proven in the U.S.

Principles of Operation

- Samples collected on filter which rests on end of oscillating hollow tube
- Mass measured (not inferred) continuously according to:

$$Change\ in\ Mass = K_0(\frac{1}{f_f^2} - \frac{1}{f_i^2})$$

[redacted]

Pros

- Gravimetric means of mass determination using Higgins-Dewell cyclone
- Provides absolute dust concentrations
- Highly accurate and precise
- Consistent maintenance of flow rate at 2.2 l/min

Cons

- No certified units in Australia
- Cost and maintenance considerations

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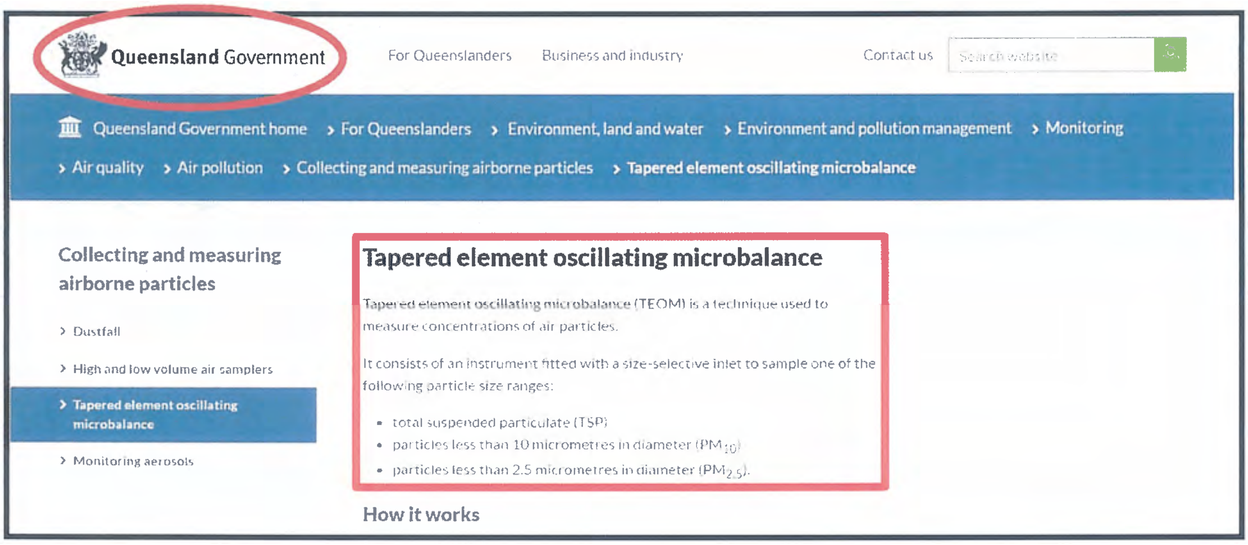
NEW TECHNOLOGY

Tapered Element Oscillating Microbalance (TEOM)

- Queensland currently recognises TEOM technology as an accepted means of dust monitoring

- PDM3700 is the first-ever wearable TEOM unit



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BENEFITS

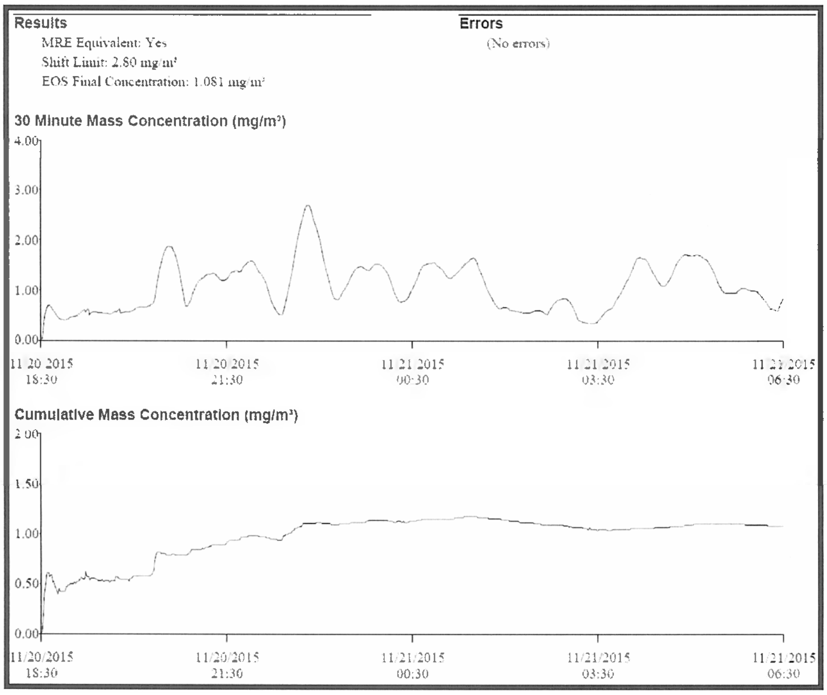
ThermoScientific™ PDM3700

What We Recognise in the PDM3700

- Immediate exposure information for faster decision-making
- Continuous monitoring for enhanced sampling detail
- Improved data analysis capabilities
- Accuracy to give coal mine workers confidence

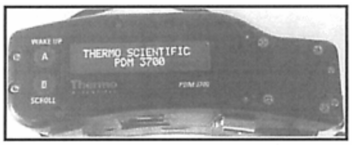
BENEFITS

ThermoScientific™ PDM3700



Dust card available after download (left)

Output screen on monitor (below)

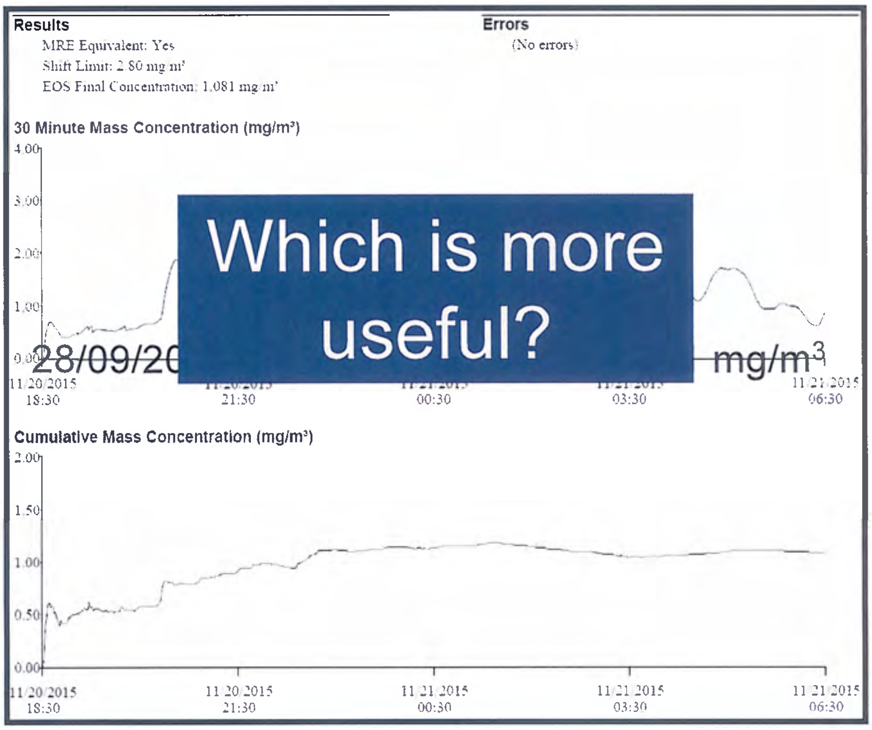



```
SHIFT LIMIT     2.00
PERCNT OF LIMIT  25%
```

```
30 MIN CONC    0.22
  CUM1 CONC    1.21
```

BENEFITS

ThermoScientific™ PDM3700



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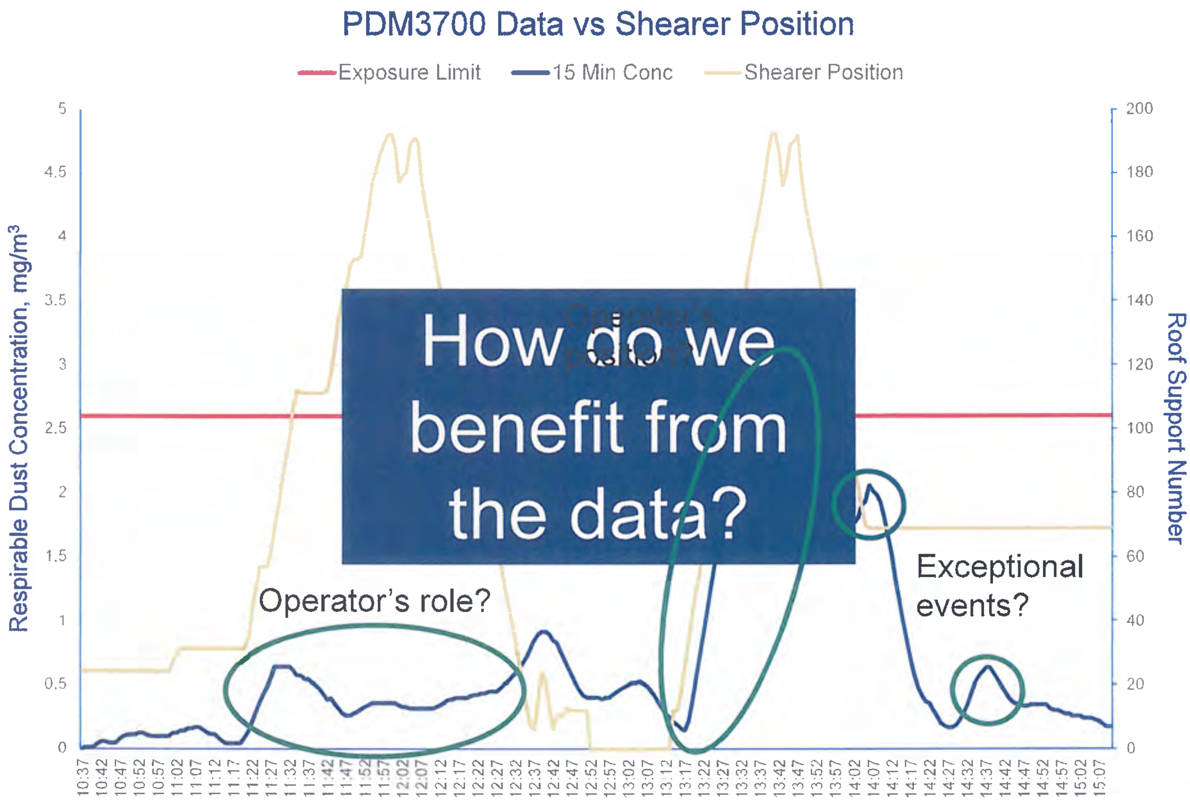

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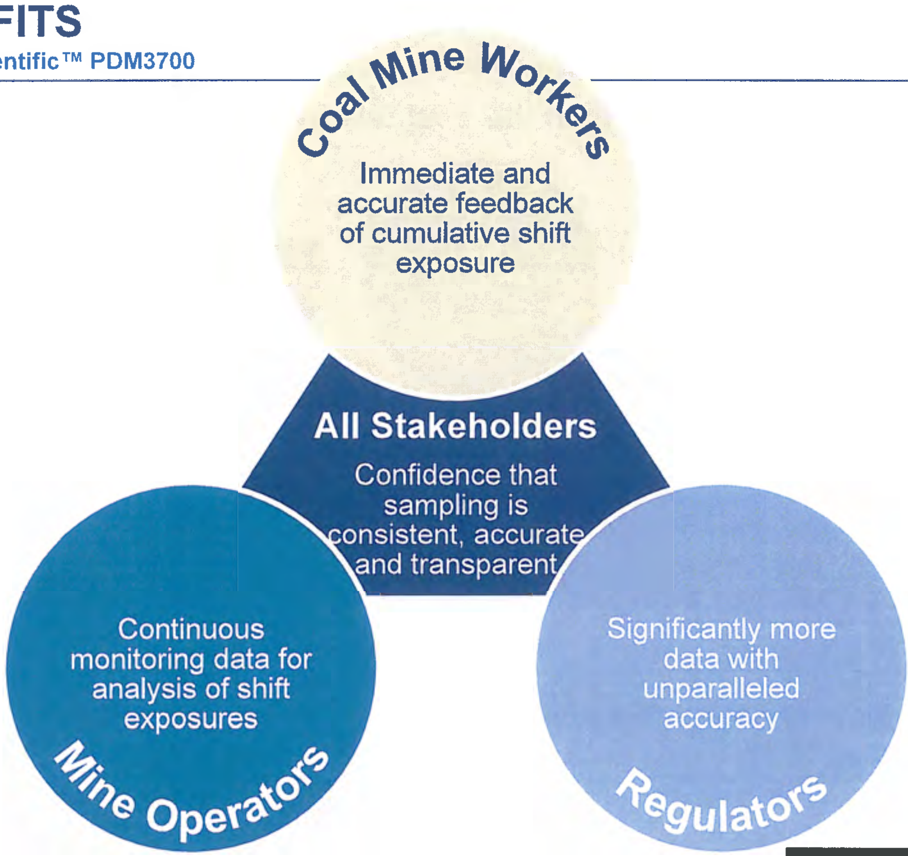
BENEFITS

ThermoScientific™ PDM3700



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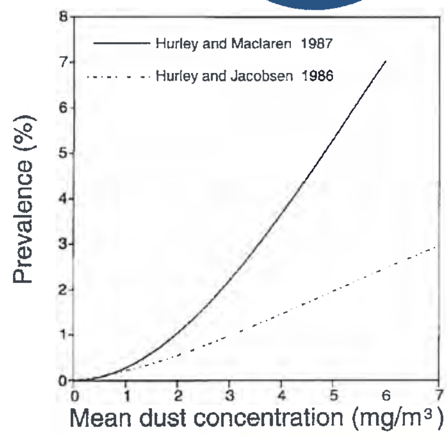
SAFETY, HEALTH AND COMPLIANCE

ThermoScientific™ PDM3700

- PDM3700 provides workers at the coal face with vital information

Information + **Action** = **Power**

- Why is this important?
  - A 1995 NIOSH report indicates that reducing exposure 0.5 mg/m^3 can significantly reduce the prevalence of CWP and PMF over a 35-year working lifetime.
  - **Every decimal point of exposure matters.**



SAFETY, HEALTH AND COMPLIANCE

ThermoScientific™ PDM3700

Electrical Certification

- Does not meet the requirements for certified portable electrical equipment under Recognised Standard 1
- Can only be used as in methane concentrations less than 0.5%
  - Excludes areas where workers are at the greatest risk of exposure to respirable dust

- Considerations for use in up to 1.25% methane:
  - MSHA and South Africa have certified these units for use in gassy underground mines
  - Time in service in the United States and South Africa provides a history of use in gassy underground mines.
  - Longwalls are typically the most heavily monitored area of a mine.
  - Typical methane concentrations on longwall faces range from 0.1% to 1.0%.

SAFETY, HEALTH AND COMPLIANCE

ThermoScientific™ PDM3700

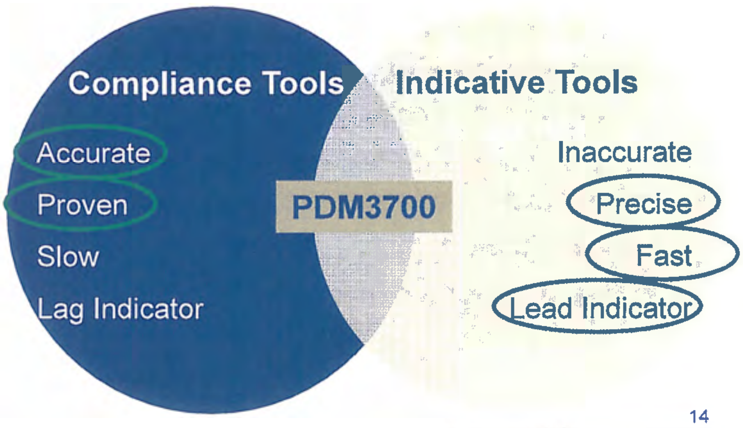
Limitations as Compliance Tool

- AS2985
- Proposed recognised standard

Criteria	Standard Gravimetric Sampling (AS2985)	PDM3700 (TEOM Technology)
Specified Size-Selective Sampling Cyclones	Higgins-Dewell, Simpeds, Aluminium	Uses Higgins-Dewell cyclone
Flow Rate	2.2 L/min (2.5 L/min for aluminium cyclones)	Controlled to 2.2 L/min
Flow Rate Accuracy	± 5%	± 2.5%
Filter Diameter	25 mm preferred, 37 mm permitted	15 mm
Timing Device	Required	Include in unit
Means of Mass Measurement	Five or six-place microbalance (10 μg or 1 μg divisions, respectively)	Tapered Element Oscillating Microbalance
Frequency of Mass Measurement	Filters weighed once before and once after sample taken	Continuous measurement via TEOM
Reporting	Final dust concentration reported as end of shift average	Dust concentration reported continuously throughout shift with end of shift average also reported

Case for use as Compliance Tool

- NIOSH proven accuracy, precision, and reliability
- Quick return on compliance samples
- Quickly build large data sets for SEGs
- Best of both technologies



SPEED HUMPS

ThermoScientific™ PDM3700

The following strategies will be pursued:

- **Seek changes** to Recognised Standard 01 to allow the PDM3700 to be used in atmospheres containing up to 1.25% methane until electrical certification is gained.

- **Assist** SIMTARS or another accredited body with expertise and manpower to expedite the certification of the PDM3700 and provide a clear timeline for the process.
  – Currently engaging MASC (South Africa) to assist us in this process

- **Ensure** that the proposed "Recognised Standard: Monitoring Respirable Dust in Coal Mines" does not preclude TEOM technology as an accepted means of gravimetric sampling.

- **Seek modification** to AS2985 to include provisions for the PDM3700 as an approved means of compliance sampling for respirable dust.

- **Actively support** SSEs should they elect to use the PDM3700 as a compliance sampling instrument.

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THANK YOU

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